COAL PREPARATION WASHING PROCESSES: A TECHNOLOGY REVIEW

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FOREWORD

Coal preparation plays a pivotal role in Canada's coal industry by providing a technically acceptable export product from a run-of-mine coal. Reliance on imported processing technologies and expertise, while having been essential to the growth of the modern Canadian coal industry, has done little to provide the incentive for development of a strong coal preparation tradition. Such a tradition, with continuity supplied by succeeding advancement of the technology in practice today, still remains very much in the realm of art.

The message of toughening market situations and increasing environmental concerns is one of potential economic difficulty for all coal producers. Maximum recovery of the resource, through optimization of washing processes and provision of reliable control methods, needs to be pursued through cooperative research and development programs involving the industry and informed professionals from various scientific and engineering disciplines.

The technology perspective presented in this report was obtained by thorough analysis of data measuring the performance of the washing processes used in Canadian plants. The report will have succeeded in its intent if, by raising a number of questions in the reader's mind, it can invite those concerned to meet the challenge of bridging the technology gaps responsible for many of the problems and headaches of coal preparation.

AVANT-PROPOS

La préparation du charbon joue un rôle essentiel dans l'industrie canadienne du charbon en fournissant un produit d'exportation techniquement acceptable à partir d'un charbon tout-venant. La dépendance de l'industrie envers les procédés de traitement et l'expertise venant de l'étranger, bien qu'ayant été essentielle à la croissance de l'industrie moderne du charbon du Canada, a peu aidé à encourager le développement d'une forte tradition de préparation du charbon. Une telle tradition, dont la continuité est assurée par les générations successives de travailleurs bien informés est fondamentale à l'avancement de la technologie utilisée aujourd'hui et demeure très bien dans le domaine de l'art.

L'indication de conditions difficiles du marché et d'une préoccupation accrue avec la protection de l'environnement est l'une des difficultés économiques potentielles pour tous les producteurs de charbon. La récupération maximale des ressources par l'optimalisation des procédés de lavage et par l'apport de méthodes de contrôle fiables, a besoin d'être poursuivie à l'aide de programmes de R-D coopératifs engageant l'industrie et les profesionnels renseignés venant de diverses disciplines scientifiques et l'ingénierie.

La perspective technologique présentée dans ce rapport a été obtenue après une analyse détaillée des données sur les mesures de performance des procédés de lavage utilisés dans les usines canadiennes. Ce rapport aura atteint son but si, après avoir soulevé un certain nombre de questions dans l'esprit du lecteur, il incite les intéressés à relever le défi afin de combler les lacunes techniques responsables des nombreux problèmes au nivau de la préparation du charbon.

COAL PREPARATION WASHING PROCESSES: A TECHNOLOGY REVIEW

By

J.L. Picard*

SUMMARY

Between 1978 and 1980, Canadian coal preparation plants produced between 18.6 and 22.5 x 10^6 tonnes of clean coal annually. Data based on samples taken at the ten operating washeries during the period indicated that the average industry-wide yield of 73.1% was achieved with an organic efficiency of 93.4% and that losses of saleable coal to the refuse amounting to 1.3-1.7 x 10^6 tonnes could be valued at close to $37.5 million per year.

Although a minor portion (<15%) of the losses might have been avoided, process performance was in most cases up to standard. The majority of losses can be attributed to deficiencies in various aspects of the technology, arising primarily from knowledge gaps relating to individual process mechanisms. Consequences of the deficiencies were particularly severe in the fine coal processes (concentrating table, hydrocyclone and froth flotation) which, while accounting for only 35% of all washed tonnage, accounted for 85% of all the saleable coal lost during preparation.

The plant data indicated that while the variables most usually cited as affecting performance of the individual processes generally held true, others were identified as apparently being needed to complete the picture. Production losses will be reduced only through a clear understanding of fundamental process mechanisms as a basis for possible equipment modifications and for design of appropriate operating and control systems. A concerted R & D effort along several fronts is needed if the technology is to advance.

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INTRODUCTION

In the interval between 1979 and 1983, value to the Canadian producer of metallurgical and thermal bituminous coal dispositions increased from $800 million to $1.1 billion. These returns resulted from overseas and domestic shipments of 18.6-22.5 million tonnes of clean coal obtained by processing 25-32 million tonnes run-of-mine coal (1). The dollar value represents the fruits of a sizeable long-term financial investment and the contribution of considerable human resources which, together with state-of-the-art mining techniques and washery processing facilities, have enabled production of a suitable product in a highly competitive international market environment.

The capricious nature of Canada's geography places its coal industry at an economic disadvantage in competing for distant markets because of high transportation costs. The producer therefore understands only too well that the viability of operations rests on his ability to output a consistently high-quality product at highest recovery and lowest cost possible. It is easily calculated that for each 1% improvement in recovery that could be achieved without loss of quality, the industry as a whole would stand to gain $11-13 million per year at current prices and production rates. It is also recognized, however, that even such a small improvement is not always easy to attain. The washery operator is usually pressed to the limit as it is to maintain production in the face of changeable quality of mine product, mechanical equipment breakdown and shortages of trained personnel. Even with these factors minimized and with processes operating at maximum effectiveness, there remains the reality that, despite all efforts, significant quantities of clean coal frequently end up not in the unit trains but in the refuse piles and tailings ponds.

The principal objective of the present report is a practical perspective on the various problems associated with application of the conventional washing processes and through this, identification of the technology gaps that contribute significantly to the washing losses and costs of cleaning for the Canadian coal preparation industry at large. The report presents an overview of individual process capabilities and limitations based on the established body of knowledge and on results of an analysis of washery data obtained from a sampling program at the ten Canadian washeries that were operating in 1978-80 (2). The report was written primarily to address those who may be familiar with coal preparation processes and practices, the terminology and conventions. Where possible, however, an attempt has been made to simplify discussion so that the information might be meaningful to other interested readers.

Following a short review of statistics relating to process applications in the Canadian coal preparation industry and an explanation of the methodology for presentation and analysis of the washery data, each of the processes will be dealt with on an individual basis. Each will begin with brief historical and technical comments followed, in order, by a general summary of process characteristics as might be presented in textbooks, presentation of the plant data, feed characteristics and, finally, the washing results with discussion of the effects of feed or other variables on performance and separation losses. A general summary with discussion and conclusions completes the report. Details of the results for each process are provided in an appendix.

PROCESS APPLICATIONS

Data relating to usage of the six processes in the washeries sampled are given in Table 1. It is noted for information purposes only that three of the washeries represented in the statistics have since ceased operation. Beginning in 1980 and up to the present, one plant expansion has taken place and seven new plants have been commissioned. These additions have provided a net increase of 3465 tph in available Canadian washery capacity to a current (1984) total of approximately 8700 tph and, it seems, with an even heavier reliance than before on the heavy medium, hydrocyclone and froth flotation processes (1).

For the period of interest, Table 1 shows that the most widely used process was the heavy medium cyclone. In six installations it accounted for between 44 and 71% of the run-of-mine feed and for 43% of all the raw coal washed. By comparison, the hydrocyclone, wet concentrating table and froth flotation used for minus 3 mm fines in the various plants, together accounted for 35% and the coarse coal washers used in the 127-10 mm nominal size range, for only 20% of washed tonnage.

It is evident from Fig. 1 that in recent years, the Canadian coal preparation industry has placed far more reliance on the small and fine coal processes and far less on those for coarse coal than has been the case on the average in other major coal-producing countries. The difference in process emphasis can be taken as reflecting the fact that the greatest proportion of washing applications in Canada is for friable western mountain coals which generally degrade readily upon handling.

Table 1 – Coal preparation processes in Canada during 1978-80

Process	No. of plants	Nominal size range (mm)	% of Washery feed			Installed capacity		Annual throughput (est)	
			min	max	mean	tph	%	10^6 tonnes	%
Coarse coal jig	4	127-10	13	100	61	631	10.3	2.06	7.0
Heavy-medium vessel	3	127-10	18	52	30	736	12.1	3.64	12.5
Heavy-medium cyclone	6	38-0.6	44	71	58	2436	39.8	12.59	43.1
Hydrocyclone	3	25-0.6	21	50	31	320	5.2	0.64	2.2
	5	<0.6	13	36	27	1082	17.7	5.55	19.0
Concentrating table	1	3-0.1	—	—	41	145	2.4	0.54	1.8
Froth flotation	5	<0.6	12	30	22	765	12.5	4.20	14.4
Total	—	—	—	—	—	6115 *	100.0	29.22*	100.0

* *Front-end plant capacity and throughput totalled 5235 tph and 25.25 × 10^6 tpa, respectively*

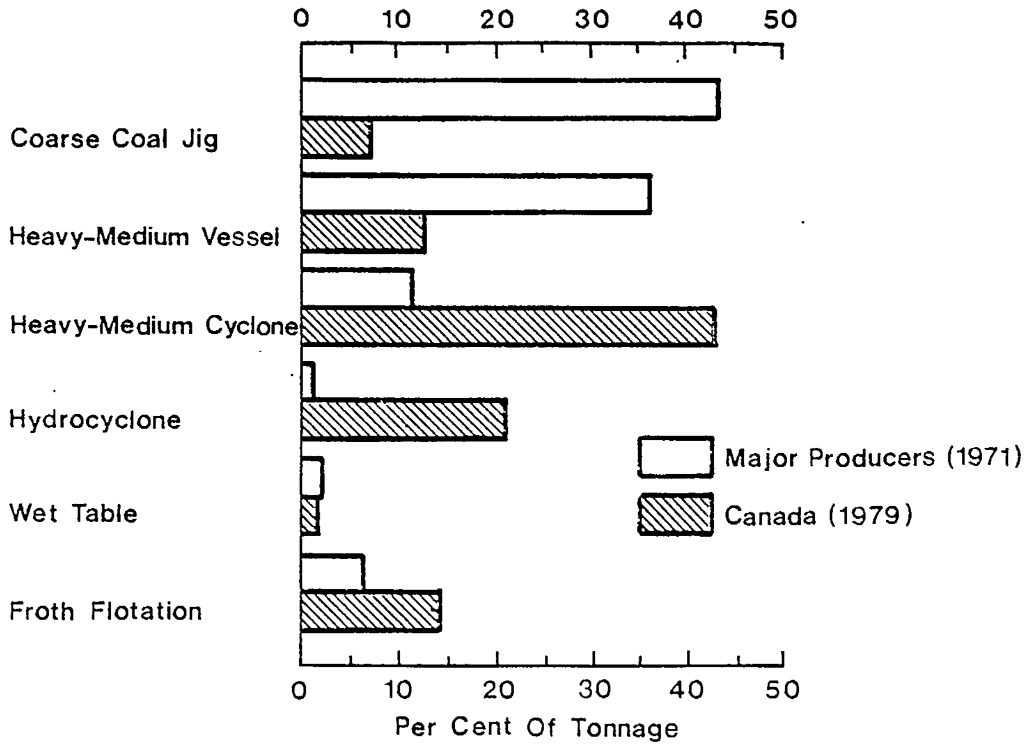

Fig.1 – Comparison of process use in Canada and other major coal-producing countries

METHODOLOGY

For simplicity in presentation and discussion of plant feed characteristics, the familiar washability curves have been replaced by plots which show only the variation in distribution of three of the density fractions with variation in particle size. These fractions are the floats at 1.4, the 1.4-1.8 and the sinks at 1.8 relative density, arbitrarily chosen to represent the coal, middlings and refuse fractions respectively in the feed.

For evaluation of coal-washing performance, the distinction is made between separation sharpness and separation efficiency. Separation sharpness or accuracy may be defined by the so-called independent criteria such as probable error, imperfection and other parameters of the partition curve. Separation efficiency, on the other hand, is a function of both separation sharpness and feed washability and is usually expressed by dependent criteria such as the yield error. Definitions of these criteria may be found in the appended Glossary. Additional discussion is available in the literature (3). Assessment of the results in terms of separation sharpness and efficiency was facilitated through the use of correlation and multiple regression analyses. For this purpose it was assumed that the separators were in good mechanical condition, properly adjusted and operating at top capability. Any differences in performance between plants were assumed, therefore, to have resulted primarily from influences related to feed characteristics or, to the extent that the data permitted in the case of cyclones, possibly also because of differences in design geometry.

For reasons of brevity, discussion of process performance has been largely limited to consideration of the probable error and of the yield error with its correlative organic efficiency. To these has been added a measure termed "product loss" which can be defined as the percentage floats coal of the same ash content as the clean coal that occurs in the refuse. This measure was determined from the float-sink analysis of the refuse and although it is similar, it is not equivalent to the criterion "floats in refuse".

COARSE COAL JIG

Use of the jig in coal preparation dates back to late 19th-century Europe. Although it has generally suffered a slight drop in popularity in recent decades, the jig still accounts for the greatest proportion of tonnage washed in the world today. Most widespread use is found in Europe where it continues in its traditional role as the work-horse of coal preparation. This is in sharp contrast to the dramatic decline in use of the jig in Canada that occurred with the industry downturn in the late 1950's. Even with the resurgence of the industry that has since taken place, the jig has not yet regained the historical primacy it enjoyed in the past. In 1980 the jig accounted for approximately 7% of all coal washed in Canada (Table 1).

Separation in the jig is accomplished through the use of a pulsating upward and downward water flow which causes stratification of particles into horizontal layers of increasing density from the top to the bottom of the bed. The jig is particularly favoured for steam coals because of its ability to treat a wide range of sizes at low cost; power consumption is generally quoted at 0.8 kW/m^2 (0.1 HP/ft^2) for throughput rates of 29-59 tph/m^2 (3-6 tph/ft^2) of screen area. In some cases, jig efficiency compares well with that of heavy medium vessels. Generally speaking, however, it is less effective for difficult separations at low cutpoints and its use is not indicated for highly variable feeds or for coals with high refuse or fines contents. Despite its basic simplicity of operation, the jig is more difficult to adjust properly and to control than most washers and requires the attention of a skilled operator.

PROCESS CHARACTERISTICS

Efficient separation in the jig is the outcome of proper stratification. This requires full mobility or expansion of the bed during one-half of each pulse cycle followed by a stage of bed contraction, then one of quiescence during which fine particles stratify through a process of consolidation trickling. Stratification is usually complete in 30-40 seconds and normally occurs at a rate that is proportional to the pulse rate, particle size and refuse content and that is inversely proportional to the pulse amplitude. Pulse rate is the single most important operating variable and is usually in the range of 30-60 pulses per minute. As a general rule, frequency is increased as particle density increases and/or as particle size decreases and is decreased as feed rate and bed depth increase and/or as particle size increases. However, this adjustment must also take into consideration the pulse amplitude, water volume and height of the refuse gate.

Jigs show a normal tendency towards losses of very fine coal in the refuse and increasing losses of refuse in clean product as particle size decreases. Significant impairment in performance can result from overfeeding or from high refuse or high fines contents in the feed. Over the years, the accepted view has been that jig operation is generally typified by an increase in cutpoint with decreasing particle size and by decrease in separation sharpness as the cutpoint rises. These tendencies can be observed in Fig. 2 which is based on data taken from the literature (4,5,6). These data show that, on the average, as particle size decreased in the range of 90 to 0.3 mm:

- cutpoint (dp) increased from 1.4 to 1.8 relative density;
- probable error increased from 0.050 to 0.155;
- the imperfection increased from 0.135 to 0.165;
- the error area increased from 50 to 100.

PLANT DATA

During the 1978-80 period, four jig plants with a total installed capacity of 630 tph were in operation in Canada. Two of the four installations were for metallurgical coal. Table 2 shows that two of the jigs were single-compartment Vissac types and two were double-compartment Baum types. Few of the operating details were available and one of the jigs (Plant D) could not be sampled because of access problems.

Plants A, B and C treated pre-screened feeds in the nominal size range 127-10 mm and produced finished clean coal and refuse. Unlike these three, the jig in Plant D was used as a primary washer treating unsized 50 mm × 0 run-of-mine, yielding finished plus 19 mm products; further washing for the minus 19 mm sizes was provided.

There were considerable differences in feed characteristics between the three jig plants sampled (A, B and C). As shown in Table 3, ash contents varied between 18 and 79% and mean particle size between 22 and 62 mm. Feed A was the finest and lowest in ash content while feed C was the coarsest and highest in ash content (Table 3, Fig. 3). Figure 4 shows that the refuse content of all feeds tended to decrease as particle size became finer.

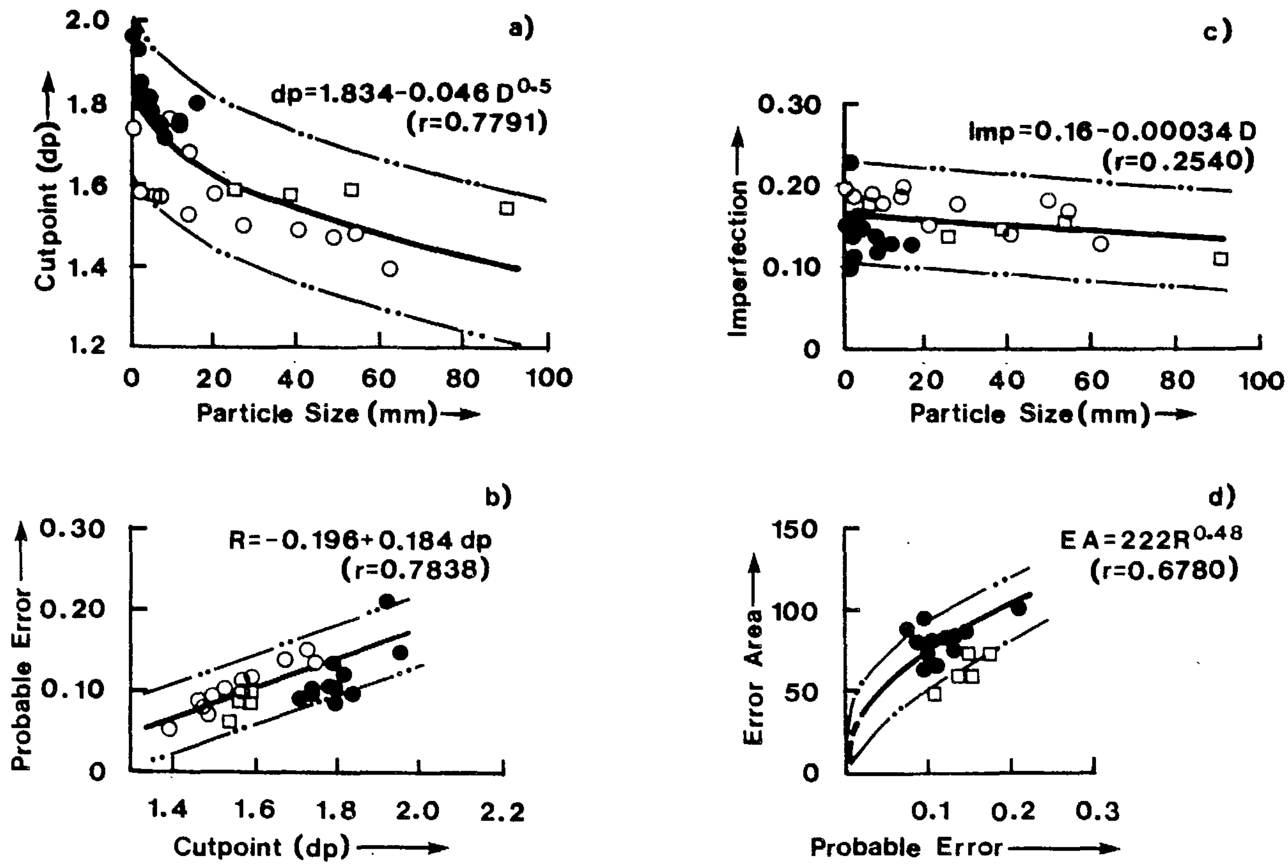

Fig. 2 – Jig separation characteristics

Table 2 – Operating data for jig plants (1978-80)

	Plant			
	A	B	C	D
Unit feed rate (tph)	227	28	27	350
Pulse rate (m^{-1})	23-28	—	—	—
Average yield (%)	85	80	17	90
Jig type	Baum	Vissac	Vissac	Baum

Table 3 – Feed characteristics for jig coal-washing plants in Canada (1978-80)

	Raw feed			Density distribution				Theoretical*		
Plant	Nominal size range (mm)	Undersize (%)	Ash (%)	Floats @ 1.4 (%)	1.4-1.8 (%)	Sinks @ 1.8 (%)	Product ash (%)	Cutpoint (d_p)	Product yield (%)	Reject ash (%)
A	50-10 ($\bar{D}$ = 22 mm)	30	18	55	32	13	13	1.74	85	53
B	100-10 ($\bar{D}$ = 36 mm)	18	47	33	9	58	8	1.60	39	78
C	127-25 ($\bar{D}$ = 62 mm)	26	79**	7	5	88	25	>2.2	17	85

* *Reconstituted feed washability data*
** *Plus 25 mm size fraction*

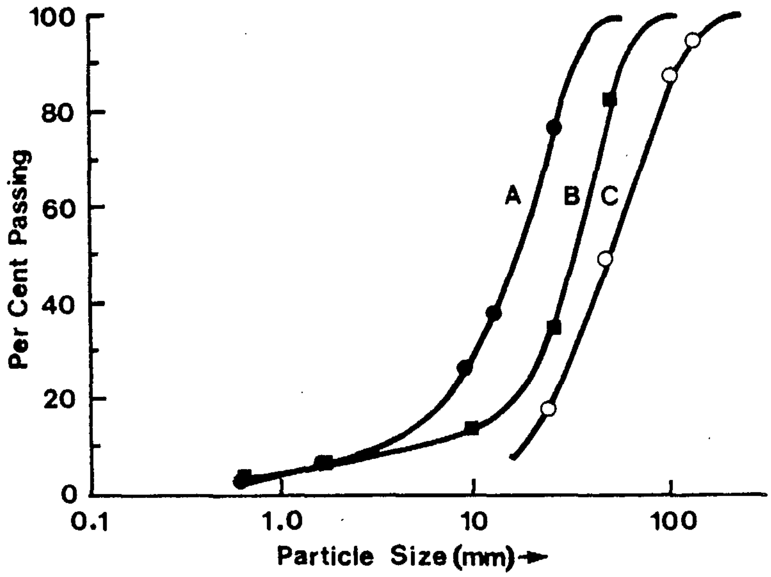

Fig. 3 – Size distribution of jig feeds (reconstituted)

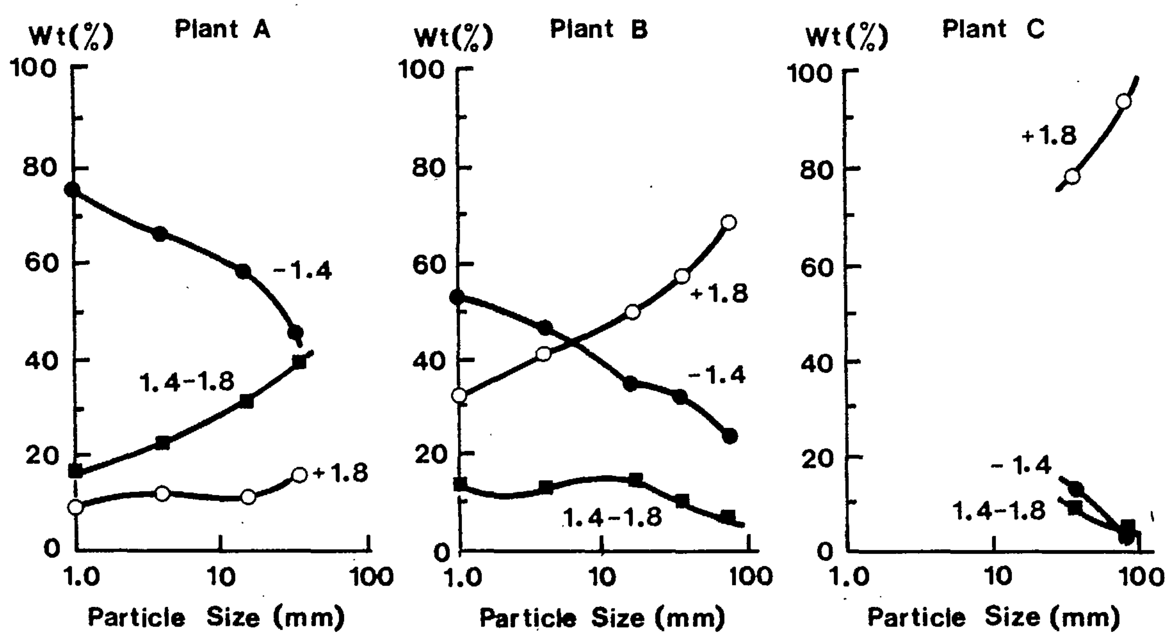

Fig. 4 – Variation of density distribution of jig feeds (reconstituted) with particle size

JIG PERFORMANCE

Overall washing results show that at cutpoints between 1.51 and 1.73, the jigs produced clean coal containing 8-25% ash at yields of 12 to 84% with organic efficiencies between 98.6% and 68.8% (Table 4).

From the cutpoints obtained and mean feed particle size, reference to the relationships in Fig. 2 indicates that separation sharpness as measured by the probable error, imperfection and error area could be judged as having been better than average for Plant A, slightly poorer than average for Plant B and much poorer than average for Plant C. These results evidently confirm that jigs are not ideally suited to the cleaning of high refuse coals and that separation sharpness deteriorates as feed refuse content increases. Decrease in separation sharpness was accompanied by a significant increase in efficiency loss as shown by the drop in organic efficiency from 98.6% for Plant A to 68.8% for Plant C (Table 4). As indicated by an increase in the sinks in clean coal from 2.5 to 17.4%, this drop in efficiency appears to have been associated with a decrease in ability to remove refuse particles as their proportions in the feed increase.

Table 4 – Overall performance of jig plants (composite feeds)

	Plant		
	A 50-10	B 100-10	C 127-25
Ash content (%)			
Raw coal	19.3	52.9	78.8
Reconstituted feed	18.9	50.3	74.9
Clean coal	12.8	7.8	24.6
Refuse	50.5	72.3	81.5
Yield of clean coal (%)	83.8	34.1	11.7
Theoretical yield (%)	85.0	39.4	17.0
Organic efficiency (%)	98.6	86.6	68.8
Separation density (d_p)	1.727	1.510	1.692
Probable error	0.105	0.116	0.240
Error area	61	74	134
Imperfection	0.144	0.228	0.347
±0.10 near density material (%)	8.8	6.1	2.9
Floats in refuse (%)	14.4	7.5	2.6
Sinks in clean coal (%)	2.5	4.6	17.4
Total misplaced material (%)	4.4	6.5	4.3
Yield error (%)	1.0	5.3	5.3
Ash error (%)	0.3	2.0	9.4

The partition curves in Fig. 5 for the individual size fractions show that whereas both jigs A and B suffered normal loss of separation sharpness as particle size decreased, jig B had a more pronounced tendency towards refuse carryover. Jig C had an abnormally high carryover in the coarser fraction and excessive losses of both coal and refuse in the 50-25 mm fraction. Such behaviour for coarse sizes is most often caused by poor bed mobility. Reasonably good coal recovery combined with very poor reject elimination, as shown for the 127-50 mm fraction of Plant C, is typical of poor bed mobility arising from an excessively high pulse rate whereas general loss of separation effectiveness, as shown for the 50-25 mm fraction, would be more typical of poor mobility because of overfeeding (7). Whichever may have been the case in this jig, it is clear that a condition of high bed density because of high feed refuse content would make it exceedingly difficult to achieve good stratification.

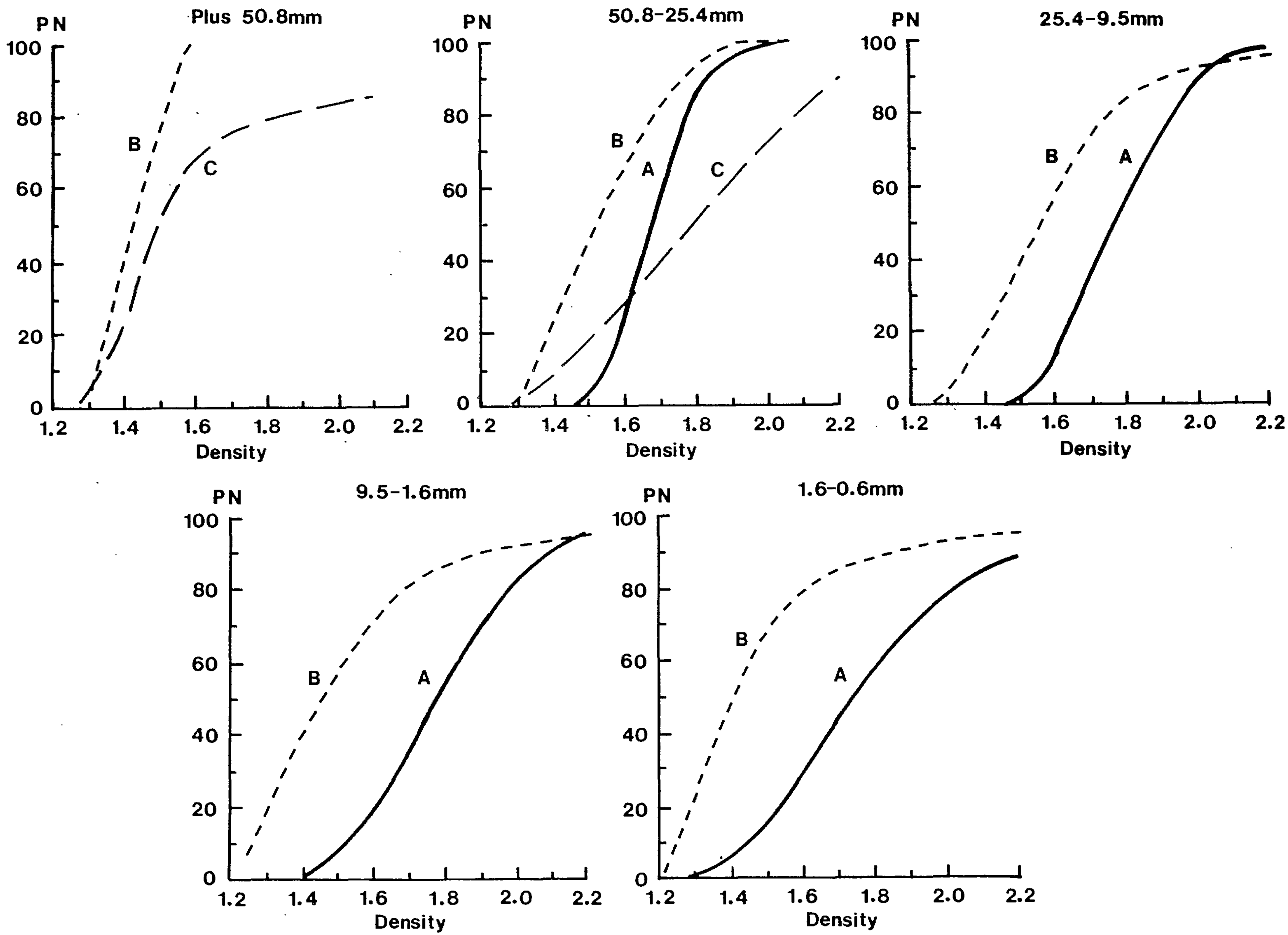

Fig. 5 – Partition curves for jig washing of individual size fractions

Effect of Feed Characteristics

The plant data conformed to the general trends of increasing probable error with increasing cutpoint and increasing imperfection with decreasing particle size shown in Fig. 2. Contrary to expectations, however, the cutpoints in Plants A and B did not show a steady rise as particle size decreased (Fig. 6a). A cursory survey of the literature showed that such a cutpoint rise is not invariably the case (4,6,7). Although little is known of the factors that influence the relationship, it has been demonstrated experimentally that, a marked rise in cutpoint as particle size decreases could be caused by a poor suction stroke (7). This point could be raised for Plant C where the cutpoint rose quickly from 1.5 to 1.8 over a relatively narrow range of decrease in particle size (Fig. 6a).

Multiple regression analysis of the data disclosed that for the coals treated, jig performance could be described in terms of the percent refuse content (S18) in the feed, the cutpoint (dp) and the particle size (D). For example, the equation for probable error ($r = 0.9609$ for $n = 16$),

$$R = -0.137 + 0.00003(S18)^2 - 0.017D^{0.5} + 0.1807dp - 0.00001D^2 \qquad \text{Eq 1}$$

gives reasonably good estimates of the observed values for all 3 Plants (Fig. 6b). The relationship confirms that separation sharpness in jigs is best in the low cutpoint range, and for coarse particles and low refuse feeds. The effects of refuse content and particle size on R as given by equation 1 are illustrated in Fig. 7 for a cutpoint, dp = 1.6.

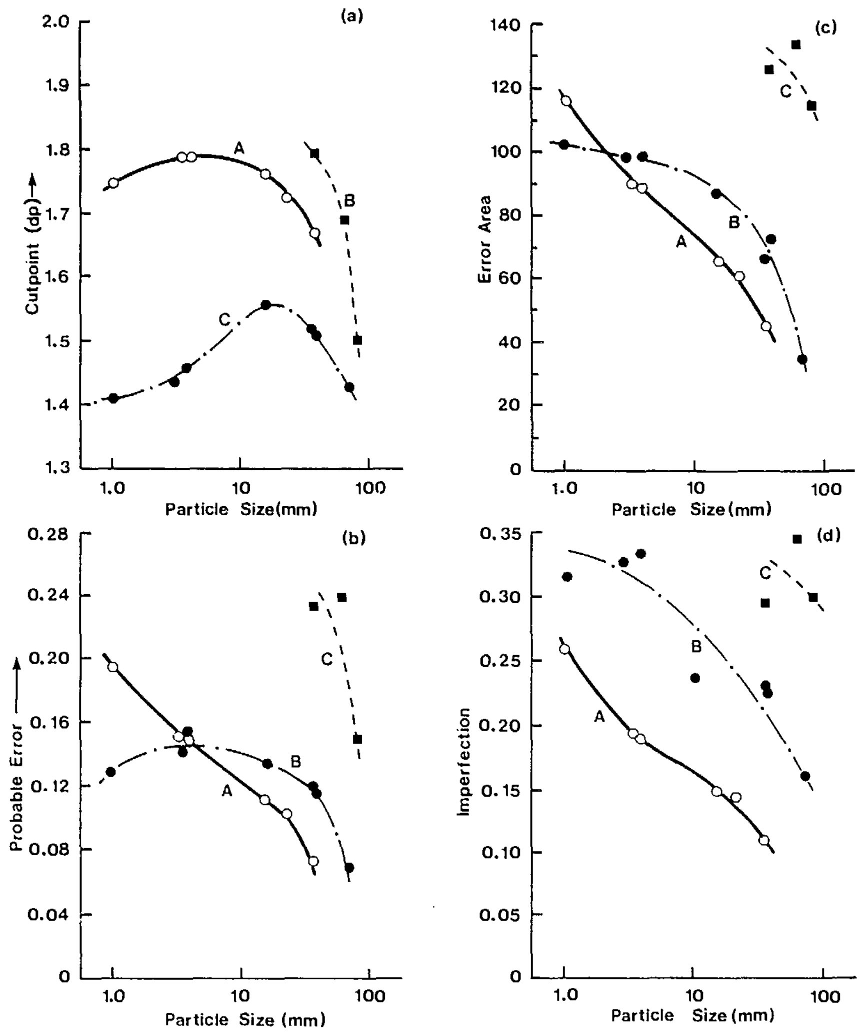

Fig. 6 – Variation of jig separation characteristics with particle size

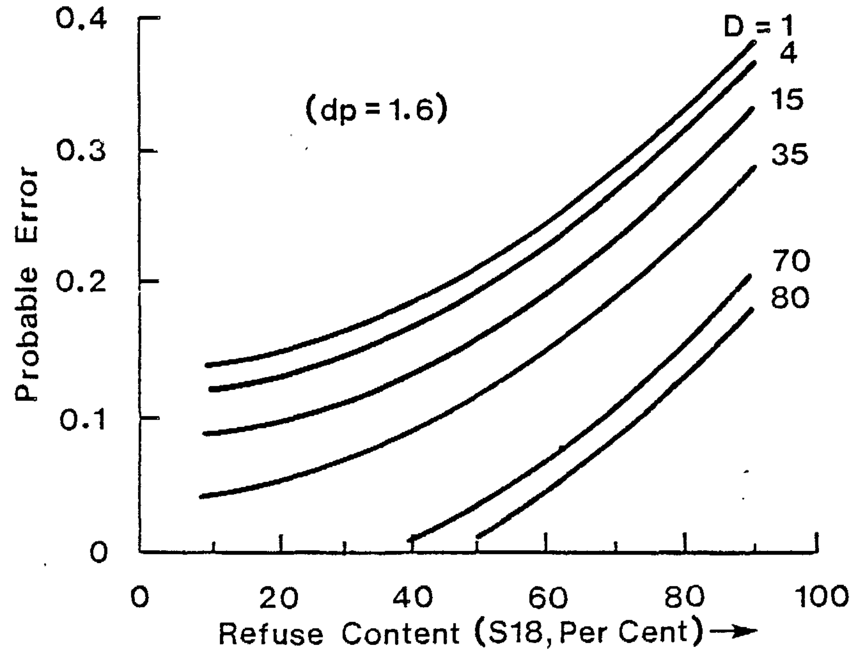

Fig.7 – Effect of refuse content and particle size on probable error for jigs A, B and C (d_p = 1.6)

Separation Losses

For the full ranges of individual size fractions, recovery efficiencies were best in Plant A (98.9-96.3%), intermediate in Plant B (89.2-63.8%) and poorest in Plant C (68.8-65.1%). The absolute recovery losses (yield error) corresponding to these efficiencies ranged between 0.9 and 25.1% for all 3 plants. Yield error increased in direct proportion to the percentage ±0.1 near-density material (NG) in the feed and to jig separation sharpness as measured by the imperfection (Fig. 8a). The loss of product quality coal in refuse (0-41.7%) showed some relationship with yield error and was similarly found to increase in direct proportion to increase in near-density material and imperfection but, additionally, in proportion to increase in the percentage of low-density coal (floats at 1.35) in the feed. This additional factor results in only a rough proportionality between product loss and yield error (Fig. 8b).

The estimated product losses in Table 5 show that as percentages of the feed, the lowest losses were in Plant A (0.9%) the highest in Plant B (4.9%), with Plant C falling in between with an estimated 2.1% of the feed. On the basis of the feed rates in Table 2 for each of the jigs, it is estimated that clean coal was lost at the rate of 2.1 tph for Plant A, 1.3 tph for Plant B and 0.6 tph for Plant C. The combined loss rate of 4.00 tph amounted to an average of 1.42% on the raw coal basis (1.95% clean coal basis) with an average clean coal content in refuse of 5.27%.

Table 5 – Estimated clean coal losses in jig refuse: plants A, B and C (1978-80)

	Size fraction (mm)					Total
	+50	50-25	25-10	10-1.7	1.7-0.6	
Plant A						
Product ash (%)	—	13.6	12.5	9.6	6.7	11.9
Wt in refuse	—	0	12.0	4.5	16.1	6.18
% of total feed	—	0	0.76	0.11	0.07	0.94
Plant B						
Product ash (%)	6.9	7.4	9.0	7.2	7.9	7.8
Wt in refuse (%)	4.7	5.2	8.2	24.2	41.7	7.89
% of total feed	0.62	1.62	1.03	0.95	0.72	4.94
Plant C						
Product ash (%)	20.2	25.8	—	—	—	24.6
Wt in refuse (%)	1.8	4.8	—	—	—	2.87
% of total feed	0.83	1.23	—	—	—	2.06

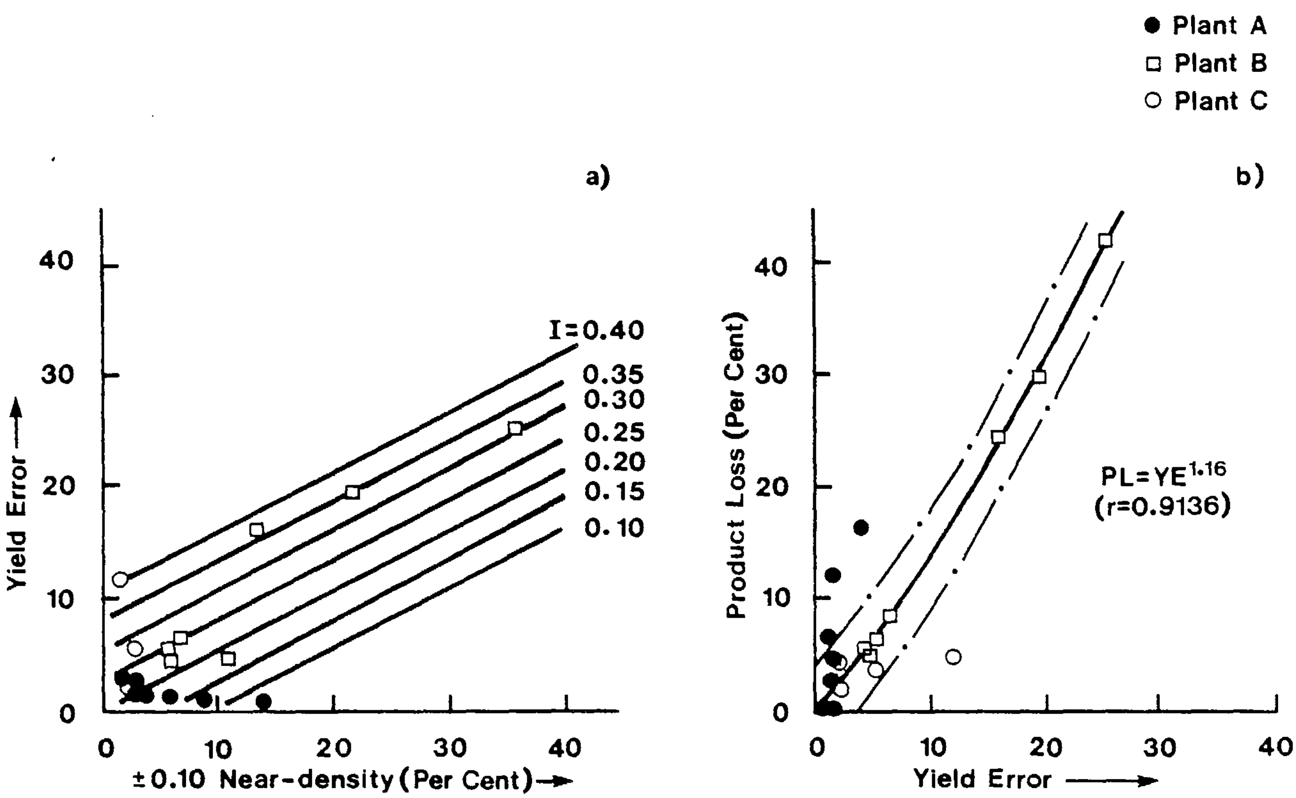

Fig. 8 – Recovery and product losses for jig Plants A, B and C

SUMMARY

1. The jigs treated feeds containing between 18 and 79% ash-producing clean coals containing 8-25% ash at cutpoints in the range 1.51-1.73 relative density and with yields of 12-84% at 98.6-68.8% organic efficiency.
2. Overall separation sharpness deteriorated as refuse content in the feed increased and was slightly better than average for Plant A (R = 0.105), slightly below average for Plant B (R = 0.116) and much below average for Plant C (R = 0.240).
3. Partition curves indicated impaired separation because of poor bed mobility in Plant C and confirmed that the high feed refuse content (90%) was beyond the range usually considered appropriate for jigs.
4. Variations in probable error and related separation criteria were largely explained by variations in cutpoint, particle size and percent sinks at 1.8 relative density.
5. Cutpoint in the jig does not always rise steadily as particle size decreases.
6. Recovery losses as expressed by the yield error for the individual size fractions were 0.9-3.6% for Plant A, 4.3-25.1% for Plant B and 2.3-11.9% for Plant C and were greatest at low separation sharpness and for high ±0.1 near-density material in the feed.
7. Estimated product losses amounted to 0.9, 4.9 and 2.1% of the feed, corresponding to 2.1, 1.3 and 0.6 tph for Plants A, B and C respectively.

HEAVY-MEDIUM VESSEL

Since it was first used for coal washing in England in 1858, use of heavy-medium separation for coarse coal has become widespread throughout the coal-producing world. The past two decades have shown only a slight growth in popularity so that the percentage of washed tonnage handled has remained relatively stable at 25-30% worldwide. Over the years, the most significant process developments have involved a number of variations in vessel design and a gradual switchover to magnetite medium. Since its first commercial use in about 1938, magnetite has become preferred over such materials as loess, crushed refuse, barytes, pyrites and sand because of its high relative density and magnetic properties which favour low suspension viscosity and ease of reclamation. In Canada, where magnetite exclusively is used, heavy-medium vessel separation accounts for more than 78% of the coarse and approximately 14% of all the washed coal.

Separation in the heavy-medium vessel results from the fact that differences between the buoyant and gravitational forces that act on the coal particles cause those that are less dense than the medium to float and those that are more dense to sink. The process is generally used where washery feed contains a significant proportion of 150-6 mm particle sizes. It is favoured where sharp separation or closely controlled cutpoint is required, thus for metallurgical coals or those containing high percentages of middlings or near-density material. In addition to general effectiveness, the process has the advantages of high capacity, low cost, applicability to a wide range of sizes and cutpoints, ease of control and ability to handle variations in feed. Top operation requires pre-wetting, feed size control by crushing and screening and maintenance of medium quality. Magnetite consumption is normally between 0.25 and 0.50 kg/tonne of feed.

PROCESS CHARACTERISTICS

Generally speaking, vessel capacity and performance are governed by the settling rate of the coal in the medium. The settling rate varies according to size of the particle, its density relative to that of the medium and viscosity of the medium, and is sensitive to vertical currents that may be present in the vessel. Efficient separation is best ensured by using a suitable medium and controlling its quality. In the case of magnetite, the size distribution must be appropriate to the intended cutpoint range, it should be non-reactive, resistant to sliming, of high relative density and should possess properties such as high magnetic susceptibility and low coercive force where needed for reclamation purposes (8,9).

Vessel performance is typified by high separation sharpness that is unaffected by the ± 0.1 near-density material and by a tendency towards increasing refuse carryover as particle size decreases. Impaired performance most often results from overfeeding, high percentages of undersize in the feed, unwetted coal and from poor stability or high viscosity of the medium. It is generally held that separation sharpness is primarily a function of particle size and, as shown in Fig. 9, that it deteriorates relatively slowly as particle size decreases (10). Deterioration accompanying rise in cutpoint is possible but is generally slight except where medium viscosity is high. Under this condition, which can result from contamination of the medium by fine coal or shale, significant deterioration in separation sharpness would be observed.

PLANT DATA

During 1978-80, three heavy-medium vessel plants with a total installed capacity of 735 tph were in operation. A single installation (Plant A) accounted for more than 50% of this capacity (Table 6). One plant employed a 3-product drum separator while the others used 2-product open baths. The pre-screened feeds were all within the nominal size range 127-10 mm and constituted between 18 and 52%, averaging 30%, of the total run-of-mine washery feed. The vessels produced finished clean coal and refuse. The middlings fraction from the 3-product separator was crushed and rewashed in the plant. On the average, medium density was controlled at between 1.40 and 1.55. Consumption was in the normal range at between 0.4-0.5 kg/tonne.

Ash contents of the vessel feeds varied between a low of 27% for Plant A and a high of 72% for Plant B (Table 7). The percentages of undersize varied within narrow limits (10-17%), consistent with the remarkably uniform feed size distributions (Fig. 10). Although the data in Table 7 might suggest that the feeds to Plants A and C were very much alike, the density distributions show that this was far from being the case. Washability characteristics of all three coals in fact showed few similarities (Fig. 11).

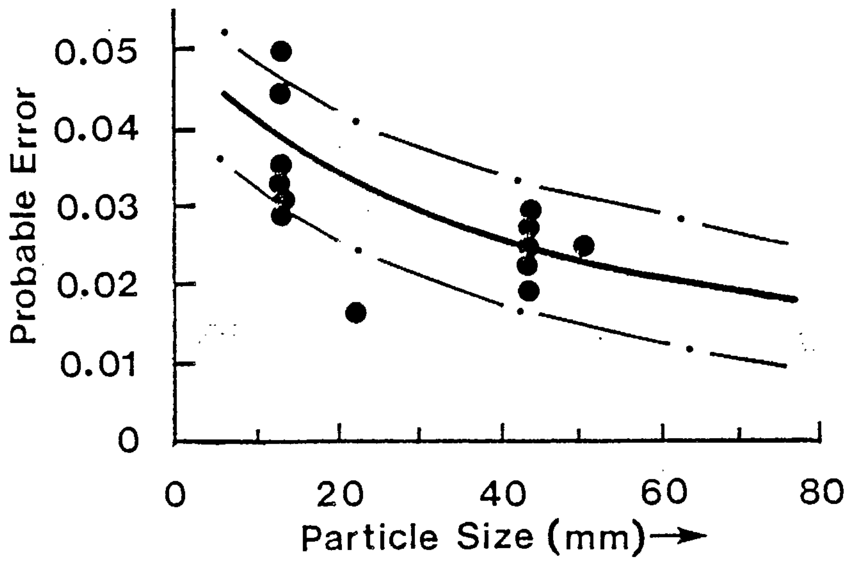

Fig. 9 – Effect of particle size on probable error of separation for heavy-medium vessels

Table 6 – Operating data for heavy-medium vessel plants (1978-80)

	Plant		
	A	B	C
Unit feed rate (tph)	372	68	113
Size range (mm)	100-12.7	127-10.0	100-10.0
Solids concentration (Wt%)	90	95	95
Medium density	1.45-1.55	1.45-1.50	1.40-1.45
Average yield (%)	65	—	64
Magnetite:			
% minus 44 μm	92	90	90
Consumption (kg/t)	0.4	0.5	0.4

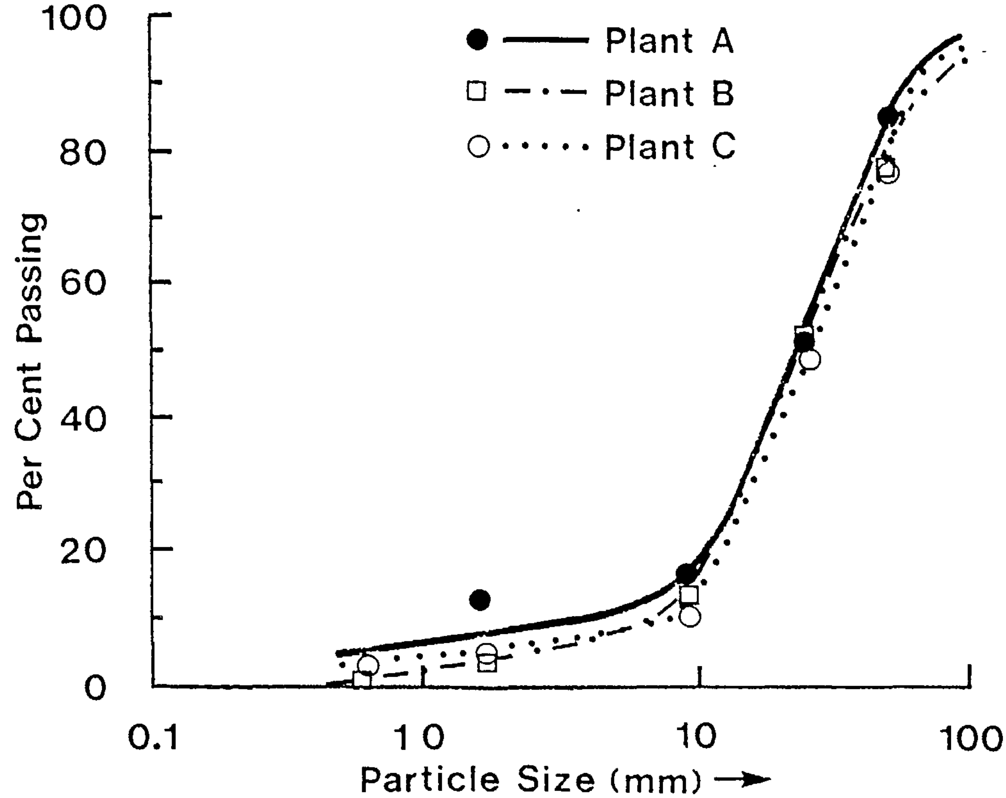

Fig. 10 – Size distribution of heavy-medium vessel feeds (reconstituted)

Table 7 – Feed characteristics for heavy-medium vessel plants

Plant	Raw feed			Density distribution			Theoretical			
	Nominal size range (mm)	Undersize (%)	Ash (%)	Floats @ 1.4 (%)	1.4-1.8 (%)	Sinks @ 1.8 (%)	Product ash (%)	Cutpoint (dp)	Product yield (%)	Reject ash (%)
A	100-127	13	27	42	36	22	12	1.56	68	60
B	127-10	17	72	13	13	74	8	1.40	13	77
C	100-10	10	30	47	31	22	13	1.50	66	59

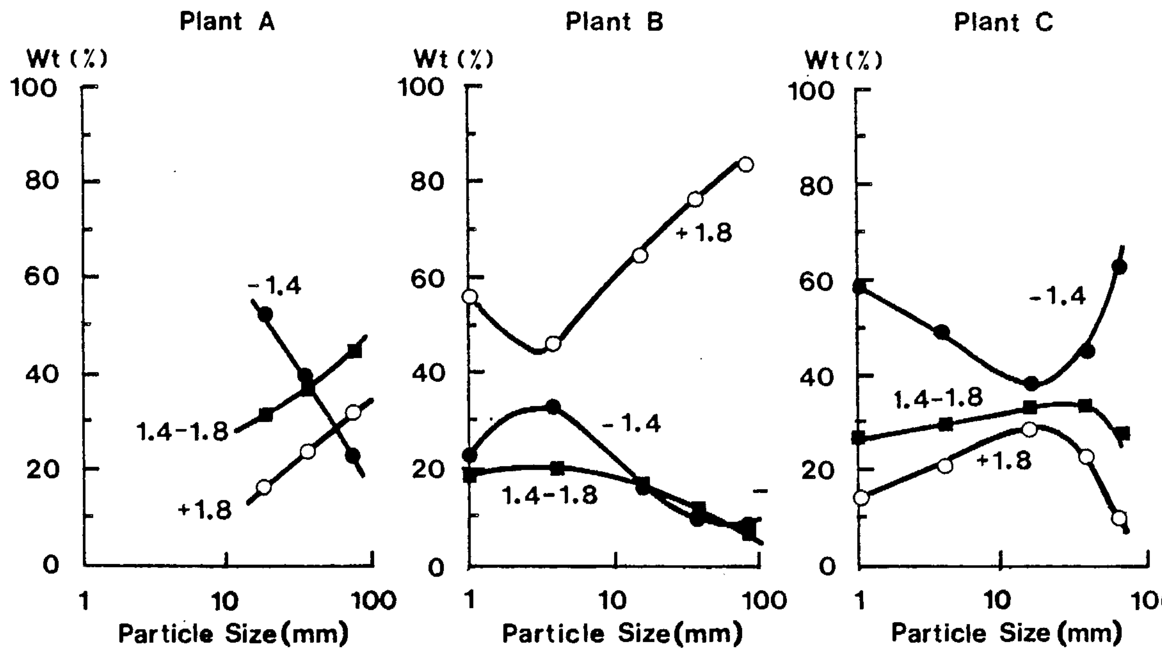

Fig. 11 – Variation of density distribution of heavy-medium vessel feeds with particle size (reconstituted)

HEAVY-MEDIUM VESSEL PERFORMANCE

Overall washing results for the three vessels show that at cutpoints between 1.37 and 1.55, clean coal products containing 8-13% ash were obtained at yields of 11 to 67% with organic efficiences between 81 and 98% (Table 8). The probable errors and error areas show that the sharpest separations were achieved in Plants A and B. On the basis of the range of probable errors (0.020-0.035) shown in Fig. 9 for overall mean particle sizes of 32-36 mm, vessel operation in Plants A and B can be considered as having been normal while that in Plant C was somewhat below par. The apparent contradiction presented by the organic efficiences (Plant A = 98.4%, Plant B = 80.9% and Plant C = 90.2%) arises primarily because of the low theoretical yield for Plant B. A true comparison of plant performance is best achieved in this case through the floats in refuse which showed a high correlation with the probable error.

Partition curves for the individual size fractions show that, except for the characteristic tendency of vessels towards increasing refuse carryover with decreasing particle size, Plants A and B, unlike Plant C, suffered little loss in separation sharpness in the size fractions above 10 mm (Fig. 12). For size fractions below 10 mm, Plants B and C both showed noticeably poorer floats recovery relative to the coarser sizes. In addition, refuse carryover in Plant C showed an especially sharp increase between the 10-1.7 mm and 1.7-0.6 mm fractions. On the whole, the indications from the curves for Plant C are of an overloaded condition in the vessel.

Table 8 – Summary of overall washing results for heavy-medium vessels (1978-80)

	Plant		
	A 100-12.7 mm	B 127-10 mm	C 100-10 mm
Ash content (%)			
Raw coal	26.4	71.6	29.9
Reconstituted coal	27.0	68.0	28.3
Clean coal	11.8	7.9	12.7
Refuse	58.1	75.1	51.3
Yield of clean coal (%)	67.1	10.6	59.6
Theoretical yield (%)	68.2	13.1	66.1
Organic efficiency (%)	98.4	80.9	90.2
Separation density (dp)	1.552	1.370	1.492
Probable error	0.032	0.024	0.068
Error area	17	13	45
Imperfection	0.021	0.018	0.046
±0.10 Near density material (%)	17.9	18.0	27.5
Floats in refuse (%)	3.9	1.8	20.2
Sinks in clean coal (%)	0.9	19.1	3.7
Total misplaced material (%)	1.9	3.6	10.4
Yield error (%)	1.1	2.5	6.5
Ash Error (%)	0.2	0.7	0.9

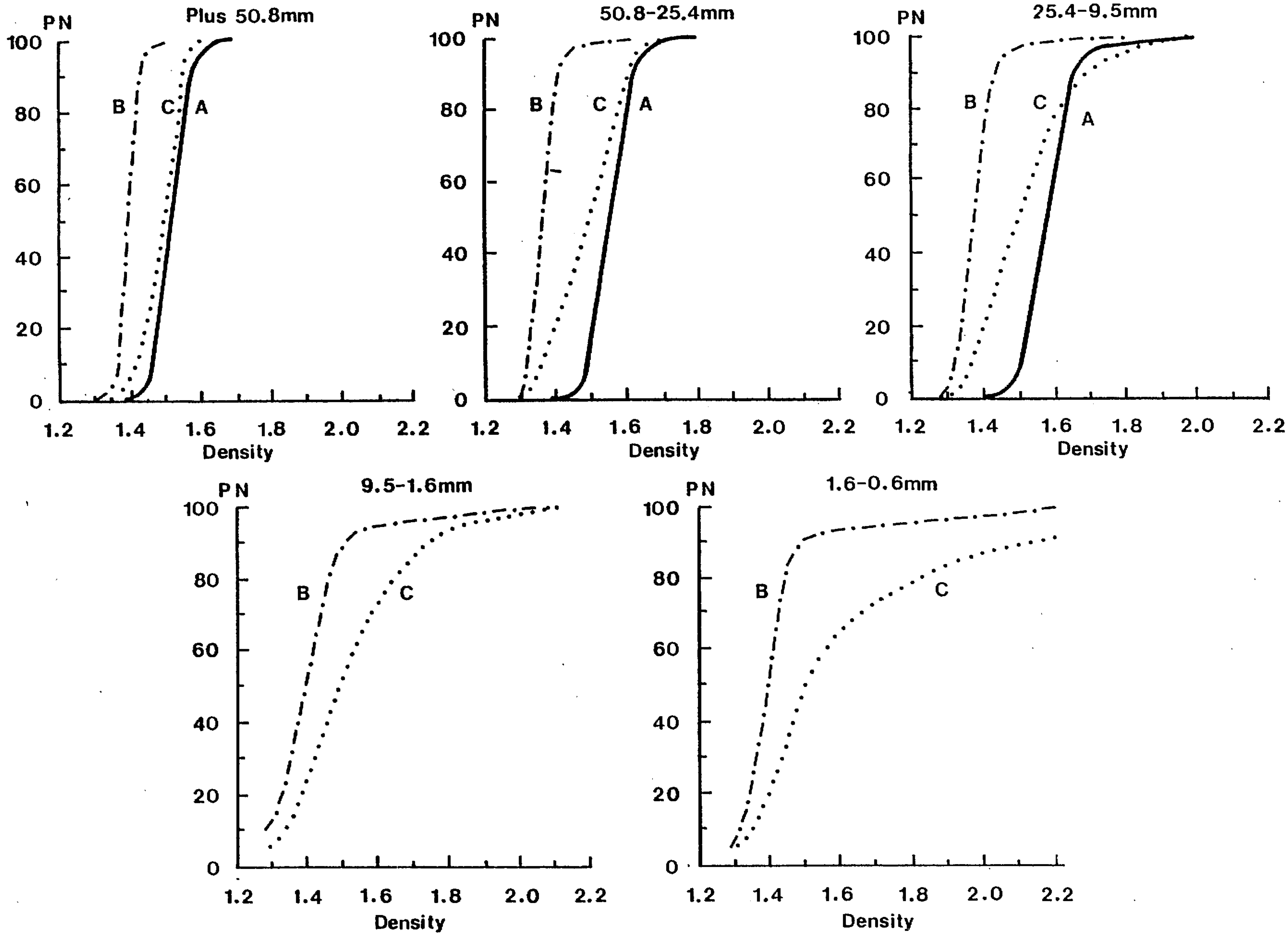

Fig. 12 – Partition curves for heavy-medium vessel separation of individual size fractions

Effect of Feed Characteristics

For the individual plants there was generally little variation in cutpoint and, with the exception of Plant C, relatively little deterioration in separation sharpness with decreasing particle size (Fig. 13). The data indicated that the differences in separation sharpness between the plants could be largely accounted for by the differences in percentage low-density coal in their respective feeds (F135). The high probable error for Plant C (0.068) was obtained in the separation of a feed with relatively high floats content. This suggests that an overloaded condition may have existed on the floats side of the vessel. It can be seen from the relationship obtained ($r = 0.9435$, $n = 18$),

$$R = 0.032 + 0.000068(F135)^2 - 0.000025(F135)D \qquad \text{Eq 2}$$

that although a minimum is indicated, the probable error tended to increase as the floats content of the feed increased and as the particle size decreased. Moreover, the effect of the floats was dependent upon particle size: the finer the particle, the greater the rate of increase in probable error and the smaller the amount of floats that could be tolerated for optimum separation sharpness. For the range of operating conditions in the three plants, it appears that best separation sharpness would probably have occurred for feeds containing no more than approximately 15-20% floats over the entire particle size range (Fig. 14).

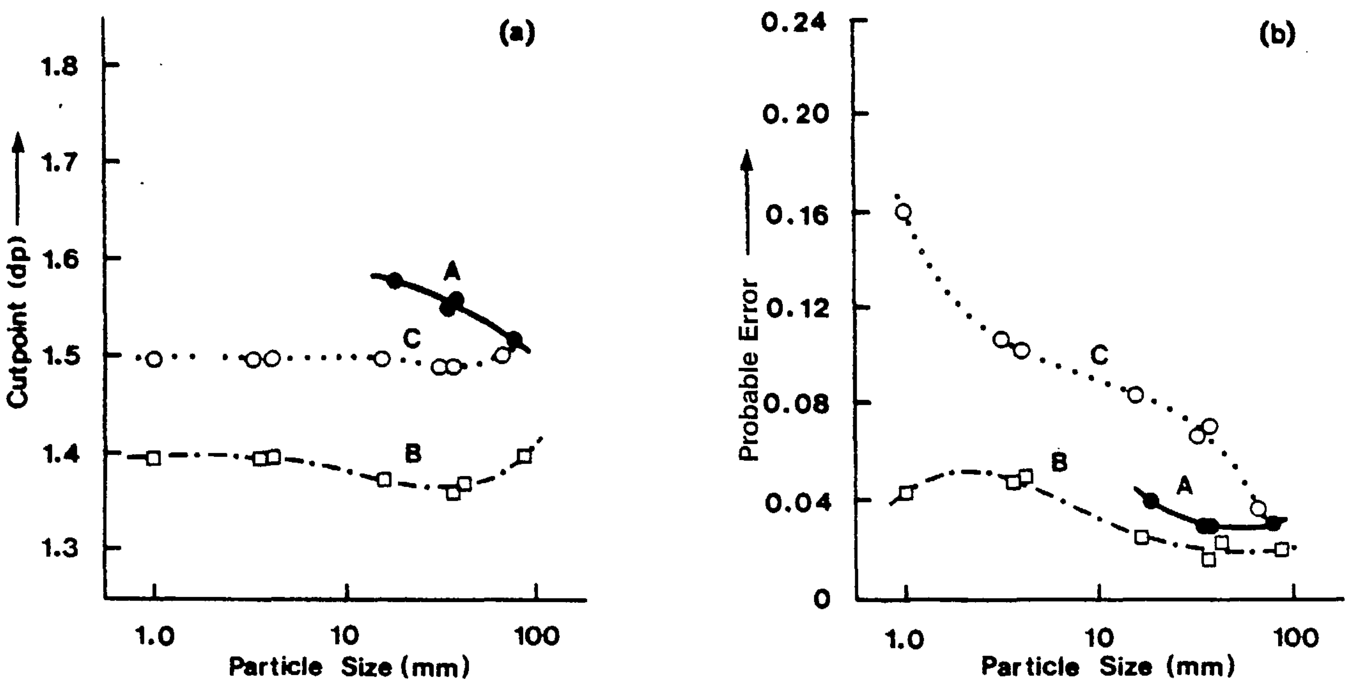

Fig. 13 – Variation of heavy-medium vessel cutpoint and probable error with particle size

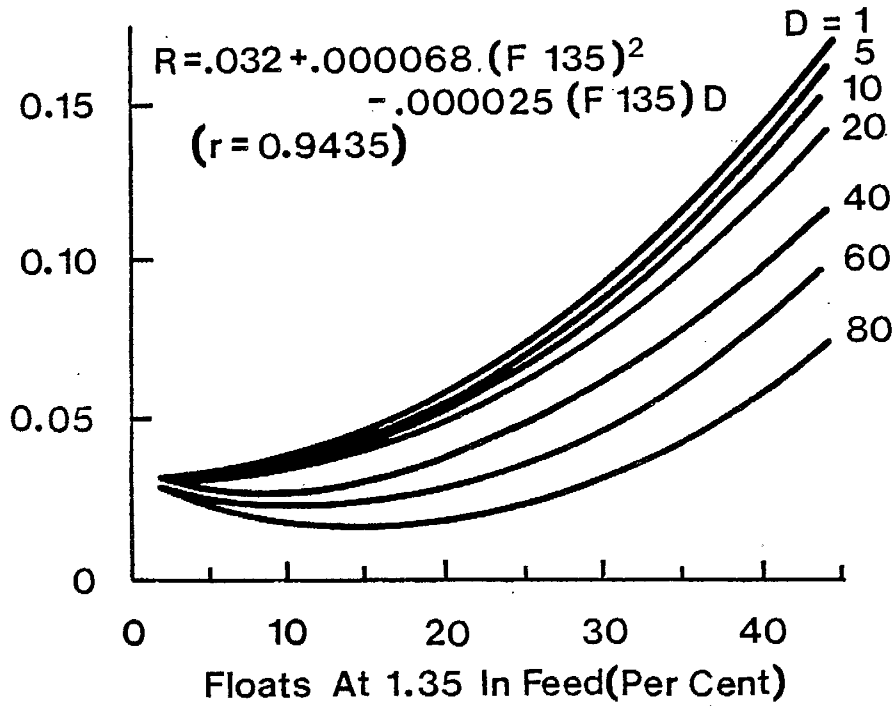

Fig. 14 – Effect of feed floats content and particle size on heavy-medium vessel separation sharpness

Separation Losses

For all size fractions in the range between 80 and 0.6 mm, organic efficiencies varied between 99.0 and 67.5% and yield losses between 0.5 and 11.6%. Yield errors tended to increase with increasing near-density material and decreasing particle size for a given sharpness of separation as measured by the imperfection (Fig. 15a). Product loss was roughly proportional to the yield error (Fig. 15b).

Table 9 shows that the estimated product loss for all plants ranged from 0 to 22.2% and tended to be smaller in the coarse fractions than in the fine. As percentages of the feed, overall losses for Plants A, B and C amounted to 0.14, 1.88 and 3.47% respectively. From the feed rates to the vessels in the individual washeries (Table 6), total product losses were estimated at 10.96 tph. This amounts to 1.49% of the feed or 2.74% of the total heavy-medium vessel clean coal output. It is noted that approximately 23% of these losses occurred in the minus 10 mm nominal undersize fractions i.e., 2.5 tph or 0.34% of the feed.

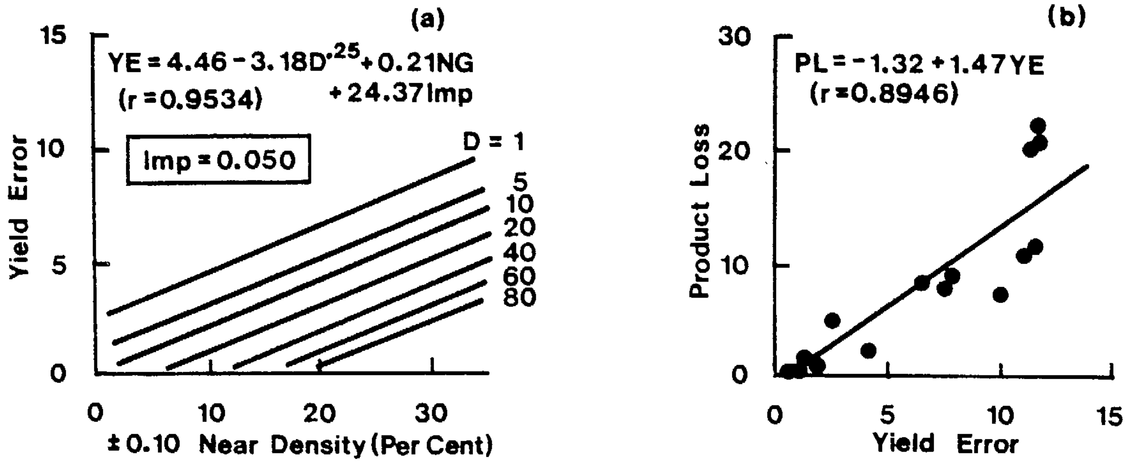

Fig. 15 – Recovery and product losses for heavy-medium vessels

Table 9 – Estimated clean coal losses in heavy-medium vessels

	Size fraction (mm)					Total
	+50	50-25	25-10	10-1.7	1.7-0.6	
Plant A						
Product ash (%)	11.4	12.1	11.7	—	—	11.8
Wt in refuse (%)	0	0.1	1.6	—	—	0.52
% of total feed	0	0.01	0.13	—	—	0.14
Plant B						
Product ash (%)	7.7	6.7	8.4	8.5	9.5	8.1
Wt in refuse (%)	0	0.8	2.3	11.5	7.3	2.18
% of total feed	0	0.18	0.78	0.78	0.14	1.88
Plant C						
Product ash (%)	12.5	12.4	13.2	11.3	10.6	12.6
Wt in refuse (%)	5.0	8.0	8.9	20.1	22.2	8.91
% of total feed	0.27	0.99	1.63	0.46	0.12	3.47

SUMMARY

1. The heavy-medium vessels treated coals in the nominal size range 127-10 mm with ash contents of 27-72% producing clean coals containing 7.9-12.7% ash at cutpoints between 1.370 and 1.552 relative density and with yields of 10.6-67.1% and organic efficiencies of 80.9-98.4%.
2. Overall separation sharpness for the plus 10 mm size range was normal in Plants A (R = 0.032) and B (R = 0.024) but below par in Plant C (R = 0.068).
3. The characteristic tendency towards refuse carryover for the finer particle sizes was especially marked in the 10-0.6 mm fractions.
4. Separation sharpness generally decreased with increasing feed floats content and increasingly so as particle size decreased.
5. The overloaded condition suggested by the partition curves for Plant C appears to have been on the floats side and could have resulted from the fact that floats content of the feed exceeded the indicated maximum of 20% for optimum performance as was achieved in Plants A and B.
6. Yield error in the individual size fractions varied between 0.5 and 11.6% and was proportional to the quantity of ±0.10 near-density material in the feed, tending to increase with decreasing separation sharpness and particle size.
7. Product losses in the individual size fractions varied between 0 and 22.2%, totalling 0.52% of the refuse for the plus 10 mm sizes in Plant A, 2.18% in Plant B and 8.91% in Plant C with both of the latter including losses in the 10-0.6 mm undersize.
8. The total clean coal loss amounting to 10.92 tph for all plants combined corresponded to 1.49% of the raw feed or 2.74% of clean coal output and included 2.51 tph lost in the nominal 10 mm undersize fraction.

HEAVY-MEDIUM CYCLONE

The heavy-medium cyclone is a relative newcomer to the coal preparation industry, having been developed during World War II at Dutch State Mines. It was first used commercially in 1948 in the Netherlands and shortly thereafter was adopted in several other European countries. The cyclone has since been widely accepted and is used in almost every coal-producing country in the world. In the past, various media such as barytes, flotation tailings and pulverized rock have been used, but magnetite is now employed almost exclusively. This is the case in Canada where the heavy-medium cyclone was first introduced in 1970 and where it now accounts for approximately 47% of all washed tonnage.

In its simplest terms, separation in the heavy-medium cyclone can be viewed as an accelerated version of the process described for the heavy-medium vessel. The principal difference arises from the fact that in the cyclone, the feed is made to rotate at high speed with the result that separation occurs under high g-force. Thus, by comparison, the cyclone is able to separate much finer particles in a much shorter time. Although capital and operating costs are higher than for many other fine coal processes, heavy-medium cyclones are capable of very sharp separations, operate efficiently at low and high cutpoints and cope well with large quantities of near-density material (11). They are most commonly used for metallurgical or difficult coals in the 25-0.6 mm size range. The practical 0.6 mm bottom limit has in the past been imposed more by magnetite reclamation considerations than by cyclone limitations as such. Good capability down to 0.2 or even 0.15 mm is said to be available and for this reason there appears to be some growing interest in extending the practice down to finer sizes (12,13). Although close attention is needed to maintain medium quality, more contamination can be tolerated than for the heavy-medium vessel. Losses of medium generally range between 0.25 and 2.5 kg/tonne, averaging approximately 1.2 kg/tonne of feed.

PROCESS CHARACTERISTICS

The most important factors affecting cyclone efficiency relate to medium selection, quality maintenance and density control during operation (11). However, it has also been observed that optimum separation sharpness will be achieved when ratios of the cyclone diameters are equal or close to the following values (14):

inlet/cyclone (d_i/d_c) = 0.20
overflow/cyclone (d_o/d_c) = 0.40
underflow/cyclone (d_u/d_c) = 0.30.

Significant deviation from these values would be expected to result in reduced separation efficiency with increased losses of floats in refuse and sinks in coal.

In practice, inlet pressure is most often in the range 69-96 kPa (10-14 psi). Although separation efficiency for the finer particle sizes can be improved by operating at higher pressures, increased wear of orifices usually results and thus can be a limiting factor. Separation sharpness of the heavy-medium cyclone tends to decrease as cutpoint increases and as particle size decreases. In addition to a possible influence of feed top size on separation sharpness for the finer particles, it may happen that separation would be at its best when the difference between the cutpoint and medium density is at a minimum (11). The cutpoint is usually higher but is occasionally lower than the medium density. The magnitude of the difference can vary according to several factors among which are the cyclone geometry, inlet pressure, medium grade and feed particle size. From Fig. 16, which is based on data covering the approximate cutpoint range 1.3-1.7 relative density, it can be estimated that, as particle size decreases to the 0.6 mm lower limit,

- probable error increases from 0.020 to 0.067;
- error area increases from 12 to 41.

PLANT DATA

During 1978-80, total installed heavy-medium cyclone capacity in six washeries amounted to 2410 tph of which approximately 80% (1930 tph) was for metallurgical coals with top sizes in the 38-10 mm range. Both single and two-stage circuits were used. Table 10 shows that for cyclone diameters between 51 and 71 cm, unit feed rates were between 37 and 69 tph. These feed rates ranged between 76 and 90% and averaged 80% of rated capacity based on cyclone diameter (Fig. 17). The plots of orifice ratios in Fig. 18 show that while the average d_u/d_c ratio for all six plants was exactly the prescribed value noted earlier (0.30), the average values of d_o/d_c and d_i/d_c were both higher than their respective optima (0.40 and 0.20). Geometry of the cyclone in Plant E was evidently at some variance with that of all other cyclones. Inlet pressures were between 54 and 96 kPa (8 and 14 psi) and within the range of normal practice. Reported magnetite losses of 0.4-2.4 kg/tonne were consistent with those that are generally observed.

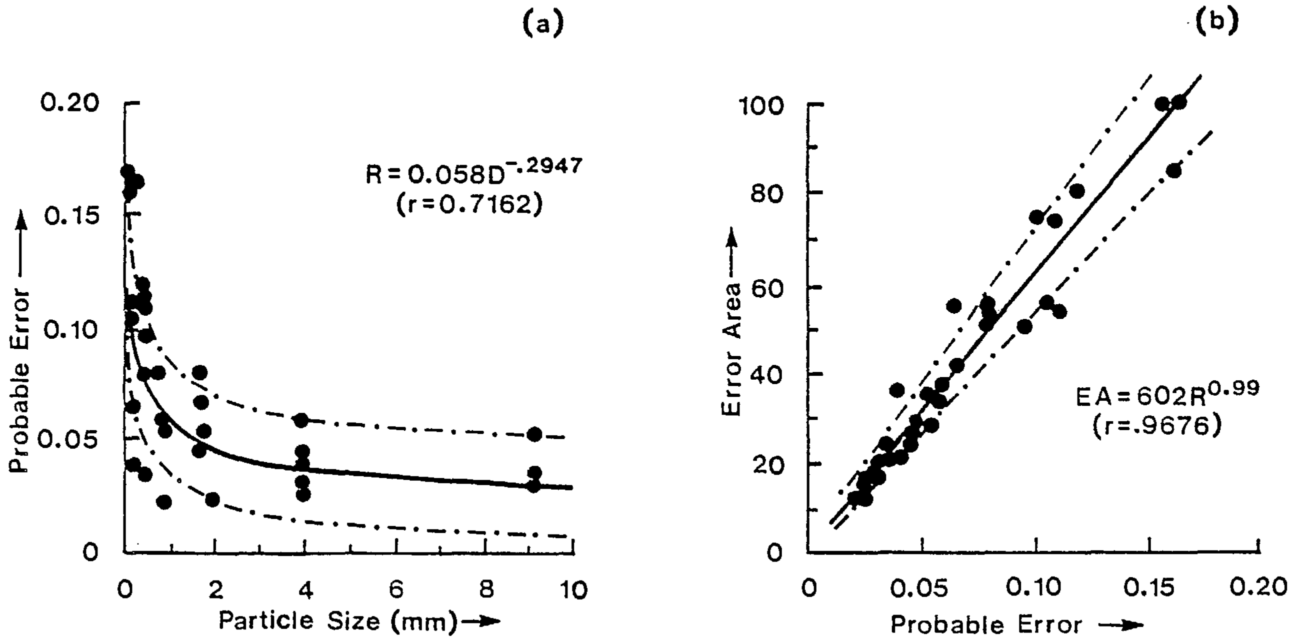

Fig. 16 – Heavy-medium cyclone separation characteristics (12,14)

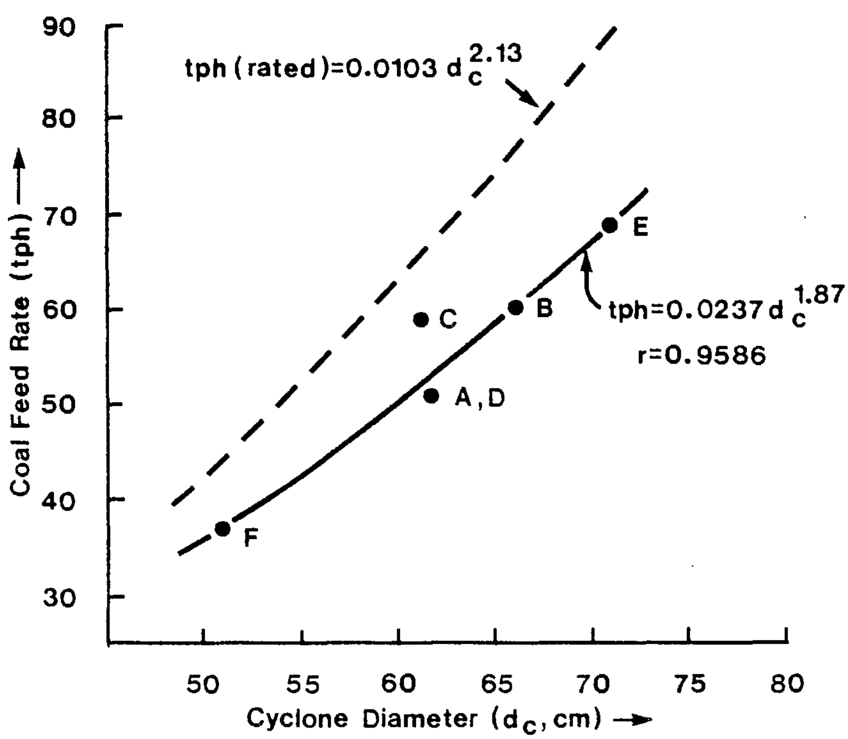

Fig. 17 – Comparison of plant feed rates and rated capacities of heavy-medium cyclones

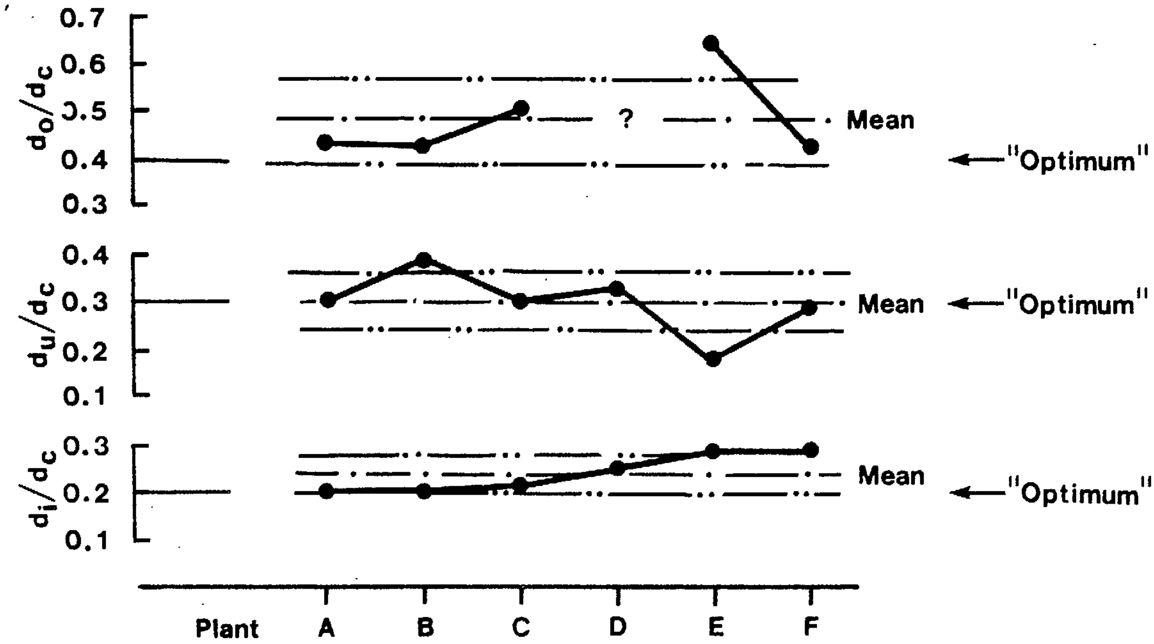

Fig. 18 – Heavy-medium cyclone orifice ratios

Table 10 – Operating Data for heavy-medium cyclone plants

	Plant					
	A	B	C	D	E	F
Cyclone diameter (cm)	61	66	61	61	71	51
Inlet diameter (cm)	12.1	13.0	12.7	15.2	20.3	15.0
Vortex finder diameter (cm)	26.0	27.9	30.5	—	45.7	21.5
Apex diameter (cm)	18.4	25.4	18.2	20.3	12.7	15.0
Feed rate (mtph)	50	60	59	50	61	37
Inlet pressure (kPa)	54	59		96	76	55
Feed top size (mm)	38	38	12.7	10	50	10
Feed solids (wt,%)	15-20	25-30	—	—	25	25
Medium density (g/cc)	1.40	1.34	1.45-1.55	1.50	1.45	1.42
Mean product yield (%)	70	70	—	77	84	75
Magnetite:						
% minus 44 μm	92-95	90	92	90	92	90
Consumption (kg/t)	2.4	1.2	0.4	1.0	0.9	2.0

Feed to the cyclones consisted of plus 0.6 mm deslimed run-of-mine and comprised between 44 and 70% of the washery feed. Of three plants using two-stage separation with rewashing of primary reject, two produced thermal grade middlings. The data shown in Fig. 19 and 20 and summarized in Table 11 indicate that the feeds could be categorized as coarse or fine with top sizes in the ranges 38-50 mm and 12.7-10 mm respectively. In general, the coarser feeds (Plants A, B and E) contained fewer middlings and lower percentages of undersize. The floats at 1.4 relative density tended to increase with decreasing particle size for the majority of the plants (Fig. 20). Ash contents varied between 19% (Plant E) and 32% (Plant A) and averaged 24%. The quantity of minus 0.5 mm undersize ranged between 1.0 and 8.2% and averaged 2.3% for the coarse feeds and 5.5% for the fine feeds.

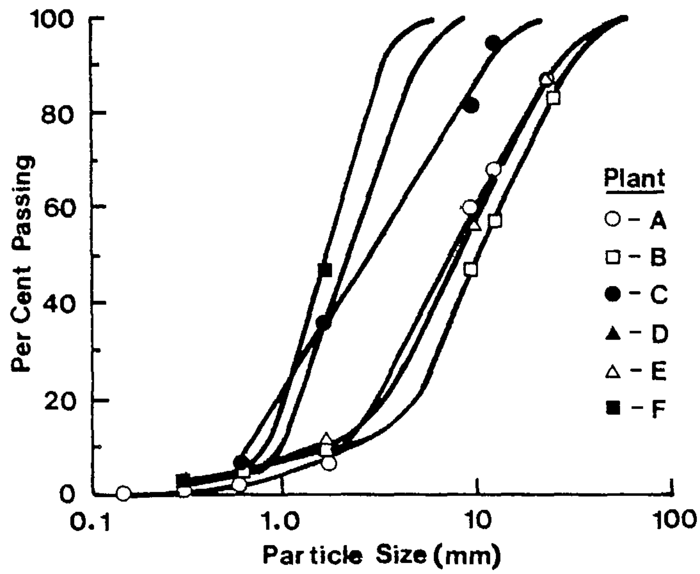

Fig. 19 – Size distribution of heavy-medium cyclone feeds (reconstituted)

Table 11 – Feed characteristics for heavy-medium cyclone plants

Plant	Raw feed			Density distribution *			Theoretical *			
	Nominal size range (mm)	Undersize (%)	Ash (%)	Floats @ 1.4 (%)	1.4-1.8 (%)	Sinks @ 1.8 (%)	Product ash (%)	Cutpoint (dp)	Product yield (%)	Reject ash (%)
A	38-0.6 ($\bar{D}$ = 10.0)	1.0	32	51	14	35	6.2	1.52	60	76
B	38-0.6 ($\bar{D}$ = 12.4)	2.9	20	81	3	16	2.3	1.33	79	69
C	12.7-0.6 ($\bar{D}$ = 5.7)	8.2	23	55	30	15	11.9	1.68	81	65
D	10-0.6 ($\bar{D}$ = 2.9)	2.9	25	52	32	16	12.8	1.65	82	65
E	50-0.6 ($\bar{D}$ = 11.3)	4.4	19	68	20	12	8.9	1.66	85	64
F	10-0.6 ($\bar{D}$ = 2.5)	5.4	27	51	26	23	10.3	1.60	74	75

* *From reconstituted feed washability data, plus 0.6 mm*

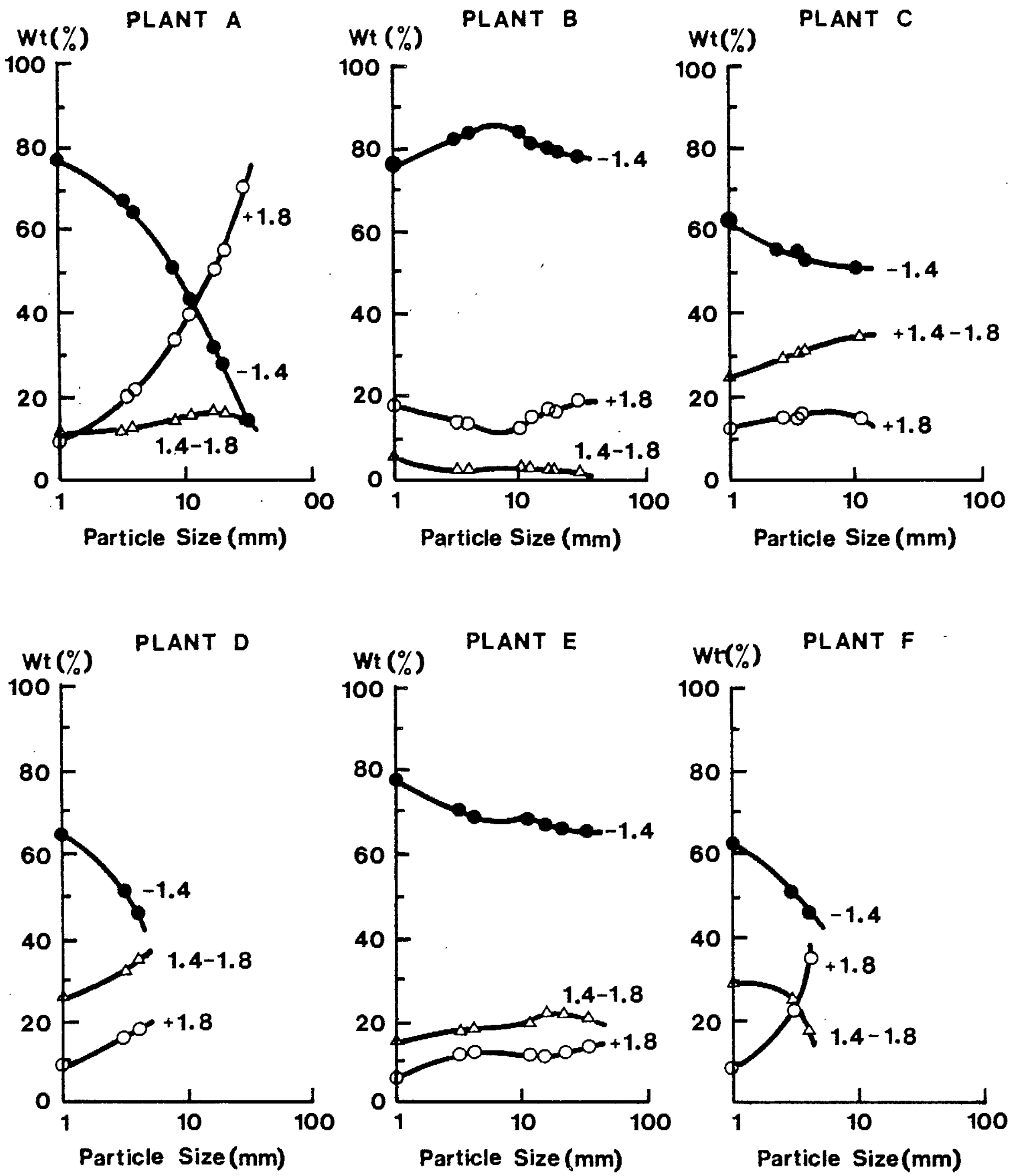

Fig. 20 – Variation of density distribution of heavy-medium cyclone feeds with particle size (reconstituted)

HEAVY-MEDIUM CYCLONE PERFORMANCE

Overall washing results for primary cyclones show that at cutpoints between 1.330 and 1.679, clean coal products containing 2.3-12.8% ash were obtained at yields of 59.4-83.9% with organic efficiencies between 96.2 and 99.9% (Table 12).

On the basis of the mean particle sizes of the feeds (25-12.4 mm) and probable errors of the individual plants (R = 0.020-0.102), reference to the relationship in Fig. 16 shows that whereas separation sharpness in Plants A, B and D was near average or better, that in Plants C and E was below average and that in Plant F much below average. The data suggested that the differences in operation could have been related to differences in feed undersize content and cyclone geometry. For example, in Plants A, B and D the feeds, on the average, contained significantly smaller quantities of minus 0.6 mm undersize (Table 11) and the cyclone apex diameters were significantly larger than in Plants C, E and F (Table 10).

As expected, the partition curves show that, with decrease in particle size, all plants exhibited some reduction in overall separation sharpness and in efficiency of coal recovery and refuse elimination (Fig. 21). However, both Plants C and F showed particularly poor refuse elimination in the minus 10 mm fractions. It seems that such behaviour could be expected to accompany a condition of instability in the cyclone, a condition that is manifested by intermittent apex discharge (surging). It can occur for separations involving coals with high fines content where the refuse contains a significant proportion of mid-density coal and where, because of the loss of its finest fractions, the magnetite needs replenishing (14).

Table 12 – Overall performance of heavy-medium cyclone plants: composite feed

	Plant					
	A 38-0.6 mm	B 38-0.6 mm	C 12.7-0.6 mm	D 10-0.6 mm	E 50-0.6 mm	F 10-0.6 mm
Ash content (%)						
Raw coal	32.1	18.6	21.4	25.4	17.4	28.2
Reconstituted coal	34.1	16.1	21.7	22.3	17.4	26.9
Clean coal	6.2	2.3	11.9	12.8	8.9	10.3
Refuse	75.1	60.8	59.0	65.2	61.5	68.5
Yield of clean coal(%)	59.4	76.4	79.0	81.9	83.9	71.5
Theoretical yield (%)	59.9	79.2	81.4	82.0	84.6	74.3
Organic efficiency (%)	99.2	96.5	97.0	99.9	99.2	96.2
Separation density (dp)	1.526	1.330	1.660	1.666	1.679	1.570
Probable error	0.025	0.025	0.052	0.036	0.059	0.102
Error area	14	11	41	20	35	73
Imperfection	0.048	0.061	0.079	0.054	0.057	0.179
±0.10 Near density material (%)	10.1	23.0	7.9	4.8	5.9	11.9
Floats in refuse (%)	3.3	13.5	11.0	4.0	11.1	11.8
Sinks in clean coal (%)	0.9	0.6	1.9	0.2	0.3	2.2
Total misplaced material (%)	1.9	3.6	3.8	0.9	2.0	4.9
Yield error (%)	0.5	2.8	2.4	0.1	0.7	2.8
Ash error (%)	0.1	0.1	0.7	0.03	0.2	0.7

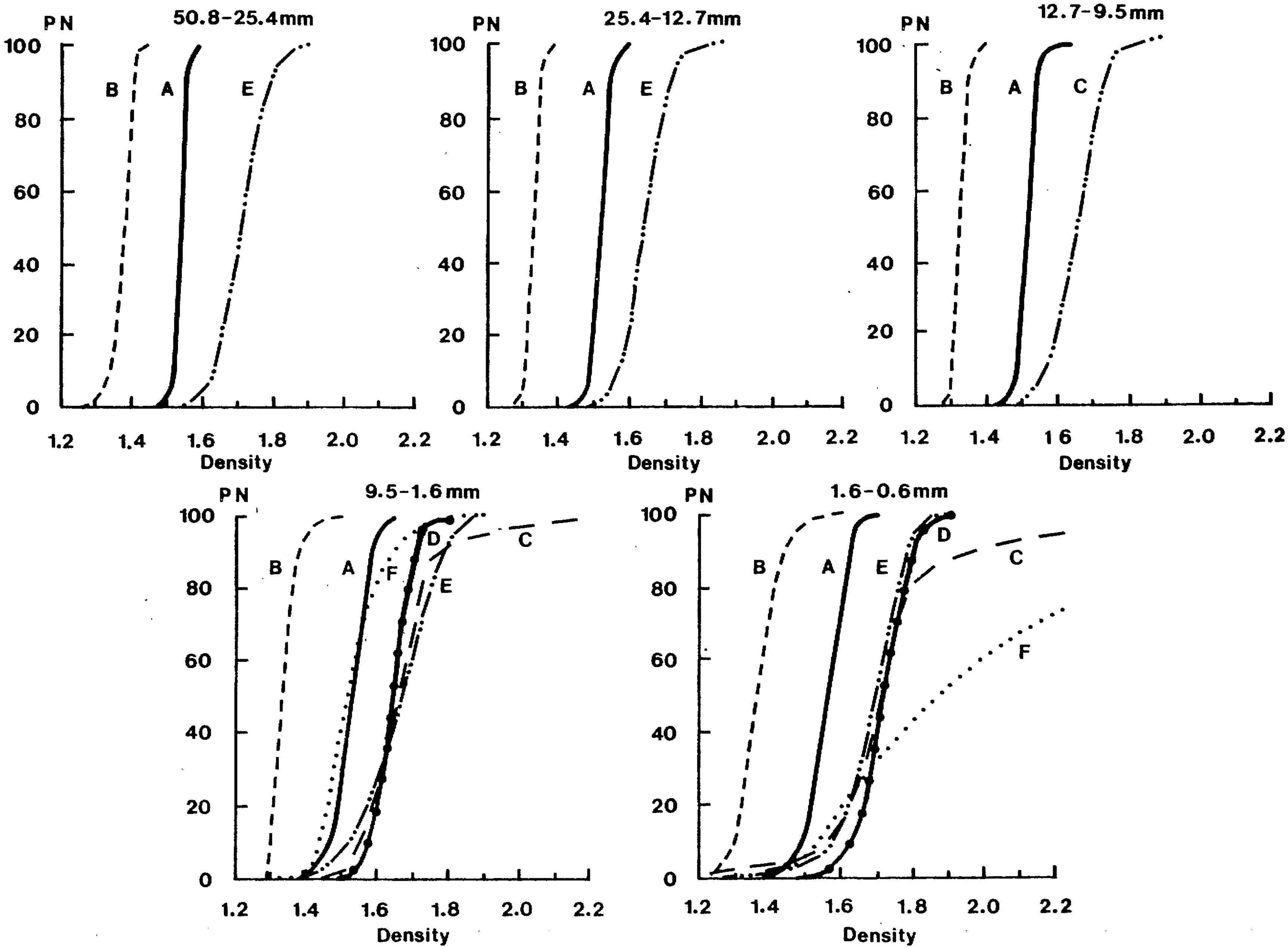

Fig. 21 – Partition curves for heavy-medium cyclone washing of individual size fractions

Effect of Cyclone Geometry and Feed Characteristics

In relation to cyclone operation in all other plants, that in Plant F showed an abnormally high increase in cutpoint and probable error with decreasing particle size (Fig. 22). This suggests that classification mechanisms were predominant in the separation of the finest size fraction. Because the quality of the recirculating medium in the plants was not known, the contribution of this factor to the result could not be determined in this case. Table 11 shows, however, that the feed to Plant F was among those containing the highest percentages of undersize (Plants C, E and F), the highest percentages of middlings (Plants C, D and F) and the highest percentages of high-density material (Plants A and F).

The available data indicated that the influence of cyclone design and of the operating variables on separation sharpness was far greater than that of any of the determined feed characteristics. This can be seen in the regression equation for probable error obtained for all observations including Plant F (correlation coefficient, $r = 0.9687$, $n = 34$),

$$R = -1.187 + 0.559dp + 0.119d_c + 27.676\, d_i/d_c - 3.555d_i^{0.5} - 0.0003D^2 \qquad \text{Eq 3}$$

and the regression equation for all observations excluding Plant F ($r = 0.9577$, $n = 31$),

$$R = 0.055 + 0.077dp - 0.101d_c/TPH - 0.0153D^{0.25} \qquad \text{Eq 4}$$

where no explicit terms relating to the feed occur other than the particle size (D). It is noted, furthermore, that for particle sizes smaller than approximately 13 mm, the last term in equation 3 is significant only to the third decimal of R and can be neglected.

Except for the fact that it accommodates the concept of an optimum ratio d_i/d_c described earlier, the general utility of equation 3 in its present form would be considered limited since it generates values R *S23* 0 (Fig. 23a). Nevertheless, it is interesting to note that for the observed range of cyclone dimensions, the optimum inlet diameter varied as d_c^2 such that probable error would be a minimum for $d_i = 0.0041d_c^2$. On this basis, cyclone F stood apart from all others by the fact that its inlet diameter was that only one to exceed the calculated optimum size.

Equation 4 based on the data for Plants A-E only shows that, as expected, separation sharpness in these heavy-medium cyclones decreased as cutpoint increased and particle size decreased (Fig. 23b). The equation shows, moreover, that performance was better in the larger diameter cyclones operated at relatively low solids feed rate.

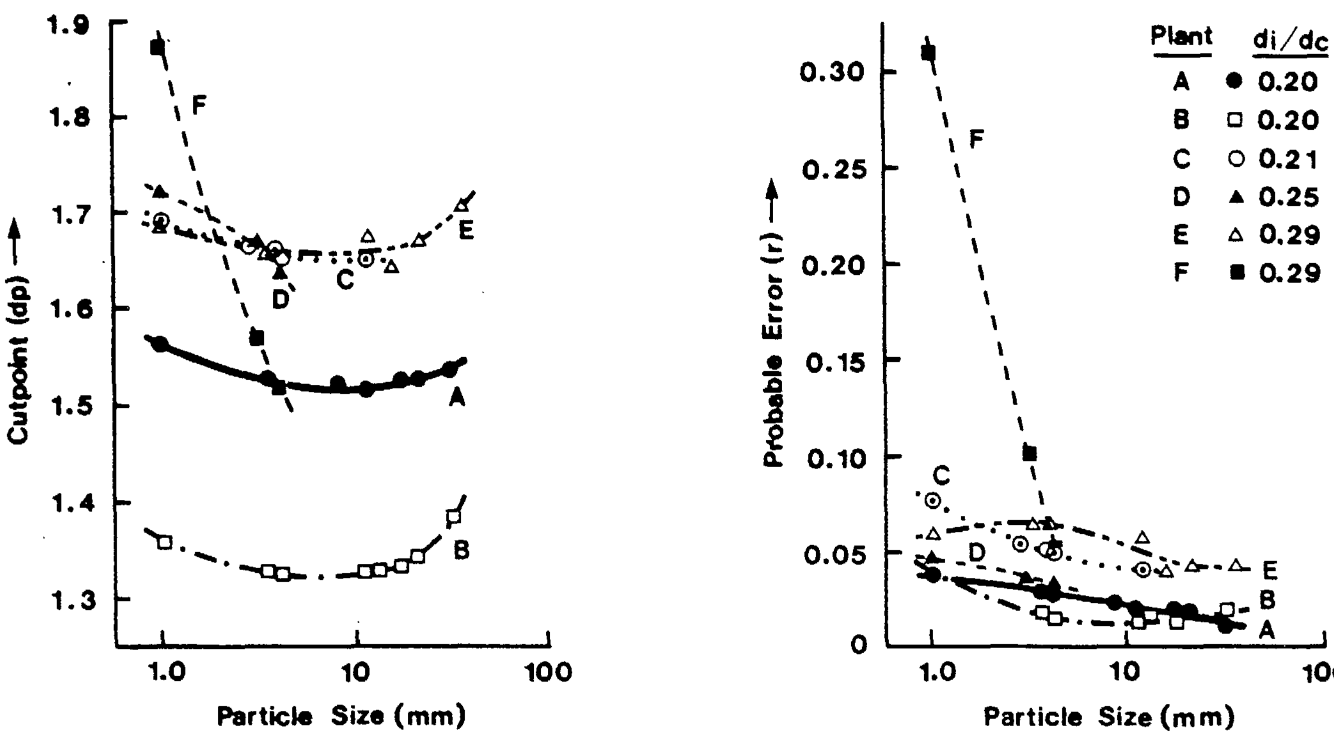

Fig. 22 – Variation of cutpoint and probable error in heavy-medium cyclone separation

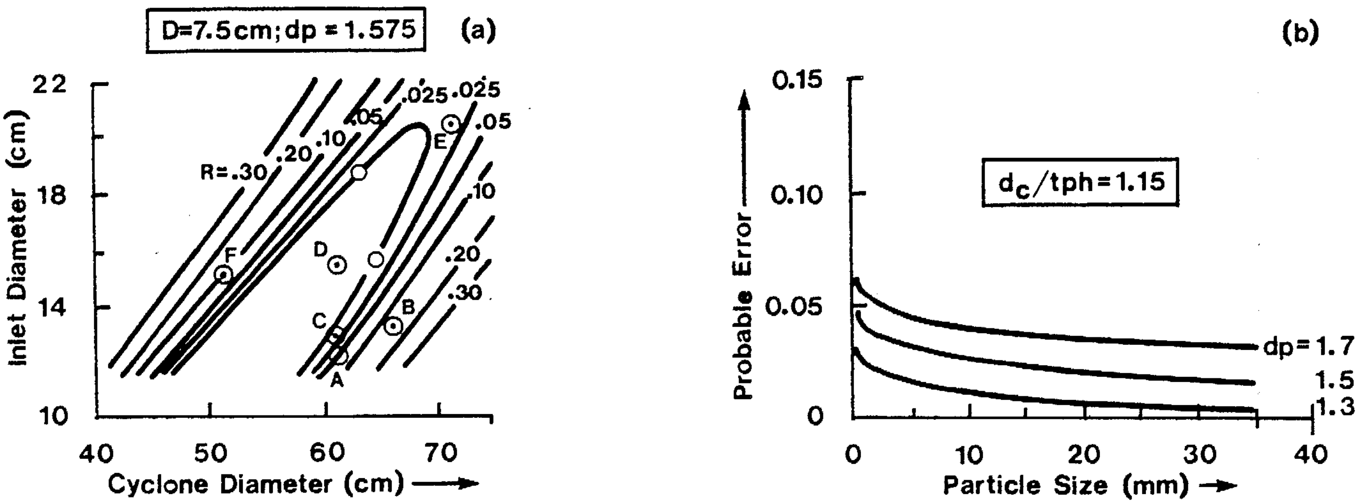

Fig. 23 – Effects of some variables on separation sharpness of heavy-medium cyclones

Separation Losses

For the full range of particle sizes between 36 and 0.6 mm, recovery efficiencies in the primary and single stage heavy-medium cyclones varied between 99.8 and 87.4% with corresponding yield losses of 0.3 to 9.5%. The data showed that yield error was greatest in the low cutpoint range, increased with increase in feed undersize content and was minimum at a particle size of 19 mm (Fig. 24a).

The relationship in Fig. 24b shows that a large apex orifice and high feed rate relative to cyclone diameter led to a higher product loss for a given yield error than did a smaller orifice and lower feed rate. Table 13 shows that while no measurable losses were found for Plant A, the rejects of the other 5 plants contained between 0.7 and 10.9% clean product in the 2.3 to 12.8% ash range. These losses corresponded to between 0.12% (Plant D) and 2.45% (Plant B) of the total feed to the cyclones.

From the total installed capacities of the plants (276-600 tph), calculated product losses varied between 0 and 11.7 and totalled approximately 17.98 tph. This represents 3.0% of the overall hourly reject production in the primary cyclones. Approximately 61% of this production was rewashed in second-stage cyclones. Prediction calculations indicated that an estimated 9.3 tph of clean coal could have been recovered in the plants providing retreatment, leaving a net loss of 8.62 tph for all operations combined. This net loss amounts to an average of 0.35% of the total raw feed to the heavy-medium cyclones or 0.47% on the clean coal basis.

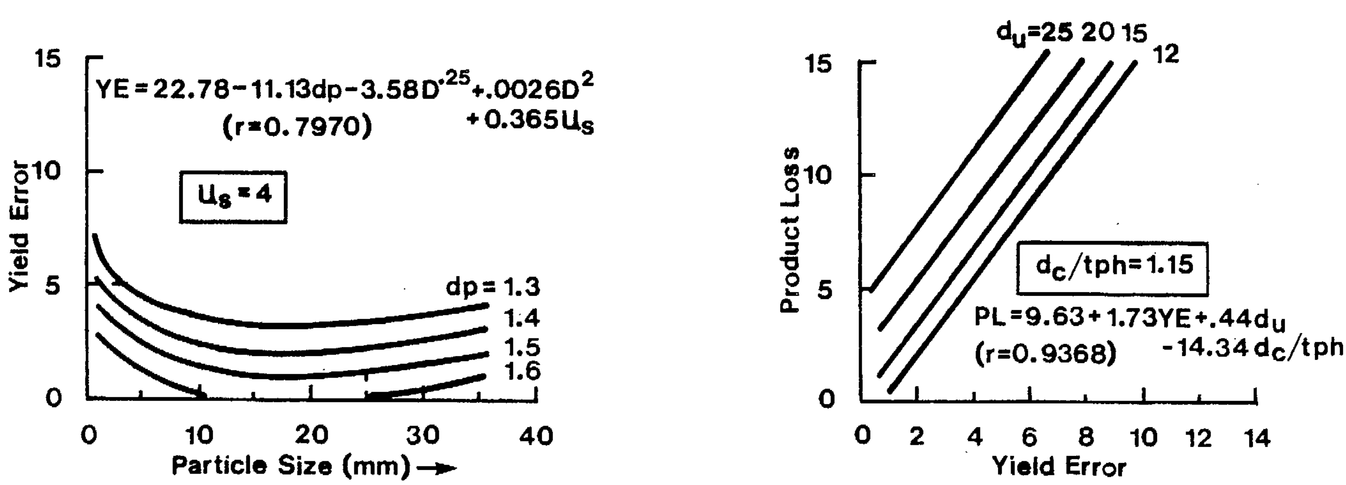

Fig. 24 – Recovery and product losses for primary and single-stage heavy-medium cyclones

Table 13 – Estimated clean coal losses in heavy-medium cyclone rejects

	Size fraction (mm)					Total
	Plus 25.4	25.4-12.7	12.7-9.3	9.3-1.7	1.7-0.6	
Plant A						
Product ash (%)	10.3	9.8	6.8	5.2	5.3	6.2
Wt in refuse (%)	0	0	0	0	0	0
% of total feed	—	—	—	—	—	—
Plant B						
Product ash (%)	3.2	2.5	2.2	2.0	1.7	2.3
Wt in refuse (%)	9.3	9.3	8.0	11.5	20.8	10.9
% of total feed	0.34	0.60	0.16	1.00	0.35	2.45
Plant C						
Product ash (%)	—	—	13.4	12.3	10.6	11.9
Wt in refuse (%)	—	—	3.9	5.4	6.5	5.5
% of total feed	—	—	0.10	0.58	0.33	1.01
Plant D						
Product ash (%)	—	—	—	13.8	10.8	12.8
Wt in refuse (%)	—	—	—	0.6	1.3	0.7
% of total feed	—	—	—	0.08	0.04	0.12
Plant E						
Product ash (%)	11.7	9.8		8.0	6.5	8.9
Wt in refuse (%)	2.6	3.1		3.0	2.7	2.9
% of total feed	0.06	0.12		0.25	0.02	0.45
Plant F						
Product ash (%)	—	—	—	10.2	10.3	10.3
Wt in refuse (%)	—	—	—	0.5	2.9	0.8
% of total feed	—	—	—	0.11	0.12	0.23

SUMMARY

1. The primary and single-stage heavy-medium cyclones treated coals with top sizes between 38 and 10 mm and ash contents of 19-32%, producing clean coals containing 2.3-12.8% ash at cutpoints between 1.330 and 1.679 relative density with yields of 59.4-83.9% and organic efficiencies of 96.2-99.9%.
2. Overall separation sharpness was near average or better in Plants A, B and D ($r = 0.020$-0.036), slightly below average for Plants C and E ($r = 0.052$-0.059) and poorest in Plant F ($r = 0.102$).
3. Cutpoint and probable error increased rapidly with decrease in particle size for Plant F indicating that classification was a predominant factor in separation of the finest size fraction.
4. Separation sharpness was primarily a function of cyclone design and operating variables with little perceptible influence of feed characteristics other than particle size, and was best for large particles at low cutpoints and in those cyclones where the inlet diameter fell in an optimum range defined by $d_i = 0.0041 d_c^2 \pm 3.2$ cm, approximately.
5. For those cyclones having inlet diameter d_i in the optimum range (Plants A-E), separation sharpness was best for large particles, at low cutpoints and in large cyclones where the ratio of cyclone diameter to feed rate was greatest.
6. Yield error varied between 0.3 and 9.5%, was greatest at low cutpoints, was proportional to undersize content in the feed and tended towards a minimum at a particle size, $D = 19$ mm.
7. The product loss, which ranged between 0 and 10.9% and represented up to 2.45% of total feed to the cyclones, was proportional to yield error, was higher in those cyclones having large apex diameters and was lower where the ratio of cyclone diameter to the feed rate was greater.
8. An estimated net product loss of 8.6 tph for all plants combined amounted to 0.36% of the raw feed or 0.47% of total heavy-medium cyclone clean coal production.

HYDROCYCLONE

The hydrocyclone, or water cyclone, appeared simultaneously with the heavy-medium cyclone as the outcome of development work at Dutch State Mines (DSM) during World War II. It wasn't until relatively recently, however, that the hydrocyclone's capabilities won it a place alongside tabling and flotation in the problematic area of fine coal preparation. Much practical development can be credited to a Canadian effort beginning in the early 1960's that was aimed at resolving the problem of washing friable western mountain coals (15). There is heavier reliance on the hydrocyclone in Canada than in other countries for fine coal cleaning (Fig. 1). Today the hydrocyclone accounts for approximately 21% of all washed tonnage in Canada.

Separation in the hydrocyclone, as in its heavy-medium counterpart, occurs under high g-forces as a result of the rotational velocity of the feed flow. Evidently, from its name, the hydrocyclone uses only water as the separating medium with no added solids other than the feed itself. The cyclone can be used for all ranks of coal and has the advantage of basic simplicity, high capacity and low capital cost. Although it can stand on its own in particular applications, the hydrocyclone is most widely used in conjunction with other processes such as flotation. This is very often the case for separations involving fine pyritic or oxidized coals. Generally speaking, the hydrocyclone requires two-stage operation to achieve high quality of both clean coal and refuse products. It is not considered suitable for metallurgical coals requiring separation at low cutpoints, for difficult coals or for those containing more than 40% refuse (16). Water and power requirements are relatively high and although operation is basically simple, some degree of operator skill and experience is needed.

PROCESS CHARACTERISTICS

There exist a number of proprietary hydrocyclones which are distinguishable primarily by differences in geometry involving such features as cone angle and inlet opening. Although the exact relationships between cyclone geometry, feed characteristics and separation performance are highly complex and not fully understood, it appears to hold with some room for deviation that ratios of the diameters should ideally be within the following ranges (17):

inlet/cyclone (d_i/d_c) = 0.15-0.30
overflow/cyclone (d_o/d_c) = 0.20-0.35
underflow/overflow (d_u/d_o) = 0.35-0.65.

It has also been shown experimentally that as the included angle of the cone increases from 45° to 95°, quality of the clean coal separation improves and is best at the largest angle whereas that of the reject deteriorates, it being best at the smallest angle (18). This behaviour places a strict limitation on design alternatives and explains the difficulty of achieving high quality of both products in single-stage operation.

The first requirement for optimum performance of the hydrocyclone lies in close control of the cutpoint during operation. This is not always an easy matter since the cutpoint is sensitive to feed solids content and especially so at low concentrations. Evidently, it is important to take steps that will not only minimize variation in the feed solids but that will also ensure a sufficiently high concentration to minimize cutpoint sensitivity. In practice, hydrocyclone feeds containing 6-10% solids by volume are the norm. Compensation for variation within this range could be achieved through means for continuous adjustment of the apex opening or of the vortex finder length (17,19). Inlet pressure basically governs cyclone throughput. It is usually in the range 48-240 kPa (7-35 psi) and is normally higher for larger diameter cyclones.

Hydrocyclones tend to suffer high losses of coal to the refuse. These losses usually increase as particle size decreases but can often be reduced by operating at somewhat higher inlet pressures. However, below approximately 0.15 mm, classification mechanisms normally predominate and the particles generally separate according to the water split regardless of density. The usual result is that the majority of fine particles report to the clean coal overflow product. This can be seen in Fig. 25a which shows an increasingly rapid rise in cutpoint as particle size decreases in the finer size ranges. The regression curves in Fig. 25, based on data from a U.S. study of (30.5-50.8 cm diameter) cyclone operation, also show the effect of solids throughput on the cutpoint and separation sharpness (20).

On average it can be expected that, at constant feed solids and as particle size decreases in the 4 to 0.20 mm range,

- cutpoint will decrease to a minimum then rise sharply;
- probable error will increase from 0.10 to 0.23;
- error area will increase from 61 to 124.

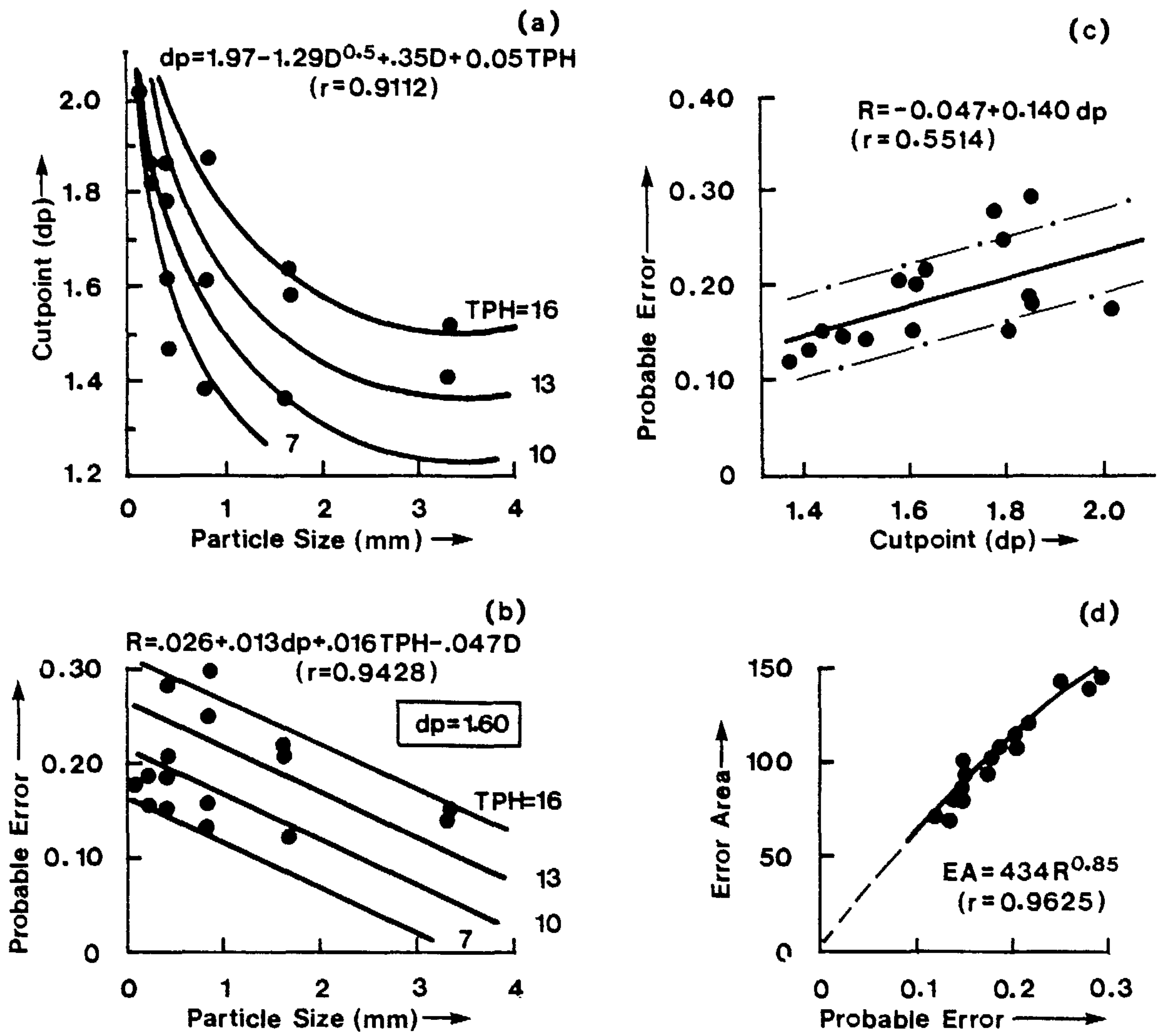

Fig. 25 – Hydrocyclone separation characteristics (20)

PLANT DATA

Of the seven washeries that employed hydrocyclones during 1978-1980, two used single-stage, four used two-stage and one used both single- and two-stage operation. Total installed capacity was approximately 1400 tph of which 75-80% was for metallurgical coals with top sizes between 32 and 0.6 mm. In addition, 84% of this total capacity (1176 tph) was used for 2-stage pretreatment of flotation and table feeds (66%) and for independent washing (18%) of coal with 3-0.6 mm top size. Single-stage washing, which constituted the balance of applications, was used only for coals with large top size.

Two types of hydrocyclones were in use, those commonly referred to as the DSM on the one hand and as the CWC (Compound Water Cyclone) on the other. Diameters varied between 20 and 61 cm and feed rates between 2-5 and 82 tph (Table 14). In the majority of cases, these feed rates were slightly below rated capacity based on cyclone diameter (Fig. 26). The inlet pressures of 93-241 kPa (13-35 psi) were in the normal range. Although the overflow/cyclone ratio, $d_o/d_c = 0.50$ for all plants was higher than the ideal maximum value of 0.35 cited earlier, the ratios d_u/d_o and d_i/d_c generally fell within the recommended ranges. However, Fig. 27 shows that, based on the available information of their geometry, cyclones E and G differed from all the others and departed from the mean in at least one respect each.

For all seven washeries, the hydrocyclone feed varied between 20 and 87% and averaged 35% of the run-of-mine. With only one exception, the feed originated as the undersize of raw coal screens preceding heavy-medium and jig-washing operations. The clean coal was either the total cyclone overproduct or the overproduct screen oversize with provision made for retreatment of the undersize. All plants but one re-treated the primary cyclone reject using either second-stage cyclones or other processes.

The data plotted in Fig. 28 and 29 and the summary given in Table 15 show that the hydrocyclone feeds varied considerably in size and washability characteristics. The coals can be roughly classified as "coarse" or "fine" with top sizes averaging 24 and 4 mm respectively. In addition, grouping according to ash content gives low (Plants A and B) medium (Plants B, D, E and F) and high ash (Plants C and G) feeds containing on the average, 12, 30 and 60% ash respectively. No distinction could be made between the low and high ash feeds on the basis of mean particle size: the highest ash coals, C and G, and likewise the lowest ash coals, A and B, were all among the coarsest and finest feeds (Table 15).

Table 14 – Operating data for hydrocyclone plants (1978-80)

Plant	A	B	B′	C	D	E	F	G
Cyclone type	DSM	CWC	CWC	CWC	CWC	DSM	CWC	DSM
Cone type	75°	L	L	L	L	75°	L	75°
Cyclone diameter (cm)	61	61	20	61	30	36	30	25
Inlet diameter (cm)	15.2	15.2	5.1	15.2	7.0	12.7	7.0	5.1
Vortex finder diameter (cm)	30.5	30.5	10.2	30.5	15.2	—	15.2	—
Apex diameter (cm)	15.2	15.2	5.1	15.2	7.6	—	7.6	4.0
Vortex finder clearance (cm)	—	19.4	4.4	—	10.2	—	6.3	11.0
Unit feed rate (tph)	74	82	2-5	23	5	9	9	—
Inlet pressure (kPa)	—	241	145	—	103	—	93	207
Feed top size (mm)	19	32	3	25	0.6	0.6	0.6	0.6
Feed solids content(%)	—	13	12-20	—	10-15	—	7	5-10
Average yield (%)	65	80	80	—	90	—	—	—

Table 15 – Feed characteristics for hydrocyclone plants (1978-80)

Plant	Raw feed			Density distribution *			Theoretical *				±0.10 Near density (%)
	Nominal size range (mm)	Minus −0.15 mm fines (%)	Ash (%)	Floats @ 1.4 (%)	1.4-1.8 (%)	Sinks @ 1.8 (%)	Product ash (%)	Cutpoint (dp)	Product yield (%)	Reject ash (%)	
A	19-0.15	0	11.6	69	29	2	10.6	1.77	97.7	63	<5
B	19 – 0	20	23	52	14	34	7.8	1.84	67	76	<5
B'	7 – 0	18	12	55	20	25	14.0	(2.3)	90	70	—
C	34 – 0	8	62	24	12	64	15.8	(2.1)	38	88	—
D	3.2 – 0	42	19	52	29	19	10.6	2.1	87	69	—
E	4 – 0	31	18	66	26	8	10.8	2.0	96	62	—
F	3.2 – 0	41	17	58	28	14	15.0	>2.3	>99	—	—
G	1.2 – 0	31	57	10	30	60	32.0	(2.3)	65	90	—

* *From reconstituted feed washability data: plus 0.6 mm – Plants A, B, C,*
plus 0.15 mm – Plant B'
plus 0.10 mm – Plants D – G.

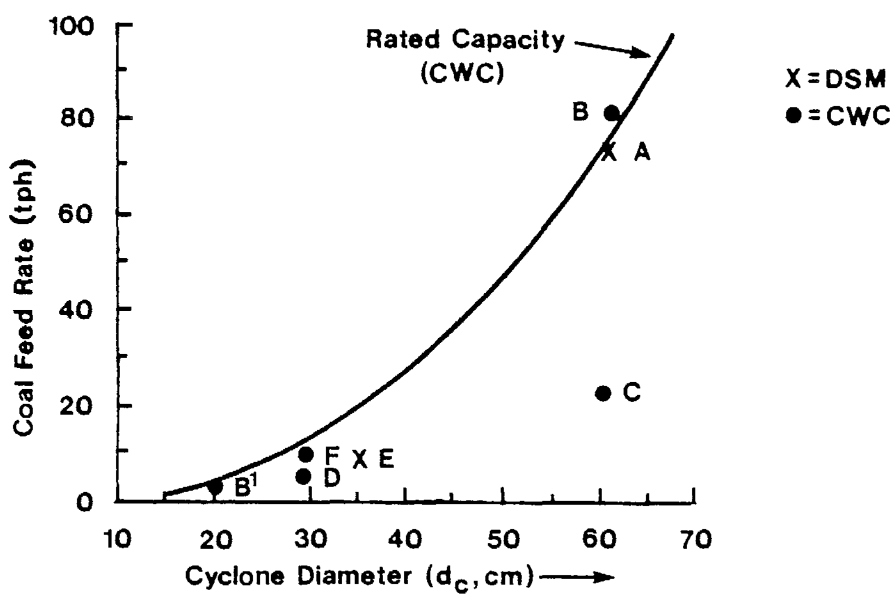

Fig. 26 – Unit solids feed rate for hydrocyclone plants

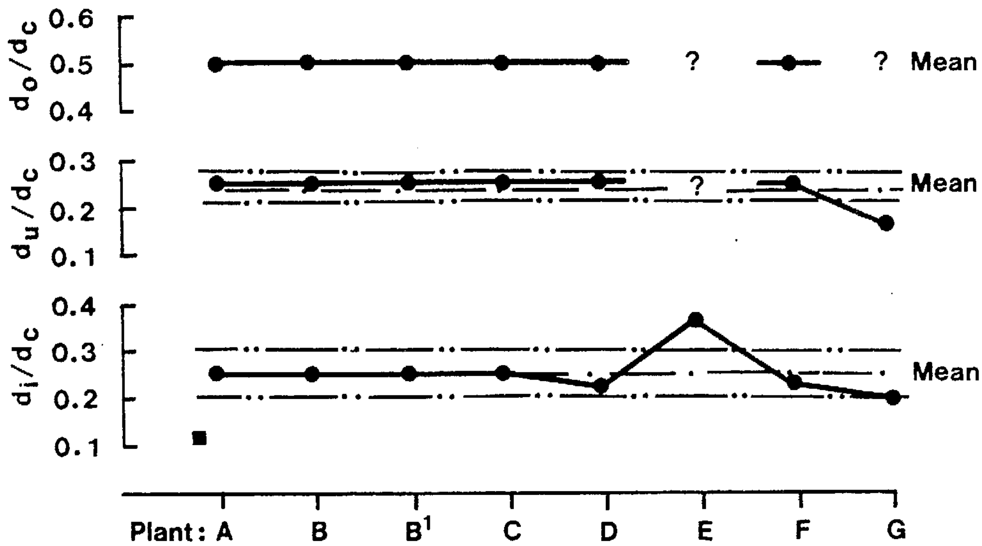

Fig. 27 – Hydrocyclone orifice ratios

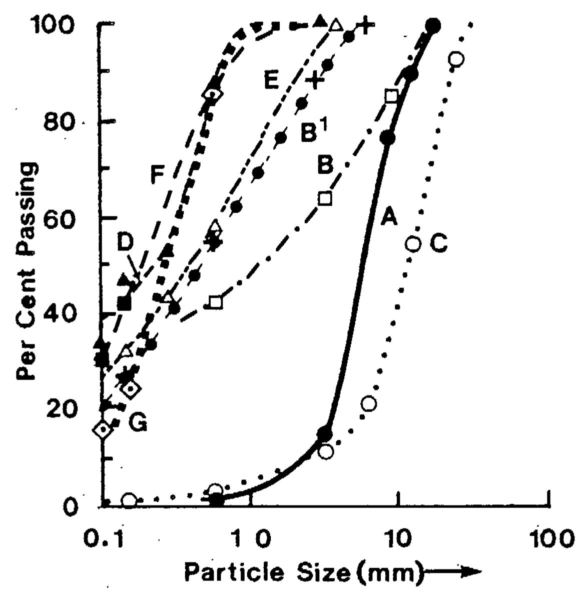

Fig. 28 – Size distribution of hydrocyclone feeds (reconstituted)

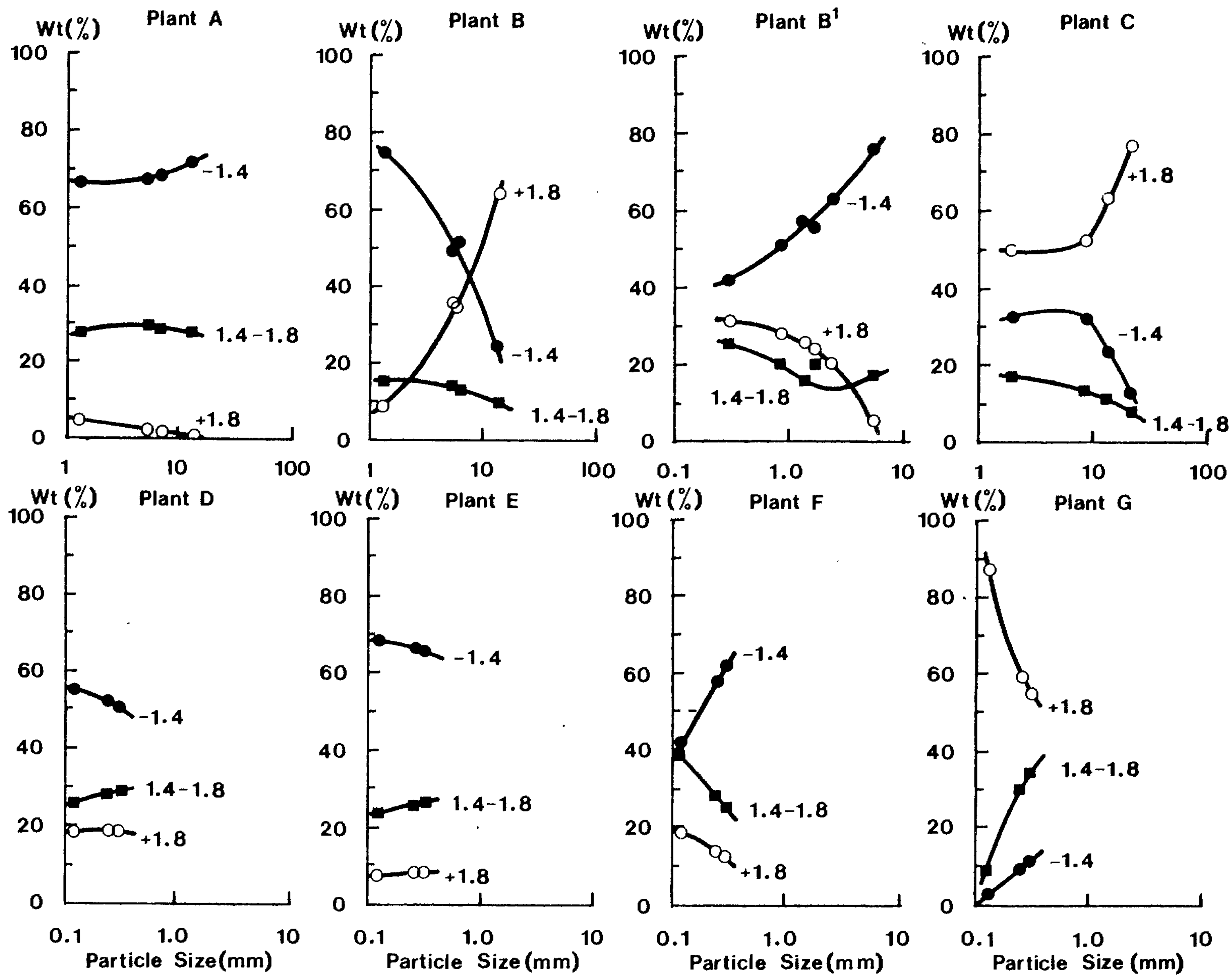

Fig. 29 – Variation of density distribution of hydrocyclone feeds with particle size (reconstituted)

HYDROCYCLONE PERFORMANCE

With the exclusion of Plant F where little cleaning was achieved, the overall washing results for primary cyclones in Table 16 show that at cutpoints between 1.59 and 2.25, clean coal products containing 7.8-32.0% ash were obtained at yields of 38.4 to 97.7% with organic efficiencies between 53.3 and 95.1%. Comparison of the refuse ash contents with the theoretical values indicates that high losses of coal occurred in the primary underflow, confirmed by the floats in refuse which ranged between 9.1 and 86.2% (Table 15).

Because the mean particle size of the composite feeds for three of the plants (A, B and C) was not within the range < 4 mm shown in Fig. 25b, evaluation of cyclone performance for all plants was by reference to the relationship between probable error and cutpoint (Fig. 25c). On this basis and the data in Table 16, separation sharpness in Plant E is judged to have been much better than average (r = 0.116 at dp = 2.022), that for Plants A, B, B′ and C near to average (r = 0.200-0.364 at dp = 1.590-2.253) and that in Plants D, F and G at best, only fair (r = 0.326-0.403 at dp = 1.610-1.640).

Although the partition curves in Fig. 30 generally show a decrease in separation sharpness and an increase in both coal and refuse losses as particle size decreased, no consistent pattern of variation between the individual operations appears. For example, the curves disclosed that as particle size decreased in the individual plants, cutpoint either increased, decreased, or passed through either a minimum or a maximum (Fig. 31). Behaviour of the probable error presents a similarly confused picture.

Table 16 – Overall performance of hydrocyclone plants: composite feed

	Plant							
	A 19-0.6 mm	B 19-0.6 mm	B′ 7-0.15 mm	C 25-0.6 mm	D 0.6-0.1 mm	E 0.6-0.1 mm	F 0.6-0.1 mm	G 0.6-0.1 mm
Ash content (%)								
Raw coal	11.5	23.7	11.4	62.7	17.8	18.3	14.7	55.9
Reconstituted coal	11.8	30.2	19.9	60.5	18.0	12.6	15.7	52.2
Clean coal	10.6	7.8	14.1	15.8	10.6	10.8	15.1	31.8
Refuse	25.6	55.2	48.2	81.1	29.2	32.3	41.8	63.0
Yield of clean coal (%)	92.2	52.6	83.0	31.5	60.4	91.8	98.0	34.6
Theoretical yield (%)	97.7	67.0	89.5	38.4	87.4	96.5	99.4	64.9
Organic efficiency (%)	94.4	78.5	92.7	82.0	69.1	95.1	98.6	53.3
Separation density (dp)	1.890	1.590	2.253	1.855	1.640	2.022	—	1.610
Probable error	0.246	0.200	0.364	0.274	0.403	0.116	—	0.326
Error area	151	118	186	142	196	84	—	172
Imperfection	0.276	0.339	0.291	0.320	0.630	0.114	—	0.534
±0.10 Near density material (%)	1.2	5.9	5.0	1.1	7.2	2.6	—	12.8
Floats in refuse (%)	86.2	27.0	49.8	9.1	57.2	67.5	—	17.7
Sinks in clean coal (%)	0.5	6.4	6.5	5.0	10.1	0.2	—	41.0
Total misplaced material (%)	7.2	16.2	13.9	7.8	28.7	5.7	—	25.8
Yield error (%)	5.5	14.4	6.5	6.9	27.0	4.7	1.4	30.3
Ash error (%)	1.4	4.7	3.6	5.1	6.1	1.3	1.2	21.1

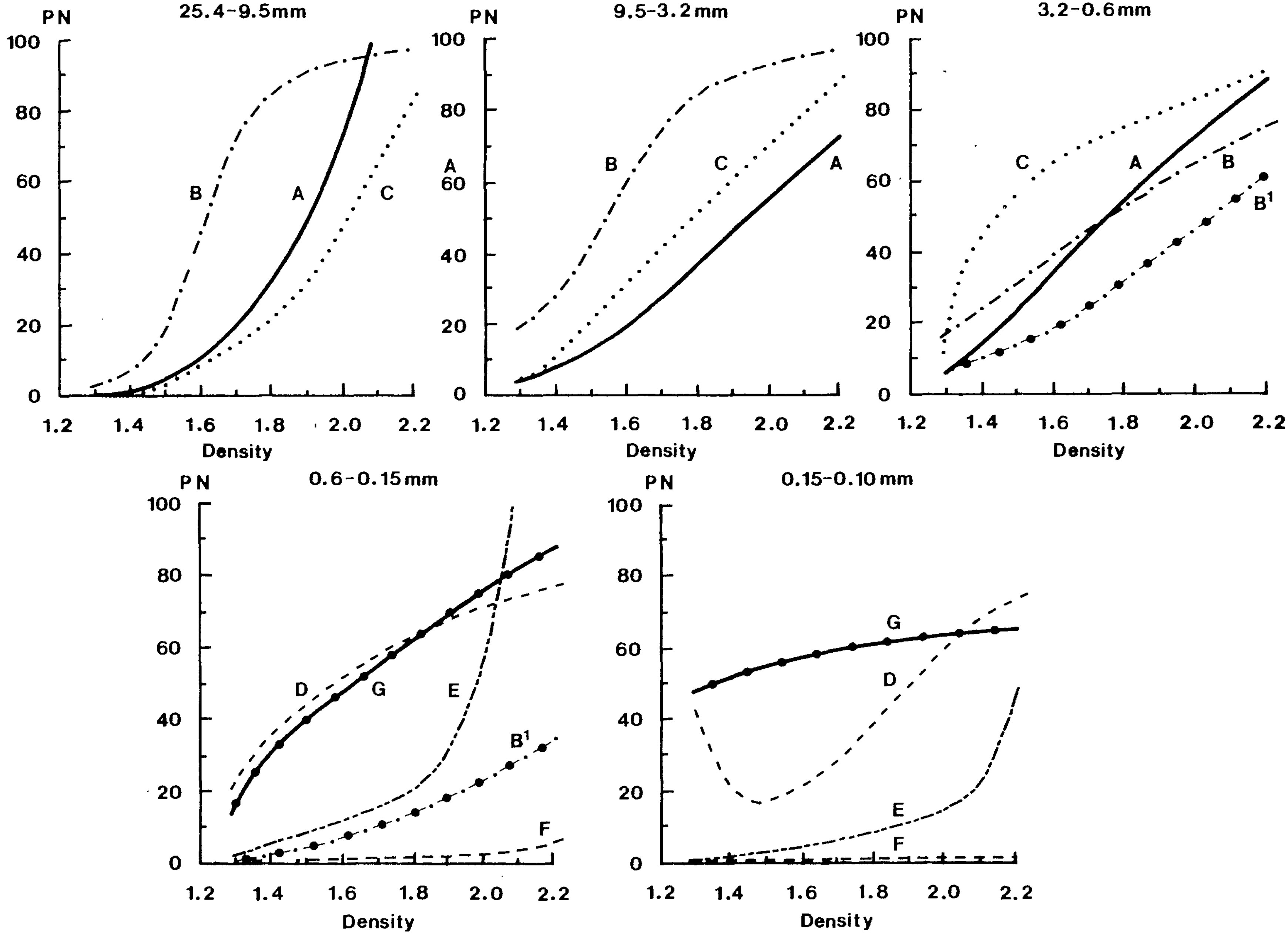

Fig. 30 – Partition curves for hydrocyclone washing of individual size fractions

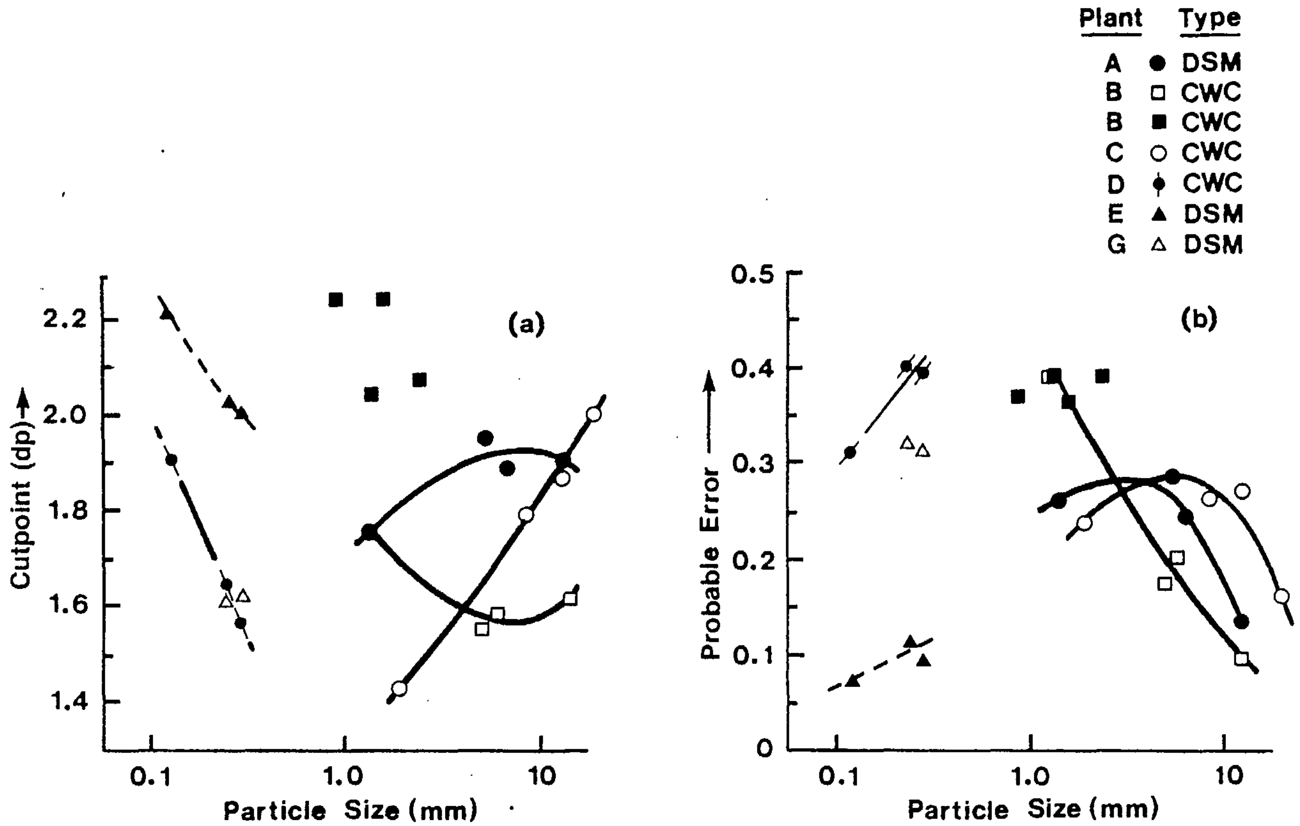

Fig. 31 – Variation of hydrocyclone cutpoint and probable error with particle size

Effect of Cyclone Geometry and Feed Characteristics

While the available details on cyclone geometry are somewhat spotty and limited in range, the data nevertheless showed that the best separation sharpness occurred for the cyclone with the highest d_i/d_c ratio (0.36) and the poorest for those cyclones with $d_i/d_c < 0.23$. At least for the CWC, separation sharpness also appeared to be best when the vortex finder clearance was high. It was found through regression analysis that much of the variation in Fig. 31 could be explained by taking into account the differences in geometry and feed characteristics for both cyclone types together. The analysis also showed there were possible differences in effects of the geometry and in relative importance of the feed characteristics between the CWC and DSM cyclones (Fig. 32).

The relationships obtained for the DSM and CWC (correlation coefficients, r = 0.9912 for n = 9 and r = 0.9011 for n = 15 respectively),

$$R_{DSM} = 0.608 - 1.449\, d_i/d_c - 0.0016D^2 + 0.013D \qquad \text{Eq 5}$$

$$R_{CWC} = 1.222 - 0.218(S18)^{0.5} + 0.018(S18) - 0.043(d_c)^{0.5} - 0.00039D^2 \qquad \text{Eq 6}$$

suggest that within the range of observed operating conditions, DSM performance was influenced principally by the ratio d_i/d_c and the particle size (D) whereas CWC performance was influenced by feed refuse content (S18), cyclone diameter (d_c) and particle size. For the DSM cyclone, probable error rose to a maximum at a particle size of 4 mm and was generally lowest for the highest d_i/d_c ratio (eq 5, Fig. 32a). For the CWC, on the other hand, probable error fell to a minimum at a refuse content of 37.5% and was generally lowest for the largest particle size and cyclone diameter (eq 6, Fig. 32b). An optimum range of feed refuse content for CWC operation is consistent with a minimum requirement below which formation of a fluid bed in the conical section of this cyclone may not occur (15) and with an upper limit beyond which crowding of the apex region and/or increase in viscosity would impair performance. As noted earlier, 40% refuse is the generally accepted upper limit for hydrocyclone feeds.

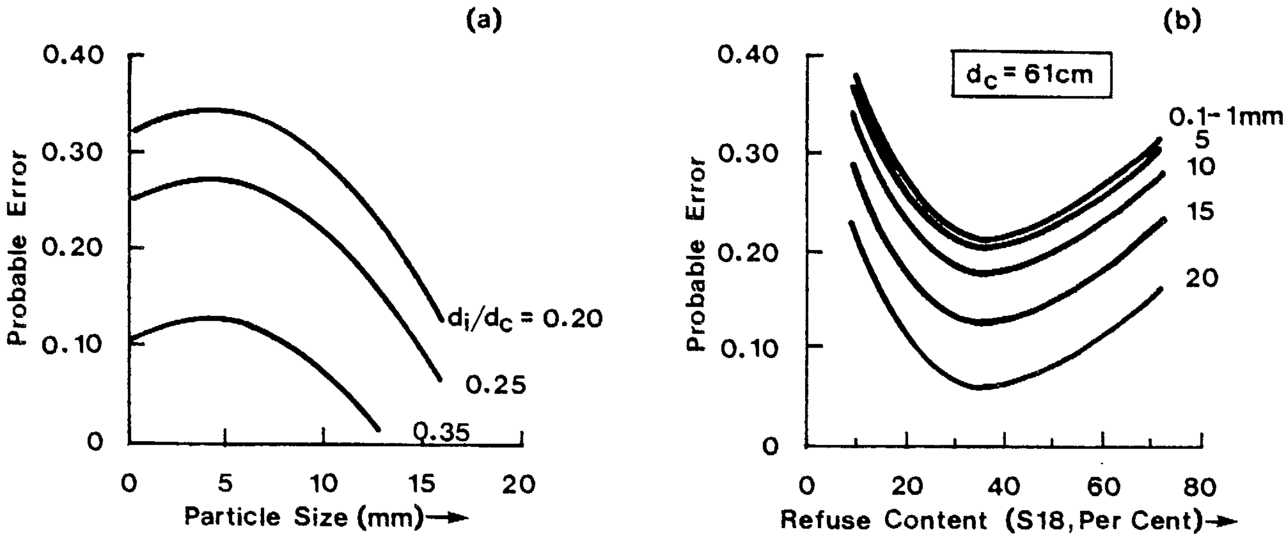

Fig. 32 – Effect of geometry and feed characteristics on probable error for a) DSM and b) CWC hydrocyclones

Separation Losses

Considering the full range of particle sizes (22 – 0.12 mm) for all plants, recovery efficiencies in these primary hydrocyclones varied between 99.7 and 42.5% with corresponding yield losses of between 0.3 and 50.9%. Regression analysis showed that the yield error increased as separation sharpness and particle size decreased and that it was greatest in the lower cutpoint range (Fig. 33a).

Product loss was directly proportional to the yield error, increased with decreasing particle size and attained a minimum at 61% feed refuse content (Fig.23b). To a large extent, the loss of product was directly attributable to misplacement of floats in the refuse. In turn, the floats in refuse showed a high positive correlation with the theoretical yield of clean coal, were generally greater in the larger diameter cyclones and lower in those cyclones operating at higher inlet pressures. The primary rejects contained between 9.3 (Plant C) and 61.9% (Plant D) clean coal product with ash contents of 7.7-32.7% (Table 17). On the total feed basis, the losses varied in the range of 0.28% (Plant F) to 17.45% (Plant G).

From the total installed hydrocyclone capacity in each of the plants (50-445 tph), the clean coal loss ranged between 0.3 and 38.0 tph and totalled an estimated 96.62 tph. This tonnage represented 23.86% of an estimated 405 tph of primary reject produced of which 361 tph containing 25.67% clean coal was rewashed. Thus, there were 92.66 tph of potentially recoverable coal. From prediction calculations, it was estimated that approximately 25.94 tph of product quality coal or 28% could have been recovered with secondary washing. On this basis, an overall balance of 70.68 tph would have amounted to an overall average loss of 5.3% on the raw coal basis or approximately 7.2% of clean coal output with 2-stage washing.

Table 17 – Estimated clean coal losses in hydrocyclone primary rejects

	Size fraction (mm)					Total
	19.0-9.5	9.5-3.17	3.17-0.60	0.60-0.15	0.15-0.10	
Plant A						
Product ash (%)	10.2	11	9.8	—	—	10.6
Wt in refuse (%)	24.1	59.8	56.5	—	—	57.2
% of total feed	0.09	3.09	1.18	—	—	4.36
Plant B						
Product ash (%)	9.3	8.0	7.1	—	—	7.7
Wt in refuse (%)	2.9	29.0	54.0	—	—	24.2
% of total feed	0.30	3.29	3.00	—	—	6.59
Plant B'						
Product ash (%)	—	8.7	14.0	16.5	—	14.1
Wt in refuse (%)	—	26.5	40.0	10.8	—	28.8
% of total feed	—	0.08	3.00	0.50	—	3.58
Plant C						
Product ash (%)	19.6	15.2	16.0	—	—	15.9
Wt in refuse (%)	0.8	10.6	30.8	—	—	9.3
% of total feed	0.27	1.98	3.93	—	—	6.18
Plant D						
Product ash (%)	—	—	—	10.0	12.7	10.6
Wt in refuse (%)	—	—	—	60.5	67.7	61.9
% of total feed	—	—	—	10.53	2.69	13.22
Plant E						
Product ash (%)	—	—	—	10.6	12.2	10.8
Wt in refuse (%)	—	—	—	54.1	38.5	52.6
% of total feed	—	—	—	1.23	0.09	1.32
Plant F						
Product ash (%)	—	—	—	12.8	24.23	15.2
Wt in refuse (%)	—	—	—	18.3	>90.0	25.5
% of total feed	—	—	—	0.22	0.06	0.28
Plant G						
Product ash (%)	—	—	—	23.7	79.4	32.7
Wt in refuse (%)	—	—	—	37.5	43.8	38.4
% of total feed	—	—	—	14.63	2.82	17.45

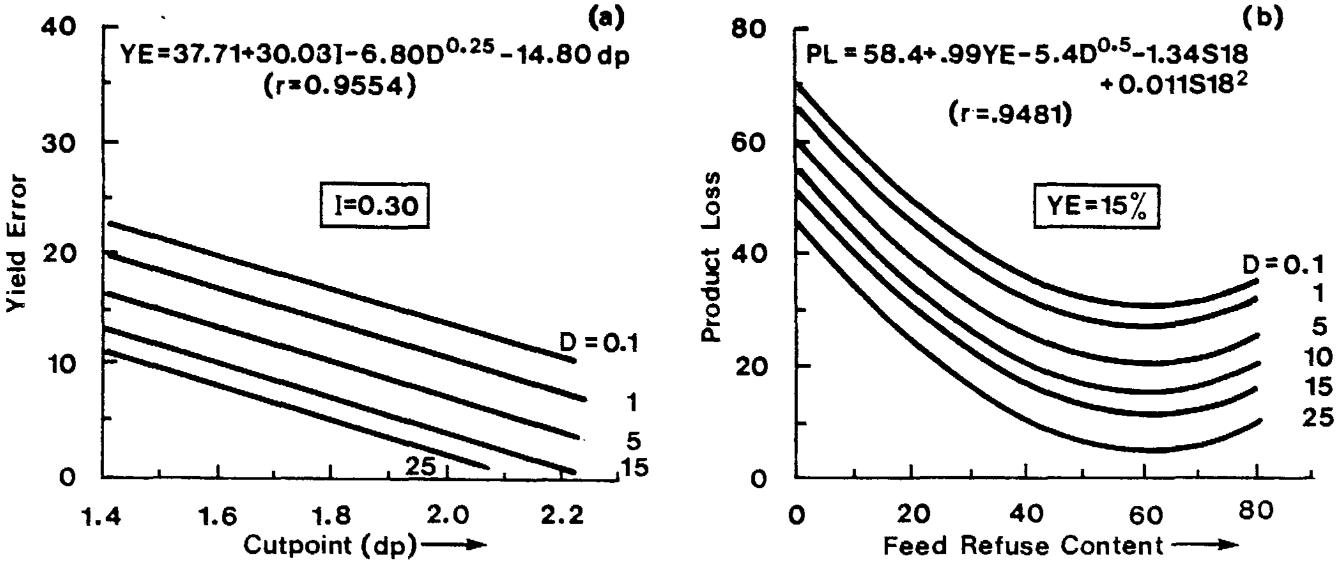

Fig. 33 – Recovery and product losses for primary hydrocyclones

SUMMARY

1. The hydrocyclones treated coals with top sizes of 34-1.2 mm and ash contents of 12-62%, producing clean coals containing 7.8-31.8% ash at cutpoints in the range 1.59 to 2.25 relative density and with yields of 31.5-98% and organic efficiencies of 53.3-99.0% in the primary stage.
2. The high misplacement of floats in primary refuse, averaging 44.9% in the range of 9.1 to 86.2%, underscored the importance of providing second-stage washing for this product.
3. Overall separation sharpness increased as the ratio of inlet to cyclone diameter (d_i/d_c) increased and was better than average for Plant E with $d_i/d_c = 0.36$, near to average for Plants A, B, B' and C with $d_i/d_c = 0.25$ and only fair for Plants D, F and G with $d_i/d_c = 0.20\text{-}0.23$.
4. Multiple regression analysis indicated that while sharpest separation was generally achieved, as expected, for larger particle sizes, best accuracy in the DSM cyclone might be expected to occur for higher d_i/d_c ratios and best accuracy in the CWC might be expected for larger diameter cyclones and for feeds with refuse contents in an optimum range of approximately 30-50%.
5. All plants considered, yield error varied between 0.3 and 50.9% and increased as particle size, cutpoint and separation sharpness decreased.
6. Product loss varied between 9.3 and 61.9%, was directly proportional to the theoretical yield of clean coal but appeared to be lower in cyclones operated at higher inlet pressures and higher in larger diameter cyclones, notwithstanding these provided sharper separation.
7. Total product loss for all plants was at the rate of 96.62 tph of which an estimated 25.94 tph could have been recovered, leaving an estimated total net loss amounting to 5.3% of the feed or 7.2% of 2-stage clean coal output.

WET CONCENTRATING TABLE

The use of wet tables for coal preparation began as long ago as 1893 in the U.S., although it was not until 1910 that riffles were introduced and until 1918 that reciprocal motion tables similar to the modern-day version came into being. With the possible exception of Australia, tables, historically have not found wide application outside the U.S. where they seem to remain especially favoured for fine coal washing. First use of the wet table in Canada did not occur until the late 1960's and then, as now, in only one plant where it accounts for less than 2% of all washed tonnage.

Separation on the table is generally ascribed to the combined action of stratification and hindered settling processes that arise from the differential motion of the table. This motion, assisted by the deck surface, the riffles and wash (dressing) water causes particles of different density and size to travel along different paths to discharge at different points along the edges of the table. The process is very low cost, almost maintenance free, flexible, easy to control and has minimal water requirements. It is applicable to coals of all ranks in the fine size ranges below 10 mm and has proven to be especially effective for removal of flat particles and of free pyrite down to 16 ᵉm (21). However, the table is not considered appropriate for highly variable coals or for those with high middlings or refuse contents. Table performance tends to be either very efficient when properly adjusted or very poor (except for easy separations) when adjustment is lacking and, therefore, benefits from the attention of a relatively skilled operator.

PROCESS CHARACTERISTICS

The main prerequisite to good table operation is a combination of even feed rate with no major variation in the size or density distribution of the coal, a relatively constant water-to-solids ratio, and provision of the proper quantity and distribution of dressing water (preferably clean). Considering the deck material, riffling and end slope as being given for a certain coal, the main factors that may potentially require adjustment during operation include the stroke length and speed, the dilution and dressing water and the cross slope. Although care is required to maintain a proper balance between the settings, adjustment is usually simple because the results can be immediately observed on the deck.

To achieve good feed distribution and the bed mobility that is required for effective separation, the cross slope is usually kept as low as possible, the ratio of feed dilution water-feed solids should remain at approximately 2:1 and the dressing water at 20-30% of the dilution water. As a rule, the coarser the coal, the slower the speed and the longer the stroke, and the higher the feed rate or refuse content, the higher the speed and the shorter the stroke.

Tables are capable of very high separation efficiency, especially in the plus 0.25 mm size range. Although small losses of very fine coal in the refuse can occur, the tendency is usually better coal recovery than refuse elimination. The most common causes of impaired performance appear to arise from overfeeding, variation in size distribution, especially fines content, and poor adjustment of the dressing water. The data shown in Fig. 34 are taken from an early U.S. study of table operation at 5 eastern washeries in that country (22). Although the maximum cutpoint shown at a particle size, D = 6 mm and minimum probable error at 5 mm would be considered uncharacteristic, factors such as near-density material which have not been taken into account in the equations were found to have a possibly important influence. For present purposes, only those features that are generally accepted and identified as characteristic of table behaviour will be considered. Thus, for low refuse and low-middlings coals, as particle size decreases in the range of 10 to 0.2 mm:

- cutpoint will decrease slowly to a minimum at between 1 and 2 mm then rise sharply;
- probable error will rise, on average, from 0.07 to 0.16;
- imperfection will rise, on average, from 0.12 to 0.27;
- error area will rise from 42 to 100.

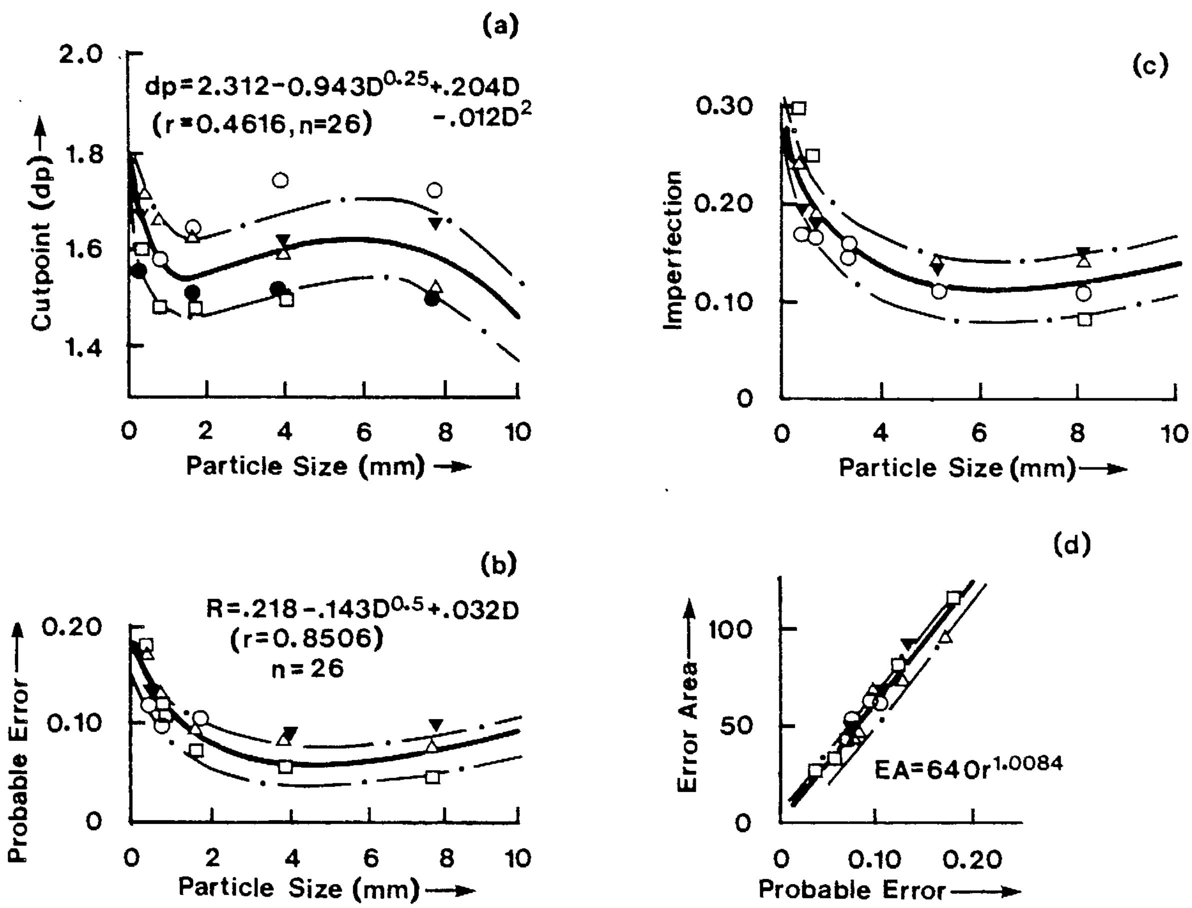

Fig. 34 – Separation characteristics of the concentrating table (22)

PLANT DATA

As noted previously, there was but a single washery using tables during the period 1978-80 in Canada. A total installed capacity of 145 tph was obtained through 16 tables arranged in banks of four. The feed was a classified, partially processed coal in the 3-0.10 mm nominal size range comprising approximately 42% of the run-of-mine to the plant. The separation was two-product with no further treatment other than water removal.

The raw coal contained 27% ash and was characterized by more than 20% 0.10 mm undersize and a relatively high middlings content averaging 32% which remained very nearly constant throughout the entire feed size range (Fig. 35 and 36). Refuse content (S18) averaged less than 10% and was highest (12.6%) in the 0.6-0.15 mm fraction.

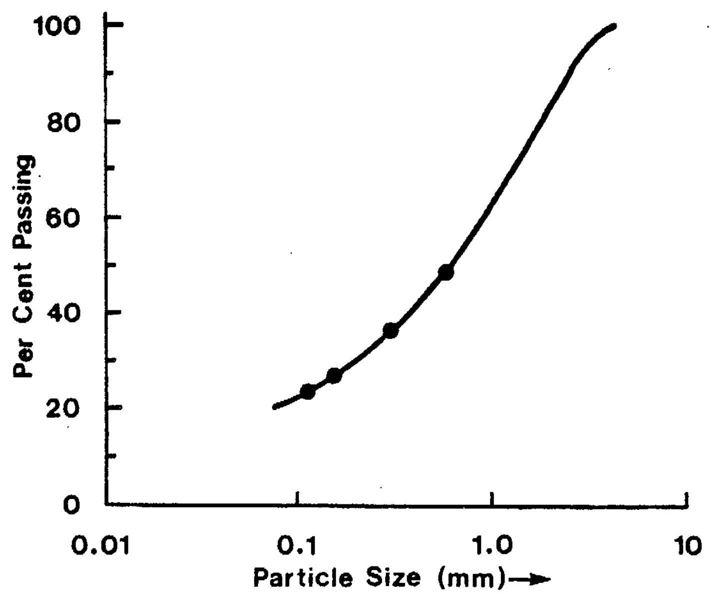

Fig. 35 – Size distribution of reconstituted table feed

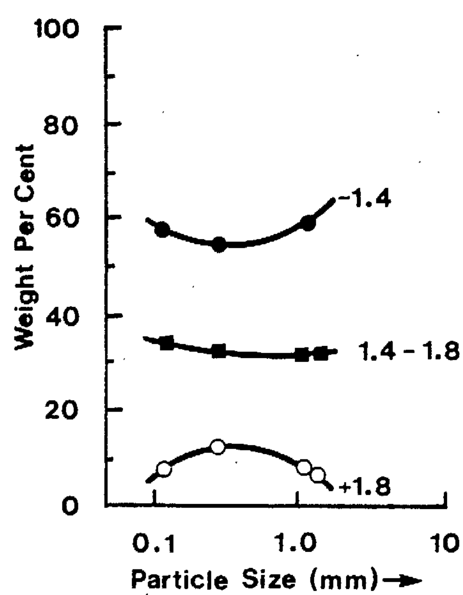

Fig. 36 – Variation of density distribution of table feed with particle size (reconstituted)

TABLE PERFORMANCE

Washing results show that for the overall 3-0.10 mm fraction, a clean coal containing 8.3% ash was obtained at a cutpoint d_p = 1.52 with a yield of 64.2% and organic efficiency of 75.4% (Table 18).

Given a weighted mean particle size of 1.01 mm for the plus 0.10 mm feed, reference to Fig. 34b indicates that the probable error, r = 0.146 cannot be considered consistent with good table performance. Similarly, reference to Fig. 34d shows that for this probable error, an error area of 109 was somewhat higher than average.

The partition curves show that the sharpest separation was achieved in the coarsest fraction but that refuse elimination was relatively poor and coal recovery only fair (Fig. 37). As particle size decreased, coal recovery became progressively worse and refuse elimination was poorest in the middle 0.60-0.15 mm size fraction. The proximity of the high-density end of this middle curve to that for the coarser particle-size curve and of the low-density end to that for the finer particle-size curve suggests some degree of mutuality in the manner of separation. This could be interpreted as washery 76 that the higher density material in the middle fraction tended to occur in the region of the upper size limit and that the low-density material was more or less confined to the region of the lower size limit. Although the curves, overall, suggest there may have been lack of bed mobility, there are insufficient data to produce a clear picture.

Table 18 – Summary of washing results for concentrating tables (1978-80)

	Size fraction (mm)				
	3.18-0.6	0.6-0.15	0.15-0.10	3.18-0.10	0.6-0.10
Ash content (%)					
Raw coal	13.4	19.9	25.3	15.2	20.2
Reconstituted coal	13.6	17.1	19.5	14.9	17.4
Clean coal	8.3	7.4	10.2	8.3	8.0
Refuse	27.2	26.4	27.0	26.7	26.2
Yield of clean coal (%)	71.8	49.0	44.4	64.2	48.6
Theoretical yield (%)	87.4	79.5	78.0	85.2	80.1
Organic efficiency (%)	82.2	61.6	56.9	75.4	60.7
Separation density (d_p)	1.560	1.452	1.448	1.515	1.450
Probable error	0.129	—	—	0.146	—
Error area	104	(119)	(106)	109	(118)
Imperfection	0.230	—	—	0.284	—
±0.10 Near density material(%)	15.0	33.0	38.5	22.5	35.0
Floats in refuse (%)	60.8	43.4	52.0	54.0	43.4
Sinks in clean coal (%)	4.1	14.5	12.8	7.2	14.4
Total misplaced material (%)	20.1	29.2	34.6	24.0	29.3
Yield error (%)	15.6	30.5	33.6	21.0	31.5
Ash error (%)	2.6	3.5	5.8	3.2	4.0

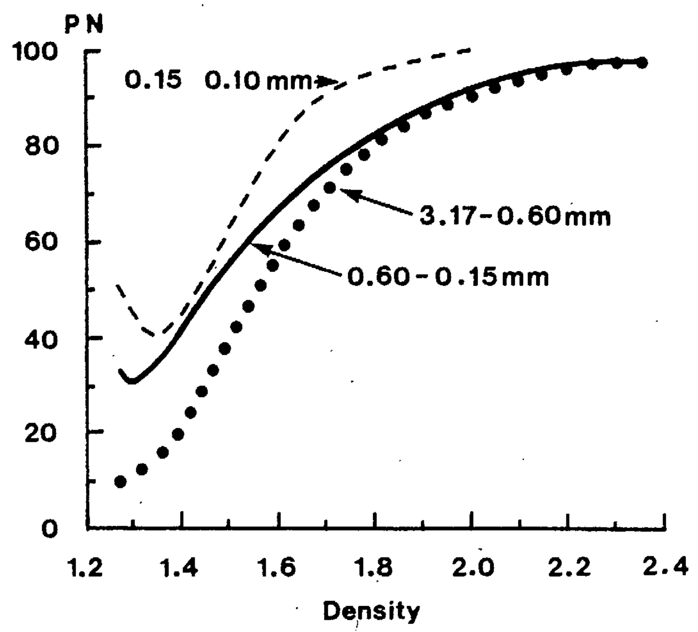

Fig. 37 – Partition curves for table separation of individual size fractions

Effect of Feed Characteristics

As the mean particle size decreased from approximately 1.4 to 0.12 mm, the cutpoint decreased, at first rapidly then very slowly (Fig. 38). On the basis of behaviour shown in Fig. 34a, a rapid rise in cutpoint would normally have been expected in this size range. The data provided no insight into the apparently uncharacteristic drop in cutpoint. The maximum error area at 0.3 mm coincided precisely with the maximum feed refuse content (S18). As shown in Fig. 39 there was a very high correlation between the two. No other feed variable was found to be significant except the percentage floats at 1.35 which showed an inverse relationship with error area.

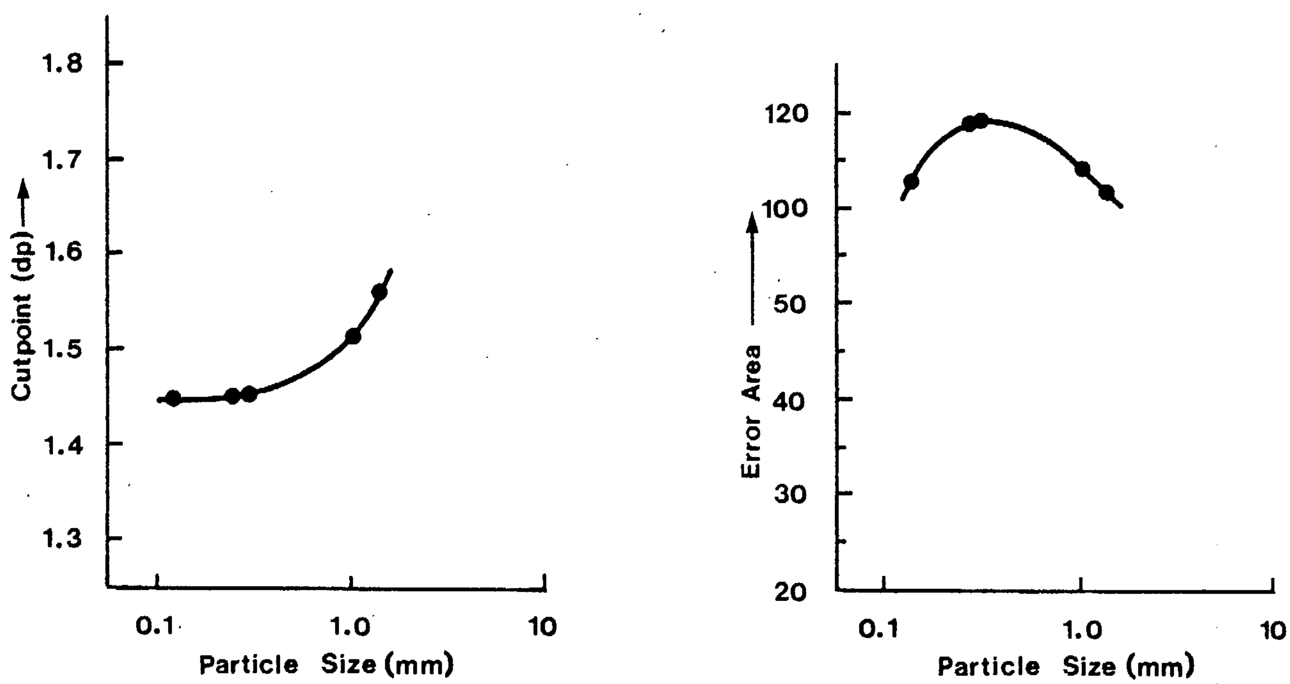

Fig. 38 – Variation of cutpoint and error area with particle size for wet table separation of 3-0.10 mm coal

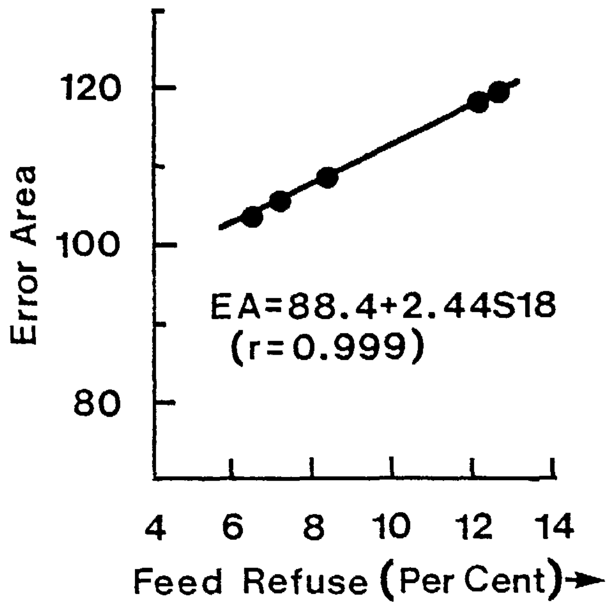

Fig. 39 – Relationship between error area and the refuse content in table feed

Separation Losses

Recovery efficiency was a direct function of mean particle size and decreased linearly from 82.2% for the largest particles (D = 1.4 mm) to 56.9% for the smallest particles (D = 0.12 mm). The absolute losses given by the yield error correspondingly increased from 15.6 to 33.6%. It was found that the yield error correlated highly with both the ±0.10 near-density material (r = 0.998) and the particle size (r = −0.999). Since it was also found that they were themselves closely related (r = −0.995), the relative importance of each of these factors must in this case remain unclear. On the other hand, the increase in product loss (Table 19) from 48.5 to 61.3% was attributable primarily to the increase that occurred in near-gravity material coincidently with decrease in particle size. The relationship (r = .9971),

$$PL = 23.48 + 0.55(NG) + 0.36(F135) \qquad \text{Eq 7}$$

indicates that the product loss was controlled by the ±0.10 near-density material and percentage floats at 1.35 in the feed. The total clean coal loss which amounted to an estimated 14.26% of the feed represents a very significant percentage of the overall table throughput capacity. On the basis of 145 tph, the product loss rate equalled almost 21 tph or approximately 22% of clean coal output.

Table 19 – Estimated clean coal losses in table refuse (1978-80)

	Size fraction (mm)			
	3.18-0.60	0.60-0.15	0.15-0.10	Total
Product ash (%)	8.3	7.4	10.2	8.1
Wt in refuse (%)	48.5	56.5	61.3	52.5
% of total feed	6.96	6.45	0.85	14.26

SUMMARY

1. The 145 tph table plant treated a high-middlings 3-0.10 mm feed containing 15% ash, and produced a clean coal with 8.3% ash at a cutpoint of 1.515 with a yield of 64.2% and 75.4% organic efficiency.
2. The overall probable error, r = 0.146 and error area = 109 indicate that separation sharpness was poorer than that typically achieved in the U.S. for low-refuse, low-middlings coals.
3. Partition curves for the individual size fractions showed that coal recovery was uncharacteristically poorer than refuse elimination and that it became progressively worse as particle size decreased.
4. Cutpoint decreased continuously as particle size decreased, but the error area reached a maximum at 0.3 mm and was highly correlated with the feed refuse content.
5. An increase in yield error from 15.6 to 33.6%, corresponding to a drop in organic efficiency of from 82.2 to 56.9%, was closely related to a decrease in mean particle size from 1.4 to 0.12 mm but also to an accompanying increase of from 15 to 38% ±0.10 near-density material in the feed.
6. Losses of 8.3% ash clean coal were estimated at 14.3% of the feed, corresponding to 21 tph or approximately 22% of the clean coal production.

FROTH FLOTATION

Froth flotation as it is known today was patented in England in 1906, although reference to a similar process is said to occur in 15th-century Persian literature (23). Its use in coal preparation was pioneered by the Dutch State Mines at the end of World War I and which, by 1925, had raised clean coal production to approximately 3800 tpd in three washeries. The process was soon adopted in Europe but high operating costs prevented general acceptance in North America until after the 1950's. The economic turning point followed from the significant increase in fine coal production that was occasioned by increased use of mechanized mining techniques. In more recent years, environmental and resource management considerations have added to the economic and sometimes technical imperative of treating the fine sizes. In 1952, flotation was limited to a single plant in Canada. Since the 1970's, however, it has become relatively commonplace and today the process accounts for approximately 14% of all washed tonnage.

Froth flotation depends primarily on differences in the surface chemical and physical properties of particles, whereby the imparted differences in their relative affinities for water distinguish the more desirable from the less desirable constituent particles in a mixture. The process of separation is accomplished through the agency of air bubbles that are introduced into the slurry in flotation cells. These bubbles provide not only the means for discrimination and collection because of preferential attachment to essentially hydrophobic coal particles but also, because of their natural buoyancy, the mode of transport for these particles to discharge in the froth at the top of the cell. Given the proper conditions, flotation can be used for coals of all ranks but in practice it is used almost exclusively for bituminous metallurgical and thermal coals. The process currently provides the only means for maximizing recovery of coals in the minus 0.5 mm particle size range where density separation may be only marginally effective or not at all. Flotation has the advantage of low capital and power costs and permits a great deal of flexibility in flowsheet design. On the other hand, it has the disadvantage of low capacity, high reagent costs, susceptibility to fluctuations in the feed and that it is difficult to control. The result of excessive reagent consumption has generally limited its use for oxidized coals or for feeds with exceptionally high slimes content where product contamination may be significant. Selectivity varies according to the degree of liberation but is not usually considered good. Achieving and maintaining optimum performance can be demanding and an experienced operator is an advantage.

PROCESS CHARACTERISTICS

The difficulty with controlling flotation can be appreciated from the fact that it is a rate-dependent process where even under optimum conditions, the rates of recovery of various particles in the feed tend to differ significantly depending on their size, density, petrographic composition and surface condition. In normal operation, this problem can be aggravated if wide variation in feed quality, pulp density, water ionic composition or pH exists. The effects of feed variability can, to some extent, be minimized through skillful use of reagents and optimization of operating variables. Much depends on the initial engineering decisions on machine capacity, flowsheet design and to some degree on machine selection which can be effectively reduced to a consideration of the relative merits of cell-to-cell and open-flow-thru tank designs (24).

Optimization is generally achieved by adjusting the rates of aeration, agitation and reagent addition, controlling pulp level, providing adequate conditioning time and through the use of special techniques. These techniques include multi-stage flotation, froth sprinkling, variations using fast flotation or starvation feeding, collector emulsification, pH adjustment and/or feed pretreatment using classifiers, hydrocyclones or wet tables (25,26,27,28). As a rule, only two reagents, those classed as collectors (e.g., kerosene and diesel oil) and frothers (e.g., Methyl isobutyl carbinol [MIBC] and pine oil) are needed. However, in cases involving pyritic or oxidized coals, pH adjustment, addition of activators and the use of special techniques may also be necessary (27,29,30).

PLANT DATA

The installed capacity of froth flotation machines totalled 765 tph in five washeries where all applications were for low- to high-volatile bituminous metallurgical coals. The feeds had nominal top sizes of between 0.6 and 0.2 mm and originated as the undersize of 0.6 mm deslime screens preceding heavy-medium cyclone operations. Three of the plants provided feed pretreatment using hydrocyclones and screening or cyclone classification ahead of the flotation cells. The feeds accounted for between 12 and 30% and averaged 22% of the washery run-of-mine. Pulp solids content ranged between 6 and 9% and retention times between 2 and 6 minutes. Either diesel oil or kerosene collector was used and in all plants but one the frother was MIBC.

Table 20 shows that raw feed ash contents varied between 14 and 32%, contained up to 14% plus 0.6 mm oversize and 27-64% minus 0.1 mm fines (Fig. 40). The density distributions show that the feeds were generally characterized by high proportions of low-density coal, low refuse contents and, except for Plants B and D, 15% or less middlings contents (Fig. 41).

Table 20 – Feed characteristics for froth flotation plants

	Raw feed			Density distribution *				Theoretical *			
Plant	Nominal size range (mm)	Oversize (%)	Minus 0.1 mm (%)	Ash (%)	Floats @ 1.4 (%)	1.4-1.8 (%)	Sinks @ 1.8 (%)	Product ash (%)	Cutpoint (dp)	Product yield (%)	Reject ash (%)
A	0.6-0 ($\bar{D}$=0.183)	0	46.6	20.6	81	4	15	7.7	>2.3	88	79
B	1.0-0 ($\bar{D}$=0.086)	0.4	64.0	15.4	58	27	15	12.3	>2.2	>99	~80
C	1.7-0 ($\bar{D}$=0.332)	13.7	26.6	16.8	77	15	8	9.4	>2.3	95	78
D	0.6-0 ($\bar{D}$=0.176)	5.4	51.2	13.6	69	22	9	8.1	>2.2	98	56
E	1.2-0 ($\bar{D}$=0.147)	2.1	46.3	32.2	91	7	2	6.8	1.42	96	42

* *From reconstituted feed washability data, plus 0.10 mm*

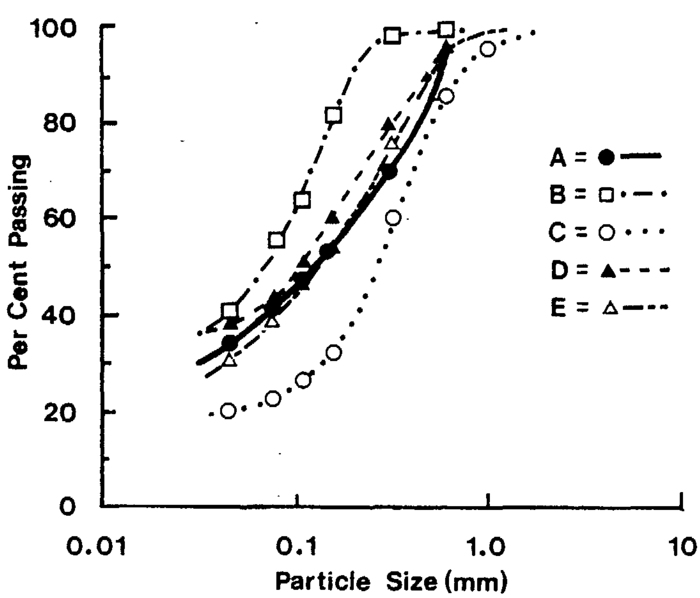

Fig. 40 – Size distribution of froth flotation feeds (reconstituted)

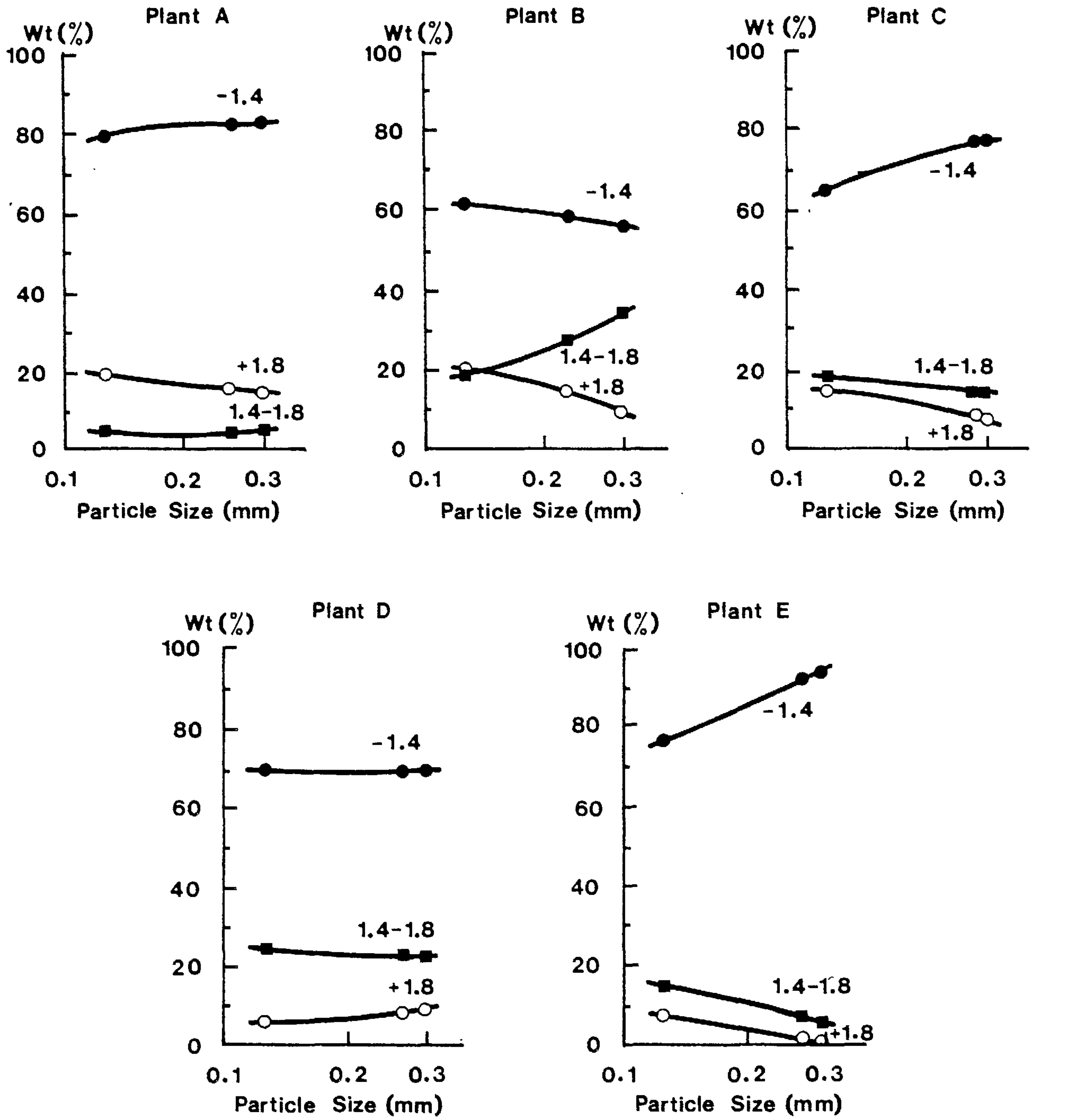

Fig. 41 – Variation of density distribution of froth flotation feeds with particle size (reconstituted)

FLOTATION PERFORMANCE

Flotation results for the total coals showed that clean coal products containing between 6.8 and 12.3% ash were obtained at yields of 48 to 92% with tailings ash contents of 18.6-65.9%. These results compare with those for the 0.6-0.1 mm fractions which show clean coal ash contents between 4.9 and 12.9% obtained at yields of 39.2 to 89.6°% and tailings ash contents of 9.5-45.0% (Table 21). The differences between the total and Table 21 results can be accounted for by the contribution of the minus 0.1 mm fines to the total ash contents of both products. These fines, which constituted between 35 and 65% of the froth and between 43 and 76% of the tailings for the individual plants, on the average contributed 63% of the total ash in the froth and 76% ± 9.6% of the total ash in the tailings. The differences between ash contents of the fines in froth and tailings shown in Table 22 nevertheless indicate that there was good differentiation between the low- and high-ash constituents-hence significant cleaning in the minus 0.1 mm size range. The mean particle sizes of the products show, furthermore, that except to a slight extent in Plants B and D, classification was not a prominent factor in the separation process.

The organic efficiencies ranged between 90% for Plant B and 39.9% for Plant D (Table 21). However, taking the difference between the reconstituted feed and clean coal ash contents, it is found that a reduction of only 0.4% ash was achieved in Plant B and of only 0.8% ash in Plant D and thus, that there was relatively little cleaning in these separations. From the yield errors of 9.9 and 59%, recovery losses per unit ash reduction amounted to 24.8 and 73.8 for Plants B and D respectively. These unit losses compare poorly with the value of 1.1 observed in Plant A, 4.6 in Plant C and 12.1 in Plant E.

Partition curves based on the conventional plots used for density separations are shown in Fig. 42 for the 0.6-0.15 mm and 0.15-0.10 mm size fractions of each plant. The trend of increasing partition number with increasing density in all of the plots bears out that at least in these size ranges, particle density is a significant factor among those others that have a direct bearing on the flotation separation process. The peak that is evident in the low density range of the plots is sufficiently prominent and consistent a feature to suggest that the effect of density was subject to modulating effects of related factors such as maceral composition or surface oxidation. On the basis that the curves represent separation events occurring within fixed time periods, Fig. 42 also reveals differences in flotation rates between the size fractions and particles of varying densities for each plant (27). For example, given that flotation rate is inversely proportional to the partition number, the plots show that although there was significant variation over the entire density range, the fine particles tended to float more quickly than the coarser particles in Plants A, B and D but more slowly in Plants C and E. This is confirmed by the respective yields of the size fractions. The plots indicate, moreover, that flotation rate tended to decrease to a minimum in the 1.35-1.55 density range, to rise to a maximum in the 1.45-1.90 density range then to drop again with continuing increase in particle density. Thus, the ability of the process to discriminate between particles of differing density was variable between and within plants but was generally better when the differential was great. Fig. 42 could also be interpreted as indicating that, for whatever reason, the flotation rate may have been either too fast or the retention time too long in Plant B and similarly that the flotation rate may have been either too slow or the retention time too short in Plant D to enable either effective particle selection and attachment or completion of the transport process.

Table 21 – Summary of froth flotation results (1978-80)

	Plant				
	A 0.6-0.1 mm	B Plus 0.1 mm	C 0.6-0.1 mm	D Plus 0.1 mm	E 0.6-0.1 mm
Ash content (%)					
Raw coal	10.3	13.0	13.3	11.0	12.1
Reconstituted coal	13.6	13.3	10.1	9.0	6.6
Clean coal (froth)	4.9	12.9	6.6	8.2	5.2
Refuse (tailings)	45.0	17.7	23.1	9.5	12.2
Yield of clean coal (%)	78.3	89.6	79.3	39.2	79.0
Theoretical yield (%)	88.2	99.3	95.3	98.2	95.9
Organic efficiency (%)	88.8	90.0	83.2	39.9	82.4
Separation density (d_p)	1.508	—	1.700	1.295	1.314
Probable error (R)	0.413	—	0.512	—	0.018
Error area	196	—	238	238	35
Imperfection	0.813	—	0.731	—	0.057
±0.10 Near density material (%)	3.2	—	3.5	68.5	28.7
Floats in refuse (%)	40.5	—	71.0	23.5	55.5
Sinks in clean coal (%)	4.4	—	5.1	59.0	10.6
Total misplaced material (%)	12.2	—	18.7	37.4	20.0
Yield error (%)	9.9	9.9	16.0	59.0	16.9
Ash error (%)	2.7	5.7	3.0	5.3	1.7

Table 22 – Mean particle sizes and fines ash contents of flotation froth and tailings

Plant	Froth		Tailings	
	Average particle size (mm)	Ash in −0.1 mm (%)	Average particle size (mm)	Ash in −0.1 mm (%)
A	0.17	10.95	0.18	88.9
B	0.08	11.97	0.12	76.8
C	0.33	14.72	0.34	46.3
D	0.13	8.03	0.23	30.5
E	0.18	7.90	0.10	25.6

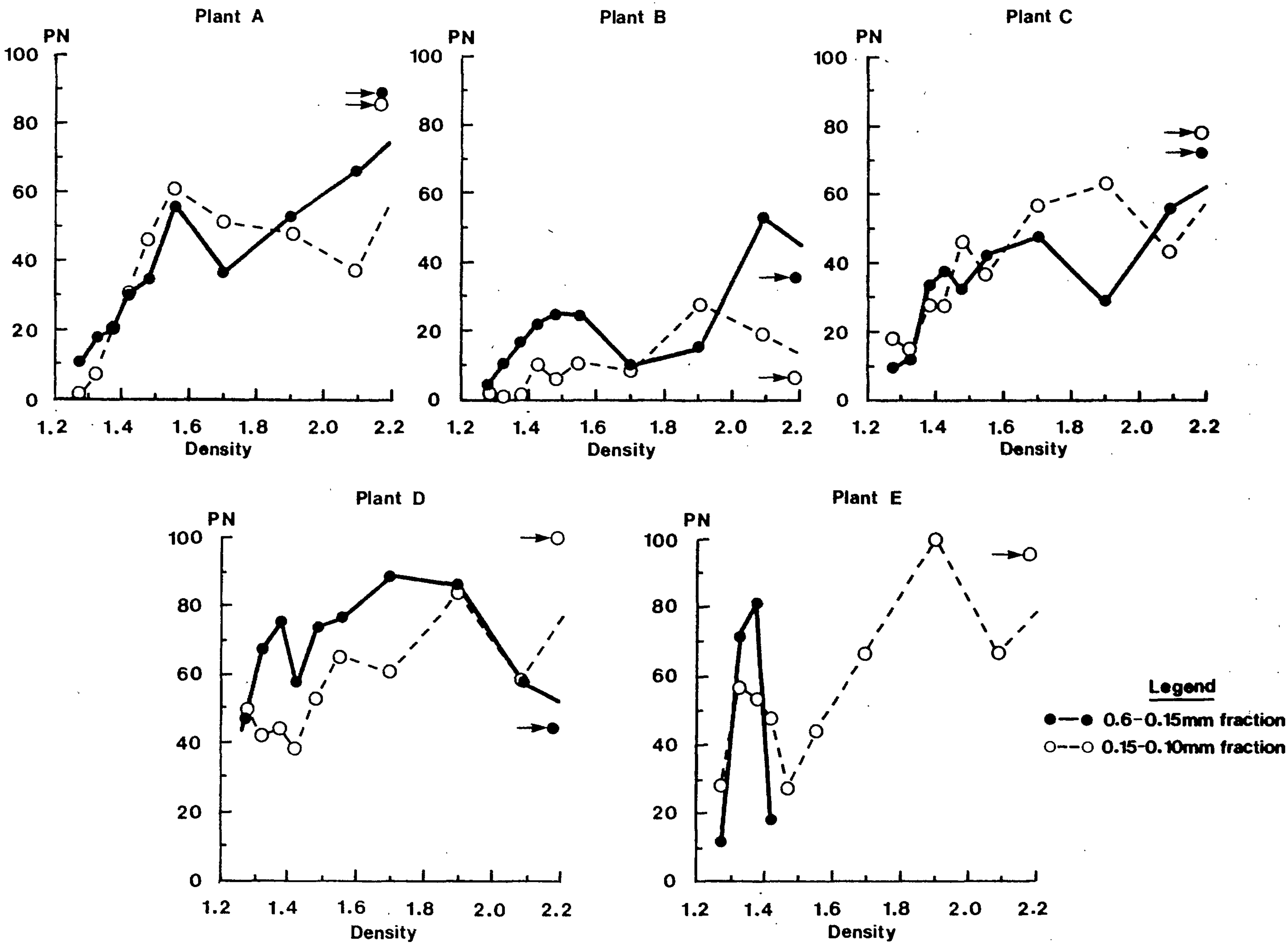

Fig. 42 – Partition curves for flotation separation of 0.6-0.15 mm and 0.15-0.10 mm size fractions

Effect of Feed Characteristics

Full assessment of the impact of feed characteristics on the individual separations requires further study and more details on the feed and on the operating variables than were available. However, the feeds to Plants B and D, which both showed the least degree of cleaning and the possibility of classification effects during separation, were distinguishable from the other feeds by the fact that they also contained the highest middlings and highest minus 0.1 mm fines contents (Table 20).

Separation Losses

For the 0.6-0.1 mm size range, recovery efficiency in all plants varied between 97.0 and 36.9% with yield losses ranging from 3.6 to 62.9%. These losses were directly proportional to the amount of ±0.10 near-density in the feed and decreased as the percentage floats at 1.4 increased (Fig. 43).

Table 23 shows that product loss ranged between 13.9 and 87.6% for the two size fractions and that, overall, it was smallest in Plant B (23.6%) and greatest in Plant C (63.2%). As percentages of the feed, the losses generally tended to be substantially higher in the plus 0.15 mm than in the 0.15-0.10 mm fraction. Based on the installed capacities of the individual plants, the total product loss amounted to an estimated 56.98 tph of which 82% or 47 tph was in the 0.6-0.15 mm size range. On the average the total loss constituted 27.7% of the tailings, 7.45% of the feed and 10.19% of flotation clean coal output.

Table 23 – Estimated clean coal losses in froth flotation plants

	Size fraction (mm)		
	0.6-0.15	0.15-0.10	Total
Plant A			
Product ash (%)	4.7	6.0	4.9
Wt in refuse (%)	47.6	62.0	49.4
% of total feed	4.85	0.88	5.73
Plant B			
Product ash (%)	8.5*	17.0	12.9
Wt in refuse (%)	13.9*	65.7	23.6
% of total feed	0.42*	0.46	0.88
Plant C			
Product ash (%)	6.4	9.7	6.6
Wt in refuse (%)	66.7	38.0	63.2
% of total feed	7.22	0.57	7.79
Plant D			
Product ash (%)	8.3*	8.1	8.2
Wt in refuse (%)	50.6*	87.6	56.1
% of total feed	12.84*	3.81	16.65
Plant E			
Product ash (%)	5.0	6.2	5.2
Wt in refuse (%)	77.8	26.7	61.0
% of total feed	5.66	0.96	6.62

** Plus 0.15mm*

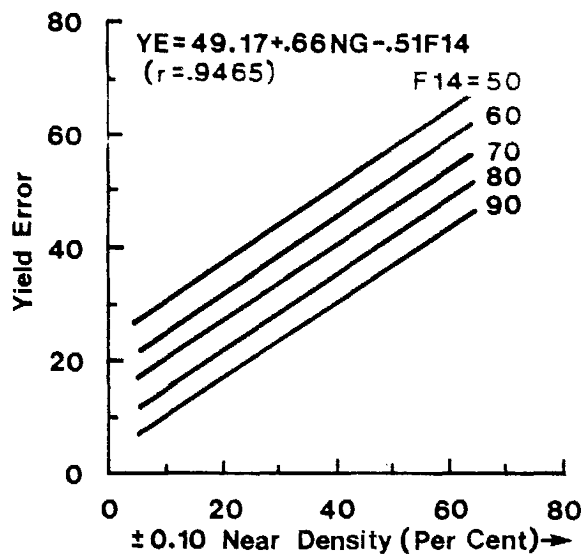

Fig. 43 – Recovery losses for froth flotation plants

SUMMARY

1. The flotation process treated feeds with a nominal top size of 0.6 mm and ash contents of 14-32%, producing clean coals containing 6.8-12.3% ash with yields of 48-92%.
2. Clean coal ash contents of 4.9-12.9% were achieved at yields of 39.2-89.6% and organic efficiencies of 39.9-90% for the 0.6-0.1 mm size range.
3. A significant degree of cleaning was achieved in the minus 0.10 mm size fraction for all plants.
4. Although density generally plays an important role in the separation process, its influence appears to be subject to effects of related factors such as maceral composition or surface oxidation.
5. The flotation rate of the 0.15-0.10 mm size fraction was faster than that of the 0.6-0.15 mm size fraction in Plants A, B and D but slower in Plants C and E.
6. Flotation rates were highest, on the average, for particles <1.35 and 1.45-1.90 relative density and lowest for particles 1.35-1.55 and *S65*1.90 relative density.
7. Minimal cleaning and possible classification effects during separation were found for Plants B and D whose feeds contained the highest percentages of middlings and minus 0.1 mm fines.
8. Yield error for the 0.6-0.1 mm particle sizes varied between 3.6 and 62.9% and was proportional to the ±0.10 near-density but inversely proportional to the percentage floats at 1.4 in the feed.
9. Product losses for the 5 flotation plants varied between 13.9 and 87.6%, totalling between 23.6% (Plant B) and 63.2% (Plant C) and were substantially higher in the coarser of the two size fractions.
10. The combined estimated product loss of 56.98 tph for all plants corresponded on the average to 27.7% of the tailings, 7.45% of the raw feed or 10.19% of flotation clean coal output.

GENERAL SUMMARY AND CONCLUSIONS

In the period 1978-1980, coal preparation plants in Canada washed 25-32 $\times$ 10^6 tonnes per year of raw bituminous coals containing on the average 23.1% ash. Through the use of coarse coal jigs, heavy-medium vessels and cyclones, hydrocylones, concentrating tables and froth flotation, outputs of 18.6-22.5 $\times$ 10^6 tonnes per year of 9.7% ash clean coal having a value of $43-$49 per tonne on average to the producer were achieved. This was accomplished at an average yield of 73.1% and organic efficiency of 93.4%, with corresponding yield losses of 1.3-1.7 $\times$ 10^6 tonnes annually, all processes and plants considered.

The plant data showed that process implementation in the ten washeries was generally in accordance with conventional practices of the coal preparation industry at large. Of the 26 separate process applications reviewed, only two cases stood out as having been possibly ill-considered. These cases include the use of a jig for a very high refuse feed and of concentrating tables for a high-middlings feed. In the light of known process limitations, both these applications would be viewed as unusual. Results indicated that by comparison with average performance of the individual processes based on reported data from various literature sources, 17 of the 26 applications performed at levels that exceeded or equalled average capability and nine, including the two aforementioned cases, performed at below-average levels.

As expected, overall effectiveness of the coarse coal/heavy-medium processes was far superior as a group to that of the category of fine coal processes (Table 24). The difference in separation capability between these two process categories was clearly demonstrated in the respective losses of saleable coal to the refuse. As shown in Table 25, the combined losses of clean coal in the jigs, heavy-medium vessels and heavy-medium cyclones totalled 19.4 tph, on the average, less than 1% of the feed based on installed capacity of these processes. By comparison, the combined losses for the hydrocyclones, concentrating tables and froth flotation totalled 148.3 tph or an average of 7.3% of the feed. These losses correspond to clean coal percentages in the refuse of 2.4 and 26.6 for the coarse and fine coal processes respectively.

Table 24 – Weighted average results for coarse coal/heavy-medium and fine coal-washing processes (weighting factor = throughput rates based on installed capacity)

	Coarse coal and heavy-medium processes	Fine coal* processes	All processes
Ash content (%)			
Reconstituted coal	24.0	20.6	23.1
Clean coal	9.1	11.5	9.7
Refuse	66.0	43.5	59.7
Yield of clean coal (%)	73.8	71.5	73.2
Theoretical yield (%)	75.3	87.3	78.4
Organic efficiency (%)	97.9	82.0	93.4
Separation density (d_p)	1.562	1.728	1.606
Probable error	0.045	0.284	0.107
Error area	28	154	61
Imperfection	0.080	0.390	0.176
±0.10 Near density material (%)	11.5	10.3	
Floats in refuse (%)	8.1	35.9	15.8
Sinks in clean coal (%)	1.0	8.4	2.9
Total misplaced material (%)	2.9	16.2	6.4
Yield error (%)	1.5	15.8	5.2
Ash error (%)	0.3	5.2	1.6

* *hydrocyclones, concentrating tables and froth flotation*

Table 25 – Summary of clean coal losses in washery reject (1978-80)

Process	Loss rate (tph)	% of refuse	% of feed	% of clean coal
Coarse coal jig	4.00	5.27	1.42	1.95
Heavy-medium vessel	6.76	2.03	0.92	1.68
Heavy-medium cyclone	8.62	1.44	0.35	0.47
Hydrocyclone*	14.20	12.03	4.44	7.03
Concentrating table	20.68	52.50	14.26	22.22
Hydrocyclone	56.48	19.68	5.22	7.10
Froth flotation	56.98	27.68	7.45	10.19

** 25-0.6 mm size range*

While some of the losses might have been avoided or at least reduced through normal plant control practices, the data indicated that more than 85% of the lost tonnage could probably be ascribed to shortcomings in the technology itself. A number of possible gaps in the knowledge and current understanding of the significance of some operating and design variables and of how they affect separator performance were indicated. Such apparent gaps are marked by an asterisk in the summary of process variables in Table 26 which also contains those most often cited in the literature.

Table 26 – Some variables affecting probable error, yield error and product loss in coal-washing processes

	Operational	Raw coal characteristics	Other
Coarse coal jig	Feed rate & variability Pulse rate Pulse amplitude Water volume	Variability Refuse content # ±0.10 near-density # Fines content	Cutpoint #
Heavy-medium vessel	Medium quality and density control Feed rate Feed pretreatment	Floats content* ±0.10 near-density* Fines content #	Cutpoint
Heavy-medium cyclone	Medium quality and density control Feed rate Inlet pressure Feed pretreatment	Top size Fines content #	Cyclone geometry # Cutpoint #
Hydrocyclone	Feed solids content Feed rate Inlet pressure	Refuse content # Middlings content	Cyclone geometry Cutpoint #
Concentrating table	Feed dilution Feed rate & variability Feed distribution Dressing water Stroke speed/amplitude Deck slope, etc.	Variability Middlings content # Refuse content # Floats content* ±0.10 near-density*	Deck material Riffling
Froth flotation	Pulp density Feed rate/retention time Aeration/agitation Feed conditioning Reagent dosages	Variability Surface properties # Degree of liberation ±0.10 near density* Floats content* Slimes content # Middlings content*	Water quality

 # Often cited and confirmed by plant data
** Possibly significant as indicated by plant data*

The first column in the table includes those variables that, with proper adjustment, usually provide the appropriate response(s) for optimization and control of separator performance. These are the variables that need more or less routine operator attention and, it is understood, are in addition to his regular equipment maintenance checks. The factors shown in columns 2 and 3 are among the main ones that govern initial process selection, plant design and, ultimately, future performance of the coal preparation plant. These factors highlight the fact that each of the processes has particular limits to its effective use, very often determined by raw coal washability or other characteristics, the cutpoint range and, in the case of cyclones, by some features of their design. Evidently, long-term performance reliability requires consistency in those feed characteristics that are shown as being critical for a given process. However, it is equally clear that this is not in itself the root cause of existing process inefficiencies, particulary those found in fine coal washing. For example, it appears from the plant results that in the case of heavy-medium cyclones and hydrocyclones, the question of geometry has not been entirely resolved as is the case for flotation, where numerous questions regarding the surface properties of the coal – among other factors – require answers.

The technology needs and areas of greatest potential benefit are reflected in the estimated annual dollar losses associated with clean coal production during the 1978-80 period (Table 27). It would be anticipated that since no real advance in the technology has occurred in the intervening years to 1984, present losses would remain roughly in the same proportions shown in Table 25 as "% of feed" for the individual processes. With rising coal production rates and the apparent trend towards increased use of the hydrocyclone and froth flotation along with heavy-medium processes that have taken place in Canada since 1980, it can be imagined that the value of losses would currently stand at correspondingly higher levels. For the future also, the industry may well have to be prepared for the real possibility of tighter market specifications and/or environmental regulations. The potential impact of such factors on production economics and on preparation technology requirements in general could by comparison even make the present situation look reasonably good.

Table 27 – Distribution and estimated value of clean coal losses annually during 1978-80

Process	Size range (mm)	Net clean coal losses		Estimated** value ('$000)
		tpa	%	
Coarse coal jig*	127-10	10 790	1.4	507
Heavy-medium vessel	127-10	33 490	4.2	1 574
Heavy-medium cyclone	38-0.6	44 065	5.5	2 071
Hydrocyclone	25-0.6	28 415	3.6	1 335
Concentrating table	3-0.1	77 000	9.7	3 619
Hydrocyclone	0.6-0.1	289 710	36.3	13 616
Froth flotation	0.6-0.1	312 900	39.3	14 706
Total	—	796 370	100.0	37 429

** Excluding Plant D; ** Based on $47 per tonne*

The need for resolution of the technological problems facing the coal preparation industry must be viewed with a sense of urgency. Real improvement in performance will only come about by first achieving a clear and thorough understanding of the mechanisms that govern particle separation and transport in a given process. On this basis alone will it be possible to arrive at the necessary modifications in equipment and design of operating and control systems. This effectively describes the philosophical approach taken at the CANMET Edmonton Coal Research Laboratory in its various coal preparation R & D programs now underway. While the approach has been able to provide some relatively quick answers by way of interim solutions to the washery operator, the intention and the objective is and must be to ensure long-lasting benefits. The hope is that others who may have gained an insight into the problems, their magnitude and their importance to the community in general will also join along with the industry in a concerted and devoted effort to find the needed answers.

ACKNOWLEDGEMENTS

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REFERENCES

1. Romaniuk, A.S. and Naidu, H.G. "Coal mining in Canada: 1983"; CANMET Report 83-20E; CANMET, Energy, Mines and Resources Canada; 1984.
2. Mikhail, M.W., Picard, J.L. and Humeniuk, O.E. "Performance evaluation of gravity separators in Canadian washeries"; Division Report ERP/ERL 81-31(OP); CANMET, Energy, Mines and Resources Canada; 1982.
3. Yancey, H.F. and Geer, M.R. "Efficiency and sharpness of separation in evaluating coal-washery performance"; Trans Soc Min Eng Am Inst Min Metall Pet Eng 190:507-517; 1951.
4. Vissac, G.A. "Scientific methods of washery controls"; First International Coal Preparation Congress; June 26-July 1, 1950; Paris; Paper A4, 1950.
5. Geer, M.R. and Yancey, H.F. "Cleaning characteristics and cleaning tests of Montana coal"; Report of Investigation 5103; U.S. Bureau of Mines; 1955.
6. Killmeyer, Jr., R.P. "A performance study of Baum and Batac jigs"; Min Congr J; 42-46; August, 1979.
7. Whitmore, R.L. "Principles of jig washing: an experimental approach"; J Inst Fuel: 3-11; January, 1958.
8. Savage, L.H. "The selection and treatment of magnetite for use in dense medium coal preparation"; Proc Symp on Magnetite in Coal Washing Practice; P.R. 68-3; Australian Coal Industry Research Laboratories; 9-18; 1968.
9. Burdon, R.G. "Applications of magnetite in the washing of coal"; Proc Symp on Magnetite in Coal Washing Practice; P.R. 68-3; Australian Coal Industry Research Laboratories; 30-40; 1968.
10. Merle, J.J., Jackson, C.R. and Burkitt, C. "Application of a dense medium vessel to upgrade bituminous coal by low gravity separation"; Fifth International Coal Preparation Congress; October 3-7, 1966; Pittsburgh; Paper E5, 333-346; 1968.
11. Geer, M.R., Sokaski, M., Jacobsen, P.S. and Yancey, H.F. "Performance of dense-medium cyclone in cleaning fine coal"; Report of Investigation 5732; U.S. Bureau of Mines; 1961.
12. Deurbrouck, A.W. "Washing fine-size coal in a dense-medium cyclone"; Report of Investigation 7982; U.S. Bureau of Mines; 1974.
13. Skolnik, E. "Heavy medium cleaning of −28 mesh coal"; Min Eng: 1235-1237; August, 1980.
14. Sokaski, M. and Geer, M.R. "Cleaning unsized fine coal in a dense-medium cyclone pilot plant"; Report of Investigation 6274; U.S. Bureau of Mines; 1963.
15. Visman, J. "The cleaning of highly friable coals by water cyclones"; Fourth International Coal Preparation Congress; May 28-June 1, 1962; Harrogate; Paper C2, 155-163; 1962.
16. Draeger, E.A. and Collins, J.W. "Efficient use of water only cyclones"; Min Eng: 1215-1217; August, 1980.
17. Weyher, L.H.E. and Lovell, H.L. "Hydrocyclone washing of fine coal"; Preprint; AIME Annual Meeting, New York; February, 1968.
18. Falconer, R.A. and Lovell, H.L. "The response of varying hydrocyclone cone angles in fine coal cleaning"; Preprint; AIME Coal Division – Preparation Section, Los Angeles; February, 1967.
19. Sands, P., Sokaski, M. and Geer, M.R. "Performance of the hydrocyclone as a fine-coal cleaner"; Report of Investigation 7067; U.S. Bureau of Mines; 1968.
20. Deurbrouck, A.W. "Performance characteristics of coal-washing equipment: hydrocyclones"; Report of Investigation 7891; U.S. Bureau of Mines; 1974.

21.Tiernan, C.H. "Concentrating tables for fine coal cleaning"; Min Eng; 1228 – 1230; August, 1980.
22.Deurbrouck, A.W. and Palowitch, E.R. "Cleaning fine coals on concentrating tables"; Fourth International Coal Preparation Congress; May 28-June 1, 1962; Harrogate; Paper C5, 181-192; 1962.
23.Lewis, J.L. "The clarification of coal washery water and froth flotation treatment of fine coals in Great Britain"; First International Coal Preparation Congress; June 26-July 1, 1950; Paris; Paper F7, 1950.
24.Plouf, T.M. "Froth flotation techniques reduce sulfur and ash"; Min Eng: 1218-1223; 1980.
25.Firth, B.A., Swanson, A.R. and Nicol, S.K. "The influence of feed size distribution on the staged flotation of poorly floating coals"; Australas Inst Min Metall Proc 267: 49-53; 1978.
26.Plaksin, I.N. and Klassen, V.I. "Methods of improving the process of froth flotation"; Fourth International Coal Preparation Congress; May 28-June 1, 1962; Harrogate; Paper D5, 263-271; 1962.
27.Miller, F.G. "The effect of froth sprinkling on coal flotation efficiency"; Trans Soc Min Eng Am Inst Min Metall Pet Eng 244: 158-167; 1969.
28.Brake, I. "Collector emulsification – a new approach to conditioning in coal flotation"; Paper presented to the Coal Preparation Society of Queensland, Moranbah, April 23, 1975.
29.Sun, S-C. "Effects of oxidation of coals on their flotation properties"; Trans Soc Min Eng Am Inst Min Metall Pet Eng; 396-401; 1954.
30.Nimerick, K.H. and Scott, B.E. "New method of oxidized coal flotation"; Min Cong J 37:21-22; 1980.

APPENDIX A

GLOSSARY

GLOSSARY

Ash error – the difference between the ash content of clean coal obtained by washing and the ash content theoretically obtainable at the same yield as given by the reconstituted feed washability curve (q.v.)

Cutpoint, separation density – conventionally, the relative density corresponding to 50% recovery on the partition curve (q.v.)

Dependent criteria – measures of separation efficiency, those that usually vary according to both the characteristics of the coal being washed and sharpness of the separation

Error area – the area enclosed by the partition curve and the lines representing perfect separation, conventionally, as cm^2 based on standardized scales

Fines – generally, coal with a maximum particle size below 4 mm and with no lower size limit

Floats – that portion of a raw coal or product which is lighter than a specified upper-density limit

Floats in refuse – that portion (wt%) of the refuse which is lighter than the relative density corresponding to the cutpoint, as determined by float-sink analysis of the product

Float-sink analysis – using a series of liquids/solutions, the separation of a coal sample into density fractions with defined limits, the proportion of each fraction expressed as percentage weight of the total sample, usually with a corresponding ash content or other characteristic as the basis for determining coal washability and partition curves

Imperfection – a measure that takes into account the effect of cutpoint on separation sharpness, conventionally defined as the ratio $R/(d_p-1)$ and considered of particular relevance to jig separation or other processes employing water medium

Independent criteria – measures of separation sharpness, those derived from the partition curve and that define the inherent separation capability of a given washing process for a given size fraction, usually considered constant and reproducible under given conditions and to be essentially independent of coal washability characteristics

Middlings – a product of separation consisting largely of the intermediate-density fractions and thus of quality intermediate between the clean coal and refuse, often consisting of intergrown constituents which may be liberated by crushing and subjected to further washing but commonly, as in the case of "true" middlings (bone), consisting of extremely finely disseminated constituents which cannot be liberated by crushing and therefore not amenable to further upgrading

Near-density material – percentage of material in the feed that lies within specified density limits on either side of the cutpoint, most commonly, $d_p \pm 0.10$

Organic efficiency – ratio between the actual yield of clean coal of given ash content obtained and the yield of clean coal of the same ash content theoretically obtainable as given by the reconstituted feed washability curve, expressed as percent (see yield error)

Partition curve, distribution curve, error curve – curve showing the percentage recovery of each density fraction of the reconstituted feed in one of the products of separation, commonly in the refuse (see independent criteria)

Partition number – ordinate scale of the partition curve (q.v.)

Probable error, ecart probable – average slope of the partition curve measured as one-half the difference between the relative densities corresponding to partition numbers 25 and 75%, a measure of separation sharpness

Reconstituted feed – composition of the feed to a plant or process calculated by combining properties of the products in the proportions obtained; most often differs from the raw feed as a result of the effects of particle degradation

Refuse, reject, tailing, discard, dirt – that portion of the raw coal which contains undesirable impurities and which is removed by washing, usually of high ash content and high relative density

Run-of-mine – raw coal as obtained from the mine unaltered by screening, crushing or other preparation

Sinks – that portion of a raw coal or product which is denser than a specified lower density limit

Sinks in clean coal – that portion (wt %) of the clean coal which is denser than the relative density corresponding to the cutpoint, as determined by float-sink analysis of the product

Theoretical yield – the maximum quantity of clean coal of specified ash content or other characteristic that is ideally obtainable by washing as given by the washability curve, expressed as percent of the feed

Total misplaced material – by reference to the cutpoint, the percentage of all those density fractions wrongly placed in each of the separation products, equal to the weighted sum of floats in refuse and sinks in clean coal

Washability – the general amenability of a coal to upgrading

Washability curve(s) – a curve or set of curves obtained from results of float-sink analysis which conventionlly define the feed-density distribution, ash distribution and ash-density relationship as the basis for evaluating washing potential and process requirements

Yield error, washing loss, recovery loss – the difference between the actual yield of clean coal of given ash content obtained and the yield of clean coal of the same ash content theoretically obtainable as given by the reconstituted feed washability curve.

APPENDIX B

PLANT DATA FOR COAL-WASHING PROCESSES

PLANT DATA FOR COAL-WASHING PROCESSES

Table B-1 – Washing results for coarse coal jig – Plant A

	Size fraction (mm)					
	50.8-25.4	25.4-9.5	9.5-1.6	1.6-0.6	50.8-9.5	9.5-0.6
Ash content (%)						
Raw coal	20.3	18.8	16.5	11.2	19.3	15.8
Reconstituted coal	21.0	17.9	15.8	11.5	18.9	15.2
Clean coal	13.6	12.5	9.6	6.7	12.8	9.1
Refuse	45.2	54.7	60.7	47.6	50.5	58.8
Yield of clean coal (%)	76.6	87.2	87.8	88.2	83.8	87.8
Theoretical yield (%)	77.5	88.2	89.0	91.8	85.0	89.3
Organic efficiency	98.8	98.9	98.6	96.1	98.6	98.3
Separation density (d_p)	1.685	1.768	1.790	1.750	1.727	1.790
Probable error	0.075	0.114	0.150	0.195	0.105	0.152
Error area	45	66	89	116	61	90
Imperfection	0.110	0.148	0.190	0.260	0.144	0.192
±0.10 Near density material (%)	14.0	6.0	3.9	2.2	8.8	3.2
Floats in refuse (%)	14.0	16.5	15.0	32.0	14.4	17.1
Sinks in clean coal (%)	3.1	2.0	1.8	1.6	2.5	1.8
Total misplaced material (%)	5.6	3.9	3.4	5.2	4.4	3.6
Yield error (%)	0.9	1.0	1.2	3.6	1.0	1.5
Ash error (%)	0.2	0.3	0.4	1.0	0.3	0.5

Table B-2 – Washing results for coarse coal jig – Plant B

	Size fraction (mm)						
	Plus 50.8	50.8-25.4	25.4-9.5	9.5-1.6	1.6-0.6	Plus 9.5	9.5-0.6
Ash content (%)							
Raw coal	66.8	49.5	38.8	29.1	22.0	52.9	26.3
Reconstituted coal	60.5	49.5	44.3	33.7	28.0	50.3	31.9
Clean coal	6.9	7.4	9.0	7.2	7.9	7.8	7.4
Refuse	74.9	72.5	69.0	53.0	43.9	72.3	50.2
Yield of clean coal (%)	21.23	35.5	41.2	42.2	44.2	34.1	42.8
Theoretical yield (%)	25.9	39.8	47.7	58.0	69.3	39.4	61.9
Organic efficiency (%)	81.8	89.2	86.4	72.8	63.8	86.6	69.1
Separation density (d_p)	1.430	1.521	1.562	1.455	1.410	1.510	1.433
Probable error	0.069	0.121	0.134	0.154	0.130	0.116	0.142
Error area	35	66	87	99	102	74	99
Imperfection	0.160	0.232	0.238	0.338	0.317	0.228	0.328
±0.10 Near density material(%)	11.0	6.0	6.8	13.5	36.0	6.1	21.8
Floats in refuse (%)	6.0	6.1	10.1	20.0	29.5	7.5	22.8
Sinks in clean coal (%)	3.5	3.0	6.8	10.5	14.0	4.6	12.1
Total misplaced material (%)	5.5	5.0	8.7	16.0	22.6	6.5	18.2
Yield error (%)	4.7	4.3	6.5	15.8	25.1	5.3	19.1
Ash error (%)	0.8	1.6	3.1	4.0	5.6	2.0	4.5

Table B-3 – Washing results for coarse coal jig – Plant C

	Size fraction (mm)		
	127-50.8	50.8-25.4	Plus 25.4
Ash content (%)			
Raw coal	83.7	76.3	80.7
Reconstituted coal	81.1	66.0	74.9
Clean coal	20.2	25.8	24.6
Refuse	83.8	77.7	81.6
Yield of clean coal (%)	4.2	22.5	11.7
Theoretical yield (%)	6.5	34.4	17.0
Organic efficiency (%)	65.1	65.4	68.8
Separation density (d_P)	1.500	1.798	1.692
Probable error	0.150	0.234	0.240
Error area	115	126	134
Imperfection	0.300	0.293	0.347
±0.10 Near density material (%)	2.6	1.6	2.9
Floats in refuse (%)	1.0	3.9	2.6
Sinks in clean coal (%)	30.8	17.6	17.4
Total misplaced material (%)	2.3	7.0	4.3
Yield error (%)	2.3	11.9	5.3
Ash error (%)	7.4	8.4	9.4

Table B-4 – Washing results for heavy-medium vessel – Plant A

	Size fraction (mm)			
	101.6-50.8	50.8-25.4	25.4-12.7	101.6-12.7
Ash content (%)				
Raw coal	32.1	25.2	25.7	26.8
Reconstituted coal	31.8	28.9	23.0	27.0
Clean coal	11.4	12.1	11.7	11.8
Refuse	54.0	58.8	60.4	58.1
Yield of clean coal (%)	52.1	64.0	76.8	67.1
Theoretical yield (%)	52.6	64.8	78.0	68.2
Organic efficiency (%)	99.0	98.8	98.5	98.4
Separation density (d_p)	1.519	1.555	1.576	1.552
Probable error	0.032	0.030	0.041	0.032
Error area	18	14	28	17
Imperfection	0.062	0.054	0.071	0.058
±0.10 Near density material (%)	33.0	20.0	16.9	17.9
Floats in refuse (%)	1.4	3.6	7.5	3.9
Sinks in clean coal (%)	2.0	0.6	0.8	0.9
Total misplaced material (%)	1.7	1.7	2.4	1.9
Yield error (%)	0.5	0.8	1.2	1.1
Ash error (%)	0.1	0.1	0.4	0.2

Table B-5 – Washing results for heavy-medium vessel – Plant B

	Size fraction (mm)						
	127.0-50.8	50.8-25.4	25.4-9.5	9.5-1.6	1.6-0.6	127.0-9.5	9.5-0.6
Ash content (%)							
Raw coal	78.2	75.4	59.2	43.4	23.6	71.3	42.6
Reconstituted coal	74.0	72.0	62.0	47.3	51.3	68.0	48.1
Clean coal	7.7	6.7	8.4	8.5	9.5	7.9	8.7
Refuse	80.6	77.0	70.5	63.7	62.3	75.1	63.4
Yield of clean coal (%)	9.0	7.1	13.6	29.6	20.8	10.6	27.9
Theoretical yield (%)	9.5	8.9	17.8	41.2	30.8	13.1	39.1
Organic efficiency (%)	94.7	79.8	76.4	71.8	67.5	80.9	71.4
Separation density (d_p)	1.397	1.360	1.374	1.397	1.392	1.370	1.397
Probable error	0.020	0.018	0.026	0.051	0.043	0.024	0.048
Error area	23	7	18	44	44	13	44
Imperfection	0.050	0.050	0.070	0.128	0.110	0.065	0.121
±0.10 Near density material (%)	11.5	15.5	23.8	36.2	30.2	18.0	34.0
Floats in refuse (%)	0.9	1.0	2.9	10.0	6.4	1.8	9.6
Sinks in clean coal (%)	2.6	15.9	19.6	14.8	22.4	19.1	15.4
Total misplaced material (%)	1.0	2.1	5.2	11.4	9.7	3.6	11.2
Yield error (%)	0.5	1.8	4.2	11.6	10.0	2.5	11.2
Ash error (%)	0.2	0.5	1.0	2.3	2.0	0.7	2.2

Table B-6 – Washing results for heavy-medium vessel – Plant C

	Size fraction (mm)						
	Plus 50.8	50.8-25.4	25.4-9.5	9.5-1.6	1.6-0.6	Plus 9.5	9.5-0.6
Ash content (%)							
Raw coal	31.1	32.9	28.2	21.6	15.4	30.3	19.6
Reconstituted coal	20.0	29.0	32.6	25.6	19.0	28.3	24.0
Clean coal	12.5	12.4	13.2	11.3	10.6	12.7	11.2
Refuse	43.4	50.7	53.9	46.7	37.8	51.3	44.9
Yield of clean coal (%)	75.8	56.7	52.4	59.8	69.2	59.6	62.0
Theoretical yield (%)	78.2	64.2	60.2	71.1	80.7	66.1	73.6
Organic efficiency (%)	96.9	88.3	87.0	84.1	85.8	90.2	84.2
Separation density (d_p)	1.502	1.490	1.498	1.496	1.498	1.492	1.496
Probable error	0.038	0.070	0.086	0.102	0.161	0.068	0.108
Error area	29	40	54	67	112	45	81
Imperfection	0.076	0.143	0.173	0.206	0.323	0.138	0.218
±0.10 Near density material (%)	23.5	29.0	26.3	26.1	21.0	27.5	26.0
Floats in refuse (%)	22.8	23.5	18.9	29.5	34.3	20.2	30.6
Sinks in clean coal (%)	0.5	4.2	6.3	6.0	7.6	3.7	6.1
Total misplaced material (%)	5.9	12.6	12.3	15.5	15.8	10.4	15.4
Yield error (%)	2.5	7.6	7.8	11.3	11.5	6.5	11.6
Ash error (%)	0.2	1.0	1.4	1.9	2.2	0.9	2.0

Table B-7 – Washing results for heavy-medium cyclone – Plant A

	Size fraction (mm)							
	38.1-25.4	25.4-12.7	12.7-9.5	9.5-1.6	1.6-0.6	38.1-9.5	38.1-0.6	9.5-0.6
Ash content (%)								
Raw coal	51.0	42.8	34.4	28.0	20.6	43.5	32.1	24.8
Reconstituted coal	62.2	48.6	39.0	23.7	13.0	51.0	34.1	22.6
Clean coal	10.3	9.8	6.8	5.2	5.3	9.0	6.2	5.2
Refuse	77.4	76.6	74.3	73.0	67.2	76.6	75.1	72.7
Yield of clean coal (%)	22.6	41.9	52.3	72.7	87.5	37.9	59.4	74.2
Theoretical yield (%)	23.3	42.3	52.8	73.2	87.8	38.5	59.9	74.7
Organic efficiency (%)	97.0	99.1	99.0	99.3	99.7	98.4	99.2	99.3
Separation density (d_p)	1.538	1.524	1.516	1.525	1.563	1.524	1.526	1.529
Probable error	0.012	0.020	0.018	0.028	0.036	0.019	0.025	0.030
Error area	8	13	11	16	18	13	14	16
Imperfection	0.022	0.038	0.035	0.053	0.064	0.036	0.048	0.057
±0.10 Near density material(%)	9.3	12.6	12.9	10.0	6.1	11.6	10.1	9.2
Floats in refuse (%)	1.9	1.7	1.4	4.5	9.8	1.4	3.3	5.5
Sinks in clean coal (%)	2.0	0.7	0.7	0.8	0.6	1.2	0.9	0.8
Total misplaced material (%)	1.9	1.3	1.0	1.8	1.8	1.3	1.9	2.0
Yield error (%)	0.7	0.4	0.5	0.5	0.3	0.6	0.5	0.5
Ash error (%)	0.4	0.1	0.1	0.1	0.1	0.2	0.1	0.1

Table B-8 – Washing results for heavy-medium cyclone – Plant B

	Size fraction (mm)							
	38.1-25.4	25.4-12.7	12.7-9.5	9.5-1.6	1.6-0.6	38.1-9.5	38.1-0.6	9.5-0.6
Ash content (%)								
Raw coal	23.7	18.4	18.0	18.1	18.2	19.3	18.6	18.1
Reconstituted coal	19.6	17.5	14.0	14.4	17.0	17.4	16.1	14.7
Clean coal	3.2	2.5	2.2	2.0	1.7	2.6	2.3	2.0
Refuse	73.2	66.2	59.5	55.3	46.5	67.2	60.8	53.8
Yield of clean coal (%)	76.6	76.5	79.4	76.8	65.8	77.1	76.4	75.5
Theoretical yield (%)	77.3	77.6	80.8	81.0	75.3	78.5	79.2	80.0
Organic efficiency (%)	99.1	98.6	98.3	94.8	87.4	98.2	96.5	93.4
Separation density (d_p)	1.380	1.331	1.327	1.327	1.357	1.340	1.330	1.327
Probable error	0.018	0.012	0.014	0.015	0.039	0.020	0.020	0.018
Error area	10	8	8	10	22	11	11	13
Imperfection	0.047	0.036	0.043	0.046	0.109	0.059	0.061	0.055
±0.10 Near density material (%)	13.5	26.0	30.0	11.2	17.5	23.0	23.0	17.5
Floats in refuse (%)	6.4	8.0	12.0	19.5	26.0	8.6	13.5	20.5
Sinks in clean coal (%)	0.6	1.1	0.8	0.3	1.8	0.4	0.6	0.4
Total misplaced material (%)	2.0	2.7	3.1	4.8	10.1	2.3	3.6	5.3
Yield error (%)	0.7	1.1	1.4	4.2	9.5	1.4	2.8	5.3
Ash error (%)	0.1	0.05	0.1	0.2	0.2	0.1	0.1	0.3

Table B-9 – Washing results for heavy-medium cyclone – Plant C

	Size fraction (mm)				
	12.7-9.5	9.5-1.6	1.6-0.6	12.7-0.6	9.5-0.6
Ash content (%)					
Raw coal	24.4	21.6	20.2	21.4	21.0
Reconstituted coal	23.8	23.0	18.8	21.7	21.4
Clean coal	13.4	12.3	10.6	11.9	11.6
Refuse	61.8	59.1	57.4	59.0	58.6
Yield of clean coal (%)	78.5	77.1	82.4	79.0	79.1
Theoretical yield (%)	79.0	79.1	85.9	81.4	81.7
Organic efficiency (%)	99.4	97.5	95.9	97.0	96.8
Separation density (d_p)	1.650	1.650	1.691	1.660	1.660
Probable error	0.042	0.050	0.076	0.052	0.055
Error area	23	36	72	41	45
Imperfection	0.065	0.077	0.110	0.079	0.083
±0.10 Near density material (%)	10.4	9.2	5.6	7.9	7.5
Floats in refuse (%)	8.2	9.5	21.5	11.0	11.4
Sinks in clean coal (%)	0.6	1.8	2.0	1.9	2.0
Total misplaced material (%)	2.2	3.6	5.4	3.8	4.0
Yield error (%)	0.5	2.0	3.5	2.4	2.6
Ash error (%)	0.2	0.6	1.1	0.7	0.8

Table B-10 – Washing results for heavy-medium cyclone – Plant D

	Size fraction (mm)		
	9.5-1.6	1.6-0.6	9.5-0.6
Ash content (%)			
Raw coal	27.1	22.8	25.4
Reconstituted coal	24.8	16.8	22.3
Clean coal	13.8	10.8	12.8
Refuse	64.8	67.2	65.2
Yield of clean coal (%)	78.4	89.4	81.9
Theoretical yield (%)	78.6	89.6	82.0
Organic efficiency (%)	99.8	99.8	99.9
Separation density (d_p)	1.637	1.720	1.666
Probable error	0.033	0.046	0.036
Error area	16	28	20
Imperfection	0.052	0.064	0.054
±0.10 Near density material (%)	7.2	2.1	4.8
Floats in refuse (%)	2.8	7.5	4.0
Sinks in clean coal (%)	0.1	0.1	0.2
Total misplaced material (%)	0.7	0.9	0.9
Yield error (%)	0.2	0.2	0.1
Ash error (%)	0.0	0.1	0.0

Table B-11 – Washing results for heavy-medium cyclone – Plant E

	Size fraction (mm)						
	50.8-25.4	25.4-9.5	9.5-1.6	1.6-0.6	50.8-9.5	50.8-0.6	9.5-0.6
Ash content (%)							
Raw coal	21.9	17.8	18.8	15.9	19.1	17.4	16.0
Reconstituted coal	21.8	17.8	17.0	11.3	19.0	17.4	16.0
Clean coal	11.7	9.8	8.0	6.5	10.4	8.9	7.7
Refuse	69.7	63.8	57.8	59.2	65.5	61.5	57.8
Yield of clean coal (%)	82.6	85.2	81.9	90.9	84.4	83.9	83.4
Theoretical yield (%)	83.1	85.5	83.4	91.6	84.7	84.6	84.8
Organic efficiency (%)	99.4	99.6	98.2	99.2	99.6	99.2	98.4
Separation density (d_p)	1.709	1.642	1.654	1.682	1.674	1.679	1.659
Probable error	0.043	0.039	0.070	0.059	0.045	0.059	0.063
Error area	28	20	39	35	26	35	35
Imperfection	0.061	0.061	0.101	0.086	0.067	0.087	0.096
±0.10 Near density material (%)	7.0	5.8	6.8	4.7	5.2	5.9	6.2
Floats in refuse (%)	8.0	6.3	13.2	16.5	7.8	11.1	14.2
Sinks in clean coal (%)	2.1	0.3	1.8	1.4	0.6	0.3	1.8
Total misplaced material (%)	3.1	1.2	3.9	2.8	1.7	2.0	3.9
Yield error (%)	0.5	0.3	1.5	0.7	0.3	0.7	1.4
Ash error (%)	0.2	0.1	0.4	0.2	0.1	0.2	0.4

Table B-12 – Washing results for heavy-medium cyclone – Plant F

	Size fraction (mm)		
	9.5-1.6	1.6-0.6	9.5-0.6
Ash content (%)			
Raw coal	33.7	25.0	28.2
Reconstituted coal	36.0	16.0	26.9
Clean coal	10.2	10.3	10.3
Refuse	68.4	69.9	68.5
Yield of clean coal (%)	55.7	90.5	71.5
Theoretical yield (%)	58.3	92.0	74.3
Organic efficiency (%)	95.5	98.4	96.2
Separation density (d_p)	1.518	1.870	1.570
Probable error	0.054	0.310	0.102
Error area	32	136	73
Imperfection	0.104	0.356	0.179
±0.10 Near density material (%)	17.0	1.1	11.9
Floats in refuse (%)	6.2	24.0	11.8
Sinks in clean coal (%)	7.0	0.7	2.2
Total misplaced material (%)	6.6	2.9	4.9
Yield error (%)	2.6	0.5	2.8
Ash error (%)	0.7	0.4	0.7

Table B-13 – Washing results for hydrocyclone – Plant A

	Size fraction (mm)			
	19.0-9.5	9.5-3.2	3.2-0.6	19.0-0.6
<u>Ash content (%)</u>				
Raw coal	10.8	11.7	11.6	11.5
Reconstituted coal	10.5	12.1	12.7	11.8
Clean coal	10.2	11.0	9.8	10.6
Refuse	28.0	24.2	27.0	25.6
Yield of clean coal (%)	98.4	91.7	83.1	92.2
Theoretical yield (%)	99.2	97.7	94.3	97.7
Organic efficiency (%)	99.2	93.9	88.1	94.4
Separation density (d_p)	1.902	1.950	1.756	1.890
Probable error	0.135	0.285	0.261	0.246
Error area	73	155	138	151
Imperfection	0.150	0.300	0.345	0.276
±0.10 Near density material (%)	0.2	1.0	3.1	1.2
Floats in refuse (%)	91.7	89.8	75.1	86.2
Sinks in clean coal (%)	0.1	0.4	1.9	0.5
Total misplaced material (%)	1.5	7.8	14.3	7.2
Yield error (%)	0.8	6.0	11.2	5.5
Ash error (%)	0.2	1.6	2.5	1.4

Table B-14 – Washing results for hydrocyclone – Plant B

	Size fraction (mm)			
	19.0-9.5	9.5-3.2	3.2-0.6	19.0-0.6
<u>Ash content (%)</u>				
Raw coal	28.9	19.3	13.7	23.7
Reconstituted coal	54.4	31.8	11.5	30.2
Clean coal	9.3	7.9	7.1	7.8
Refuse	74.5	53.1	23.9	55.2
Yield of clean coal (%)	30.9	47.2	73.6	52.6
Theoretical yield (%)	34.4	62.9	93.6	67.0
Organic efficiency (%)	89.8	75.0	78.6	78.5
Separation density (d_p)	1.608	1.540	1.750	1.590
Probable error	0.099	0.172	0.393	0.200
Error area	72	102	199	118
Imperfection	0.163	0.318	0.524	0.339
±0.10 Near density material (%)	4.6	7.8	3.3	5.9
Floats in refuse (%)	3.5	25.8	73.0	27.0
Sinks in clean coal (%)	6.5	7.3	4.2	6.4
Total misplaced material (%)	4.4	17.1	22.3	16.2
Yield error (%)	3.5	15.7	20.0	14.4
Ash error (%)	3.0	4.8	4.7	4.7

Table B-15 – Washing results for hydrocyclone – Plant B′

	Size fraction (mm)					
	Plus 3.2	3.2-0.6	0.6-0.15	Plus 0.6	Plus 0.15	3.2-0.15
Ash content (%)						
Raw coal						
Reconstituted coal	9.7	22.2	21.5	19.0	19.9	21.9
Clean coal	8.7	14.0	16.5	12.3	14.1	15.4
Refuse	48.8	51.0	46.2	51.1	48.2	48.0
Yield of clean coal (%)	97.5	77.7	83.2	82.8	83.0	80.2
Theoretical yield (%)	98.5	86.2	90.1	89.0	89.5	88.2
Organic efficiency (%)	99.0	90.1	92.3	93.0	92.7	90.9
Separation density (d_p)	—	2.045	—	2.070	2.253	2.253
Probable error	—	0.398	—	0.392	0.364	0.370
Error area	164	202	173	196	186	192
Imperfection	—	0.380	—	0.366	0.291	0.295
±0.10 Near density material (%)	—	1.8	—	4.6	5.0	5.3
Floats in refuse (%)	—	41.5	—	43.8	49.8	49.2
Sinks in clean coal (%)	—	8.6	—	6.2	6.5	8.2
Total misplaced material (%)	—	15.9	—	12.7	13.9	16.3
Yield error (%)	1.0	8.5	6.9	6.2	6.5	8.0
Ash error (%)	0.5	4.8	3.5	3.5	3.6	4.4

Table B-16 – Washing results for hydrocyclone – Plant C

	Size fraction (mm)			
	25.4-12.7	12.7-6.4	6.4-0.6	25.4-0.6
Ash content (%)				
Raw coal	72.8	64.8	49.7	62.7
Reconstituted coal	70.6	52.7	49.4	60.5
Clean coal	19.6	15.2	16.0	15.8
Refuse	86.4	81.1	64.1	81.1
Yield of clean coal (%)	23.6	43.1	30.5	31.5
Theoretical yield (%)	28.6	50.4	54.6	38.4
Organic efficiency (%)	82.5	85.5	55.9	82.0
Separation density (d_p)	2.005	1.788	1.428	1.855
Probable error	0.162	0.260	0.238	0.274
Error area	95	134	131	142
Imperfection	0.161	0.323	0.556	0.320
±0.10 Near density material (%)	0.8	0.5	38.0	1.1
Floats in refuse (%)	1.4	9.2	20.1	9.1
Sinks in clean coal (%)	5.4	2.8	22.8	5.0
Total misplaced material (%)	2.3	6.4	20.9	7.8
Yield error (%)	5.0	7.3	24.1	6.9
Ash error (%)	3.6	4.3	9.2	5.1

Table B-17 – Washing results for hydrocyclone – Plants D and E

	Plant D			Plant E		
	Size fraction (mm)			Size fraction (mm)		
	0.6-0.15	0.15-0.10	0.6-0.10	0.6-0.15	0.15-0.10	0.6-0.10
Ash content (%)						
Raw coal	18.3	16.5	17.8	18.4	17.6	18.3
Reconstituted coal	17.8	18.4	18.0	12.4	13.9	12.6
Clean coal	10.0	12.7	10.6	10.6	12.2	10.8
Refuse	28.6	30.9	29.2	30.0	46.1	32.3
Yield of clean coal (%)	58.0	68.4	60.4	91.2	95.0	91.8
Theoretical yield (%)	86.2	90.2	87.4	96.4	97.1	96.5
Organic efficiency (%)	67.3	75.8	69.1	94.6	97.8	95.1
Separation density (d_p)	1.568	1.900	1.640	1.980	2.208	2.022
Probable error	0.396	0.314	0.403	0.094	0.072	0.116
Error area	185	206	196	70	66	84
Imperfection	0.697	0.349	0.630	0.096	0.060	0.114
±0.10 Near density material (%)	12.8	3.1	7.2	3.9	2.8	2.6
Floats in refuse (%)	52.1	60.2	57.2	67.5	66.0	67.5
Sinks in clean coal (%)	12.6	6.0	10.1	0.2	0	0.2
Total misplaced material (%)	29.2	23.1	28.7	6.1	3.3	5.7
Yield error (%)	28.2	21.8	27.0	5.2	2.1	4.7
Ash error (%)	5.6	7.4	6.1	1.3	1.1	1.3

Table B-18 – Washing results for hydrocyclone – Plants F and G

	Plant F			Plant G		
	Size fraction (mm)			Size fraction (mm)		
	0.6-0.15	0.15-0.10	0.6-0.10	0.6-0.15	0.15-0.10	0.6-0.10
Ash content (%)						
Raw coal	13.8	21.1	14.7	47.8	81.3	55.9
Reconstituted coal	13.4	24.4	15.7	46.8	83.2	52.2
Clean coal	12.8	24.2	15.1	23.7	79.4	31.8
Refuse	36.9	66.7	41.8	58.7	85.5	63.0
Yield of clean coal (%)	97.6	99.5	98.0	34.0	37.6	34.6
Theoretical yield (%)	99.3	99.8	99.4	59.8	88.5	64.9
Organic efficiency (%)	98.3	99.7	98.6	56.9	42.5	53.3
Separation density (d_p)	—	—	—	1.618	—	1.610
Probable error	—	—	—	0.315	—	0.326
Error area	—	—	—	157	>238	172
Imperfection	—	—	—	0.510	—	0.534
±0.10 Near density material (%)	—	—	—	14.4	—	12.8
Floats in Refuse (%)	—	—	—	19.8	—	17.7
Sinks in Clean Coal (%)	—	—	—	31.5	—	41.0
Total Misplaced Material (%)	—	—	—	23.8	—	25.8
Yield Error (%)	1.7	0.3	1.4	25.8	50.9	30.3
Ash Error (%)	1.2	0.5	1.2	14.6	24.8	21.1

Table B-19 – Washing results for froth flotation – Plants A and B

	Plant A			Plant B		
	Size fraction (mm)			Size fraction (mm)		
	0.6-0.15	0.15-0.10	0.6-0.10	Plus 0.15	0.15-0.10	Plus 0.10
Ash content (%)						
Raw coal	10.2	10.5	10.3	12.8	13.2	13.0
Reconstituted coal	12.8	18.0	13.6	9.3	17.6	13.3
Clean coal	4.7	6.0	4.9	8.5	17.0	12.9
Refuse	41.3	71.3	45.0	13.5	32.3	17.7
Yield of clean coal (%)	77.7	81.6	78.3	83.7	96.0	89.6
Theoretical yield (%)	88.8	85.2	88.2	99.2	99.0	99.3
Organic efficiency (%)	87.5	95.8	88.8	84.4	97.0	90.0
Separation density (d_p)	1.515	1.485	1.508	2.058	—	—
Probable error	0.403	0.446	0.413	—	—	—
Error area	201	204	196	200	73	109
Imperfection	0.782	0.920	0.813	—	—	—
±0.10 Near density material (%)	2.8	3.2	3.2	3.1	—	—
Floats in refuse (%)	44.7	12.6	40.5	85.6	—	—
Sinks in clean coal (%)	4.3	4.6	4.4	3.9	—	—
Total misplaced material (%)	13.3	6.1	12.2	17.2	—	—
Yield error (%)	11.1	3.6	9.9	15.5	4.0	9.9
Ash error (%)	2.6	2.5	2.7	3.1	1.9	5.7

Table B-20 – Washing results for froth flotation – Plants C and D

	Plant C			Plant D		
	Size fraction (mm)			Size fraction (mm)		
	0.6-0.15	0.15-0.10	0.6-0.10	Plus 0.15	0.15-0.10	Plus 0.10
Ash content (%)						
Raw coal	13.6	11.8	13.3	10.8	11.6	11.0
Reconstituted coal	9.6	15.0	10.1	8.6	10.9	9.0
Clean coal	6.4	9.7	6.7	8.3	8.1	8.1
Refuse	22.6	27.2	23.1	8.7	13.8	9.5
Yield of clean coal (%)	80.2	69.8	79.3	36.8	50.2	39.2
Theoretical yield (%)	95.7	90.2	95.3	99.7	93.5	98.2
Organic efficiency (%)	83.8	77.4	83.2	36.9	53.7	39.9
Separation density (d_p)	2.085	1.585	1.700	1.290	1.460	1.295
Probable error	0.522	0.479	0.512	—	—	—
Error area	220	221	238	204	133	238
Imperfection	0.481	0.819	0.731	—	—	—
±0.10 Near density material (%)	0.5	7.3	3.5	66.1	27.7	68.5
Floats in refuse (%)	79.0	58.6	71.0	20.2	72.5	23.5
Sinks in clean coal (%)	2.1	8.9	5.1	57.8	13.5	59.0
Total misplaced material (%)	17.3	23.9	18.7	34.0	42.9	37.4
Yield error (%)	15.5	20.4	16.0	62.9	43.3	59.0
Ash error (%)	2.7	5.5	3.0	5.6	5.0	5.3

Table B-21 – Washing results for froth flotation – Plant E

	Size fraction (mm)		
	0.6-0.15	0.15-0.10	28-150
Ash content (%)			
Raw coal	7.9	20.5	12.1
Reconstituted coal	5.5	12.1	6.6
Clean coal	5.0	6.2	5.1
Refuse	8.0	20.3	12.2
Yield of clean coal (%)	83.1	50.1	79.0
Theoretical yield (%)	97.7	92.0	95.9
Organic efficiency (%)	85.1	63.2	82.4
Separation density (d_p)	1.314	1.300	1.314
Probable error	0.018	0.273	0.018
Error area	35	153	35
Imperfection	0.057	0.910	0.057
±0.10 Near density material (%)	17.9	51.5	28.7
Floats in refuse (%)	62.6	36.1	55.5
Sinks in clean coal (%)	7.8	34.1	10.6
Total misplaced material (%)	17.1	35.9	20.0
Yield error (%)	14.6	33.9	16.9
Ash error (%)	1.5	3.5	1.7