Coal-preparation routes for maximum coal recovery

by T. A. CLAASEN*, Pr. Eng., B.Sc. (Eng.), M.S.A.C.P.S.

SYNOPSIS

Exceptionally low yields in the coking-coal fields of the Soutpansberg area in the northern Transvaal, are taxing the skill and ingenuity of preparation engineers. The coals are highly intergrown, and a series of crushing and washing steps may be required to extract the maximum usable tonnage. In addition, novel preparation techniques may have to be employed.

This paper, which is a case study of work recently undertaken, gives a general account of raw coal properties, emphasizing those related to maximum recoveries, and outlines the steps involved in the design of appropriate flow sheets based on laboratory studies.

SAMEVATTING

Die besonder lae opbrengste in die kookskoolvelde van die Soutpansberggebied in Noord-Transvaal stel hoë eise aan die vaardigheid en vernuf van die bereidingsingenieurs. Die steenkool is in 'n hoë mate vergroei en 'n reeks vergruis- en wasstappe kan nodig wees om die maksimum bruikbare tonnemaat te ekstraheer. Verder sal daar moontlik nuwe bereidingsmetodes toegepas moet word.

Hierdie verhandeling, wat 'n gevallestudie is van werk wat onlangs gedoen is, gee 'n algemene verslag oor die eienskappe van die rou steenkool met die klem op dié wat op maksimum herwinnings betrekking het en beskryf in hooftrekke die stappe betrokke by die ontwerp van gepaste vloeidiagramme aan die hand van laboratoriumstudies.

INTRODUCTION

The depletion of metallurgical coal has resulted in Iscor having to consider all the potential sources of coking coal, even though the yields are exceptionally low in some cases. Durban Navigation Collieries is an illustration of how a source of metallurgical coal was depleted for uses other than coking. These collieries have been producing coal for seventy-five years, and it was only after 1955, when Iscor purchased the mine, that the total production was routed to the coke ovens.

The occurrence of coal seams in the Karroo succession north of the Soutpansberg has been known for many years. In the vicinity of Lilliput siding, 20 km south of Messina, coal was mined during 1902 by the Messina Copper Company for smelting and general purposes. The Department of Mines drilled 14 boreholes during 1957 and 1958 to reconnoitre the Soutpansberg coalfield. Encouraged by the good swelling properties of the coal, Iscor took 15 farms under option in the early 1960s and drilled 32 boreholes in the vicinity of Waterpoort station, north of Soutpansberg.

Towards the end of 1973, an extensive exploration programme was initiated north of the Soutpansberg, where sufficient reserves to warrant exploitation were proved in three different areas: Blocks C, D, and E. Block C is a multi-seam opencast proposition that dips 12 to 20 degrees north. Blocks D and E are underground propositions with seam thicknesses of approximately 2,3 m and 3,2 m respectively.

The coal from Block C eastwards through Block D to Block E is essentially a bright coal with a vitrinite content of approximately 86 per cent. Coal yields for Block C are approximately 20 per cent at 12 per cent ash, depending on the selectivity of mining and the beneficiation process employed, while the whole seam yields approximately 41 per cent and 50 per cent in Blocks D and E respectively at 12 per cent ash.

Steyn[1] plotted a family of curves of equal abrasion and of equal coke strength, expressed in terms of M10- and M40Y indexes as a function of the RoR-index and the composition balance index. These curves are reproduced in Fig. 1, which indicates the position of the coal from Blocks C, D, and E in relation to the area of optimum blend composition and the low vitrinite coals currently used for blending. Area 1 indicates high-rank, low-volatile, low-vitrinite coals such as Hlobane, Vryheid Coronation, and Indumeni. Area 2 indicates the low-rank, high-volatile, low-vitrinite Witbank coking coals.

The quality of South African and U.S. coals is compared in Table I.

Float and sink analyses of borehole samples from the Soutpansberg coalfield indicated that coking properties persisted to high densities and high ash contents. The characteristic of the Soutpansberg coalfield is its high vitrinite content; therefore, any middling produced during the cleaning process will consist of vitrinite with interbanded mineral material. The process development must, therefore, take this factor into account when considering the extent to which the coal should be liberated.

The friable nature of the seams in the northern Transvaal could have a profound effect on the mining and

TABLE I

COMPARISON OF THE QUALITY OF SOME COALS

Source	Roga	Volatile material %	Ash	Vitrinite %	RoV
North-eastern Natal	65	22	13,0	45	1,2
Witbank	50	34	10	54	0,8
U.S.A.	80	24	8,0	86	1,3
Soutpansberg	85	23 to 30	12†	86	1 to 1,4

†Still to be confirmed

*Formerly South African Iron and Steel Industrial Corporation Limited; now Welmet Equipment, P.O. Box 41189, Craighall 2024, South Africa.

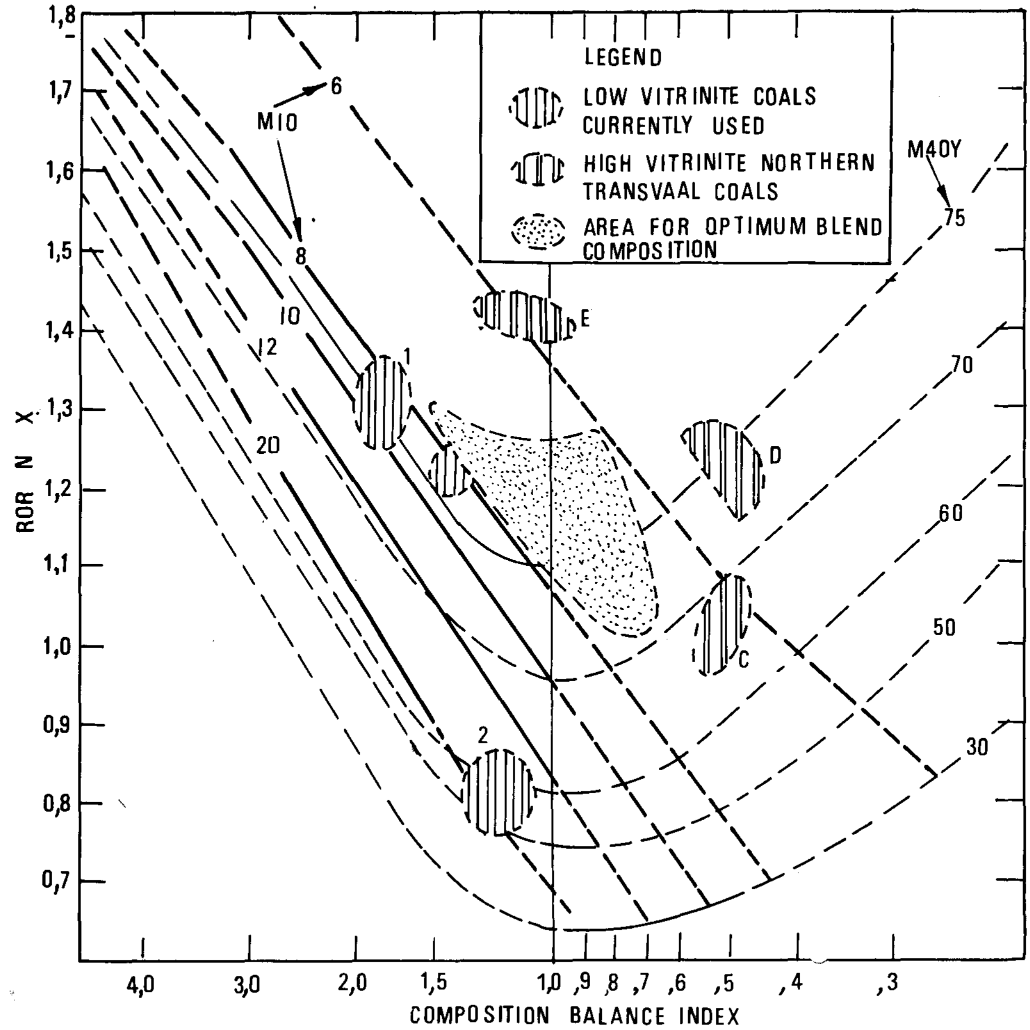

Fig. 1–Area of optimum blend and location of Soutpansberg high-vitrinite coals with reference to some Natal and Witbank coals[1]
1 North-eastern Natal 2 Witbank

process development, especially where opencast mining is used.

METALLURGICAL COAL FROM BLOCK C

An investigation of a number of large-core boreholes drilled on the farm Fripp, Block C, indicated that the method of mining will affect the design of the extractive process for the recovery of the coal. It was therefore decided to sink a boxcut on the farm Fripp, mine the coal seams, and extract a bulk sample for large-scale process testing.

During the initial planning of the excavation, it was decided to drill six large-core boreholes in the location of the boxcut for the express purpose of

(a) obtaining more information for the mining engineers to finalize their plans for the boxcut,
(b) obtaining test results on a sample taken in accordance with the mining method suitable for these seams,
(c) obtaining samples of coal 'selectively mined' and 'less selectively mined' for comparison with the results obtained from bulk samples,
(d) obtaining sink–float analyses of samples of relative densities in excess of 1,9,
(e) comparing the yields of coking coal obtained at 12 per cent and 14 per cent ash,
(f) obtaining preliminary information regarding the partition densities, crusher settings, and screen sizes for optimum yields of coking coal, and
(g) obtaining enough sample of middling coal for the investigation of processes such as the flotation of micro-fine material and oil agglomeration.

The six borehole cores were divided into two groups so that a selective mining method could be simulated by selective sampling of the one group, and a less-selective mining method by suitable sampling of the other group. So that the yields of coking coal possible from the two groups of samples could be compared, an identical flowsheet was followed for the tests. This is summarized in Fig. 2. Table II summarizes the results obtained.

Full float and sink analyses were conducted on all the fractions to determine the relevant partition densities for the removal of shale and for the recovery of coking coal at 12 per cent ash.

SAMPLING OF MINED MATERIAL

Since limited mining methods could be tested in the boxcut, it was decided that the boxcut should be wide enough for three 5 m strips embracing all the coal zones (Fig. 3). This meant that a large sample-storage area had to be prepared for the bulk samples from two of these strips (Fig. 4). It was realized from the onset of this exercise that the success depended on the accuracy of sampling and the reproducibility of the bulk sub-samples.

It is well known that it is extremely difficult to obtain a representative sample of raw coal especially at a top size of plus 150 mm. This problem was made more difficult by the need for the sample to be divided into at least four sub-samples of approximately 500 t each. Each of the sub-samples had to be representative of all the zones mined. To achieve this, it was decided that the increments would be as small as practically possible.

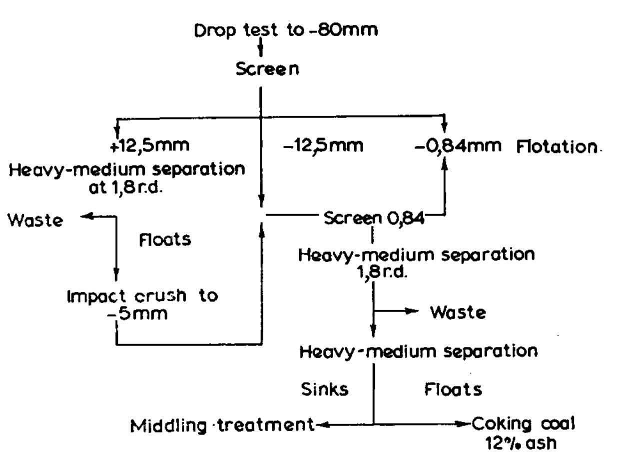

Fig. 2—Flowsheet followed for the tests on the yield of coking coal at 12 per cent ash

For the purpose of this discussion, only the sampling of the less selectively mined strip is considered.

It was calculated that approximately 3600 t of coal would be mined from a strip 5 m wide by 3 m deep. This meant that seven sub-samples of approximately 500 t each could be extracted. It was therefore decided that four 500 t samples would be prepared, and the balance of the material would be stockpiled in a separate area.

The division of the four zones was clearly marked in the boxcut, and each zone was mined separately, weighed, and tipped onto the marked-off areas in the prepared sample area. So that the increments of each zone could be correctly tipped, it was decided that the tipping would be on a cyclic basis. Each cycle was made up of seven lorry loads that were weighed and then tipped onto the marked-out area, e.g., N S 1 then N S 2 and so on until the complete coal zone was mined. The same procedure was then followed for the next zone, and so on. The interbanded mineral material was re-

TABLE II

YIELD OF COKING COAL AT 12 PER CENT ASH

Particle-size fraction	Group A Selectively mined, %		Group B Less-selectively mined, %
	As sampled	Calculated on basis of Group B	As sampled
−12,5+0,84 mm	13,07	9,83	10,84
−0,84 mm	8,23	6,19	7,53
Middling (−0,21 mm)	4,20	3,16	3,60
Total	25,50	19,18	20,73

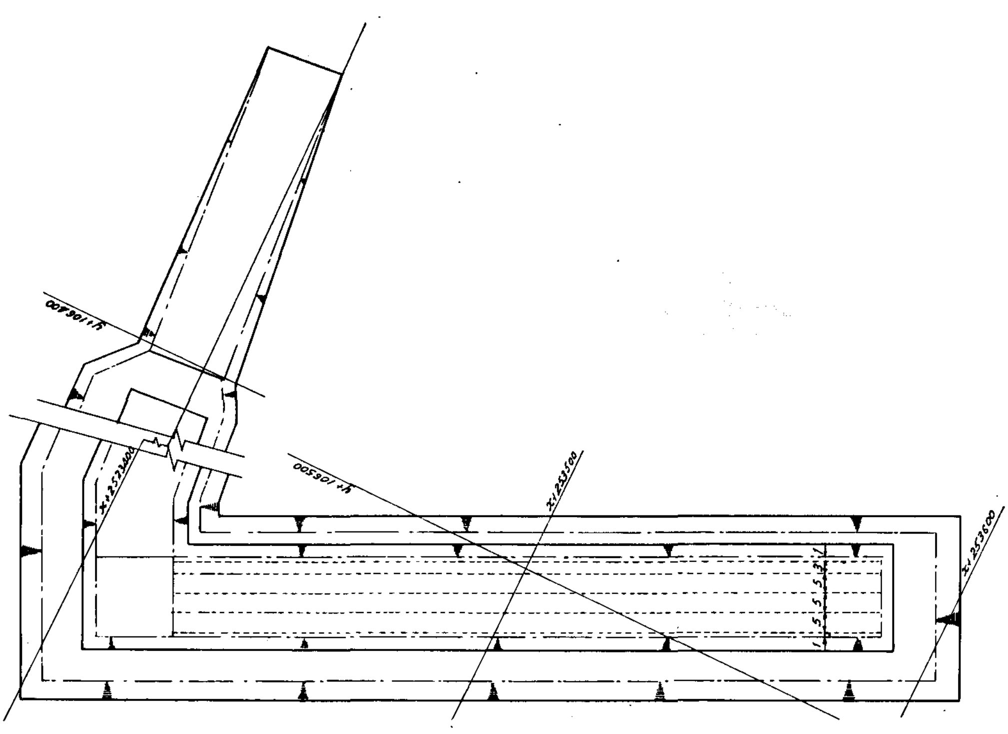

Fig. 3—Plan of the boxcut on the farm Fripp, Soutpansberg Block C

moved from the boxcut as well and stockpiled adjacent to the sample (Fig. 4).

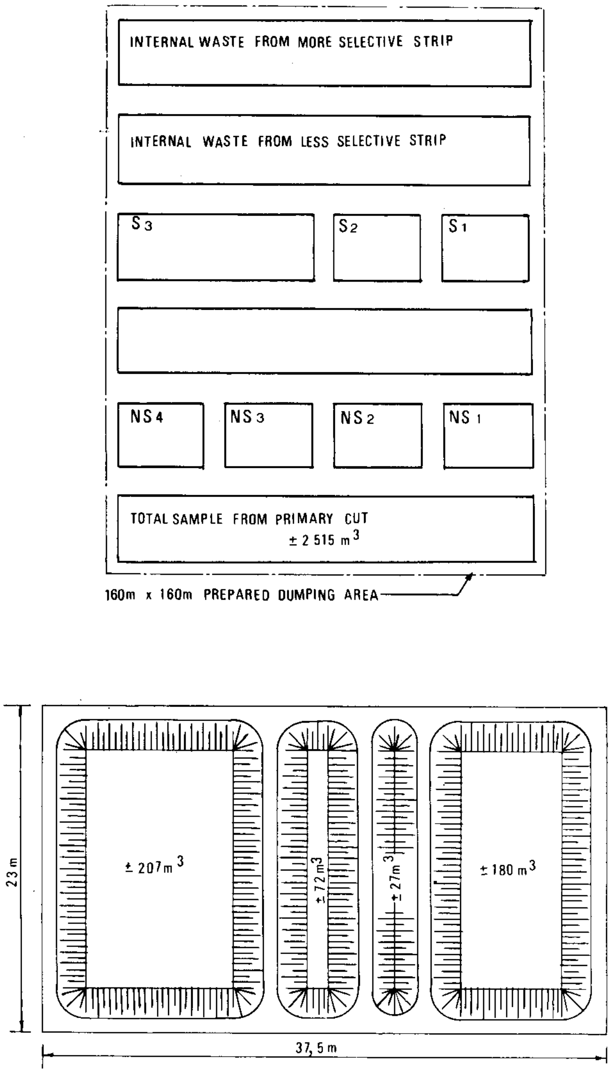

Fig. 4—Plan of the area from which the bulk samples were taken

MINING OF THE SAMPLES

The main advantage of a selective mining method is that less waste has to be transported to the plants. As the strike length of the Block C coalfield is 21 km, this could prove to be a considerable saving.

The disadvantage is that, as the bands of coal in the seam are thin, coal can be stripped off with the waste unless extreme care is exercised. The friable nature of the coal dictates that equipment should be moved as little as possible on the coal bands, which is difficult during selective mining.

FLOWSHEET DESIGN AND PROCESS DEVELOPMENT

The pilot-plant tests on the large-diameter borehole cores had given an indication of the flowsheet by which a high percentage of the coking coal available in the raw-coal feed could possibly be recovered (Fig. 2). What remained to be done included bulk testing of this flowsheet, checking on individual processes (for example, the recovery of fines), and considering the equipment that would perform the task efficiently and cheaply.

The inherent features of the Block C deposit are that the crystals of high-vitrinite coal are closely associated with the shales, are present in very thin bands, and are very friable; the coal zones are interlayered with very hard stone bands and lenses. The mining method that would be the most successful is possibly a compromise between the selective and the non-selective methods. It was for this reason that the less selectively mined material was used for the bulk tests.

A screen analysis (Table III) of one of the sub-samples indicated that the d_{50} size for this particular sample, less selectively mined, was 24 mm.

To simulate the wet screening on 0,5 mm slotted screens that is practised in commercial plants, the test samples were screened on 0,84 mm square mesh. The plus 300 mm and the plus 150 mm fractions, when examined for coking-coal carry-over, were found respectively to contain 0,14 per cent and 0,05 per cent run-of-mine coking coal at 12,0 per cent ash.

The project was divided into three sub-projects:

(1) removal of shale and recovery of plus 0,5 mm coking coal,
(2) recovery of minus 0,5 mm coking coal, and
(3) treatment of the middling to maximize the overall yield of coking coal.

REMOVAL OF SHALE

Rotary Breaker

One of the sub-samples was transported to Grootegeluk coal mine and broken in one of the rotary breakers. The screen opening, 170 mm, was not modified, even though initial drop tests indicated that a top size of

TABLE III

SCREEN ANALYSIS OF LESS SELECTIVELY MINED FEED

Screen size, mm	Fractional %	Cumulative %
+300	9,32	9,32
+150	9,26	18,58
+100	5,92	24,50
+ 75	3,50	28,00
+ 50	7,37	35,37
+ 25	13,42	48,79
+ 15,8	9,68	58,47
+ 6	13,63	72,10
+ 0,84	17,64	89,74
− 0,84	10,26	100,0
	100,0	

TABLE IV

SCREEN ANALYSIS OF THE UNDERFLOW FROM THE ROTARY BREAKER

Screen size, mm	Fractional %	Cumulative %
−150+125	0,61	0,61
−125+100	3,88	4,49
−100+75	9,40	13,89
− 75+50	7,47	21,36
− 50+30	12,35	33,71
− 30+20	10,59	44,30
− 20+15	5,48	49,78
− 15+12,5	3,89	53,67
− 12,5+5	13,85	67,52
− 5+3	5,70	73,22
− 3+0,84	13,75	86,97
− 0,84	13,03	100,00
d_{50} = 14,8 mm		

TABLE V

WASHABILITY OF THE SCREEN FRACTIONS OF THE PLUS 12,5 MM UNDERFLOW FROM THE ROTARY BREAKER (AS PERCENTAGES OF RUN-OF-MINE COAL)

Relative density	Screen fractions																Combined	
	−150+125 mm		−125+100 mm		−100+75 mm		−75+50 mm		−50+30 mm		−30+20 mm		−20+15 mm		−15+12 mm		−150+12,5 mm	
	Cum. floats		Cum. floats		Cum. floats		Cum. floats		Cum. floats		Cum. floats		Cum. floats		Cum. floats		Cum. floats	
	Yield %	Ash %	Yield %	Ash %	Yield %	Ash %	Yield %	Ash %	Yield %	Ash %	Yield %	Ash %	Yield %	Ash %	Yield %	Ash %	Yield %	Ash %
F1,30															0,01	6,44	0,01	6,4
1,30 to 1,35									0,01	15,0	0,03	12,6	0,05	12,6	0,06	11,6	0,15	12,4
1,35 to 1,40					0,01	18,6			0,12	19,5	0,21	16,8	0,22	16,9	0,21	16,1	0,77	17,0
1,40 to 1,45					0,06	22,0	0,06	23,0	0,41	21,6	0,59	20,4	0,52	20,1	0,45	19,5	2,09	20,5
1,45 to 1,50			0,01	24,1	0,09	23,8	0,13	24,9	0,74	23,8	0,95	22,8	0,79	22,4	0,66	21,8	3,37	22,8
1,50 to 1,55			0,04	29,0	0,15	26,3	0,23	27,4	1,01	25,7	1,24	24,7	1,01	24,4	0,82	23,6	4,50	24,9
1,55 to 1,60			0,07	32,4	0,23	28,9	0,34	30,1	1,25	27,5	1,51	26,5	1,18	26,0	0,97	25,4	5,55	26,8
1,60 to 1,65	0,01	39,4	0,08	33,4	0,32	31,6	0,47	32,5	1,56	29,7	1,76	28,2	1,34	27,6	1,09	26,8	6,63	28,8
1,65 to 1,70	0,02	40,0	0,09	34,4	0,42	33,9	0,62	35,1	1,82	31,6	2,04	30,2	1,53	29,5	1,22	28,4	7,76	30,8
1,70 to 1,80			0,14	39,2	0,63	38,7	0,86	38,6	2,32	35,4	2,55	33,9	1,84	32,7	1,44	31,4	9,80	34,4
1,80 to 1,90	0,05	50,2	0,19	43,8	0,77	41,8	1,00	40,7	2,76	38,6	3,03	37,2	2,16	36,0	1,65	34,3	11,61	37,7
1,90 to 2,00			0,25	49,7	0,95	45,6	1,12	42,6	3,13	41,2	3,49	40,4	2,45	39,0	1,86	37,3	13,30	40,7
S2,00	0,61	81,0	3,88	8,04	9,40	78,1	7,47	74,4	12,35	70,1	10,59	67,3	5,48	62,0	3,89	59,4	53,67	70,8
	0,61		3,88		9,40		7,47		12,35		10,59		5,48		3,89		53,67	

TABLE VI

WASHABILITY OF THE FRACTION BETWEEN −12,5 AND +0,84 MM OF THE UNDERFLOW FROM THE ROTARY BREAKER (AS PERCENTAGES OF RUN-OF-MINE COAL)

Relative density	Screen fractions						Combined	
	−12,5 +5,0 mm		−5,0 +3,0 mm		−3,0 +0,84 mm		−12,5 +0,84 mm	
	Cumulative floats		Cumulative floats		Cumulative floats		Cumulative floats	
	Yield %	Ash %	Yield %	Ash %	Yield %	Ash %	Yield %	Ash %
F1,30	0,15	6,8	0,47	5,8	3,89	4,8	4,51	5,0
1,30 to 1,35	0,58	11,1	0,81	8,0	4,75	6,2	6,15	7,0
1,35 to 1,40	1,37	15,0	1,22	11,6	5,60	8,0	8,19	9,7
1,40 to 1,45	2,13	17,8	1,54	13,9	6,12	9,3	9,79	11,9
1,45 to 1,50	2,81	20,0	1,81	15,8	6,59	10,5	11,21	13,8
1,50 to 1,55	3,39	21,9	2,03	17,4	6,99	11,7	12,41	15,5
1,55 to 1,60	3,92	23,7	2,23	19,0	7,36	12,9	13,51	17,1
1,60 to 1,65	4,32	25,2	2,37	20,1	7,59	13,7	14,28	18,3
1,65 to 1,70	4,74	26,7	2,52	21,5	7,82	14,6	15,08	19,6
1,70 to 1,80	5,43	29,5	2,77	24,0	8,36	17,0	16,56	22,3
1,80 to 1,90	6,14	32,4	3,01	26,4	8,79	18,9	17,94	24,8
1,90 to 2,00	6,83	35,3	3,22	28,7	9,16	20,7	19,21	27,2
S2,00	13,85	57,9	5,70	50,5	13,75	39,4	33,3	49,0
	13,85		5,70		13,75		33,3	

TABLE VII

WASHABILITY OF THE FRACTION SMALLER THAN −0,84 MM OF THE UNDERFLOW FROM THE ROTARY BREAKER (AS PERCENTAGES OF RUN-OF-MINE COAL)

Relative density	Cumulative floats	
	Yield %	Ash %
F1,30	4,1	3,4
1,30 to 1,40	6,2	5,6
1,40 to 1,50	7,2	7,6
1,50 to 1,60	7,7	9,0
1,60 to 1,70	8,3	11,2
1,70 to 1,80	9,0	14,8
1,80 to 1,90	10,2	21,0
1,90 to 2,00	11,2	25,9
S2,00	13,0	33,9

TABLE VIII

WASHIBILITY OF THE RAW COAL AFTER THE ROTARY BREAKER

Relative density	Cumulative floats	
	Yield %	Ash %
F1,30	8,6	4,2
1,30 to 1,40	15,1	8,4
1,40 to 1,50	21,8	13,1
1,50 to 1,60	26,8	16,8
1,60 to 1,70	31,1	20,2
1,70 to 1,80	35,4	23,7
1,80 to 1,90	39,8	27,6
1,90 to 2,00	43,7	31,0
S2,00	100,0	58,7

below 80 mm was ideal for this coal. In this test the breaker was not very successful as a shale-removing device, since it removed only 0,82 per cent of the run-of-mine coal, which contained 0,003 per cent of the run-of-mine coking coal with 12 per cent ash.

Although the tests undertaken so far had indicated the ideal top size and partition density for maximum waste removal in the primary heavy-medium separation washer, confirmation was necessary. The breaker underflow was therefore screened, and float and sink analyses were carried out on the size fractions (Tables IV to VIII).

The decision to primary dry screen at 12,5 mm was taken more for convenience, but it was reasoned that 12,5 mm was the minimum size that coal of this nature should be dry-screened especially since it was possible that moisture would be present in the feed, some of the mining being below the water table.

Heavy-medium Separation

Sample of Underflow from the Rotary Breaker

Some of the decisions that had to be taken right at the outset of the investigations concerned the ideal top size of the feed to the primary and secondary heavy-medium separators, and the cut-point relative densities for maximum shale discard without undue coal loss in the discard.

To determine the partition density for the removal of the shale in the plus 12,5 mm fraction, the relative density fractions 1,70 to 1,80, 1,80 to 1,90, and 1,90 to 2,00 were combined separately, and crushed to minus 1,0 mm to liberate as much as possible of the coking coal that was present in very narrow bands at these high relative densities. Each density fraction was subjected to detailed float and sink analysis so that the amount of 12 per cent ash coal present could be determined (Table IX). The results indicate that 1,80 is a suitable partition density for the removal of shale.

The results confirmed that liberation occurred and that the screen openings of the rotary breaker need not be reduced any further for this coal.

Bulk Test[2]

One complete sub-sample was delivered to the Fuel Research Institute pilot plant for the purpose of

(i) testing the process so far indicated,
(ii) obtaining information for the sizing of equipment, and
(iii) producing enough minus 0,5 mm raw coal for subsequent treatment of the fines.

Fig. 5 shows the material flow and mass balance.

The preparation of the feed differed from the tests featuring in Table IX in that the raw coal was crushed to a nominal size of 50 mm, i.e. the crusher was set at 50 mm. Liberation was excellent, which was confirmed by sampling the discard 1,8 relative density fraction for pieces containing visual coal. This hand-sorted material was crushed to minus 0,5 mm and subjected to float and sink analysis. Only 0,15 per cent of the fraction could be recovered at 12,0 per cent ash. This also indicated that the 1,8 relative density fraction used for the de-shaling of the run-of-mine coal gave an effective and efficient separation.

Separation by Jig Washer

To obtain information from geological exploration, it is standard practice at Iscor to selectively sample the coal zones in a drill core, crush to minus 13 mm, screen at 0,5 mm, and subject the plus 0,5 mm to float and sink analysis. Accordingly, the Soutpansberg drill cores were float-and-sink analysed up to 1,6 relative density to obtain the yield of coking coal at 12 per cent ash.

Mr D. W. Horsfall, of Anglo American Corporation, extrapolated the washability obtained from the drill cores, and predicted the washability of the raw coal from this rather meagre information (Fig. 6). The results indicated that the separation at 1,8 relative density was not a difficult separation and, on the strength of this, it was decided to wash a sub-sample of the underflow raw coal from the Bradford Breaker in a jig washer.

The Blantyre jig at the pilot plant of the Fuel Research Institute was modified to treat a sample of raw coal containing a high percentage of sinks at a relative density of 1,8 to 1,9. The exercise was undertaken by Messrs Norton Tividale South Africa (Pty) Ltd, who also computed the results.

The washability of the raw coal fed to the jig washer

TABLE IX

YIELDS OF 12 PER CENT ASH COAL IN THE PLUS 1,70 RELATIVE DENSITY FRACTIONS (AFTER CRUSHING TO MINUS 1,0 MM)

Sample	Relative density fraction	Yield of 12% ash coal as % of run-of-mine coal
−150 mm+12,5 mm	1,70 to 1,80	0,26
	1,80 to 1,90	0,33
−12,5 mm+0,84 mm	1,70 to 1,80	0,06
	1,80 to 1,90	0,10

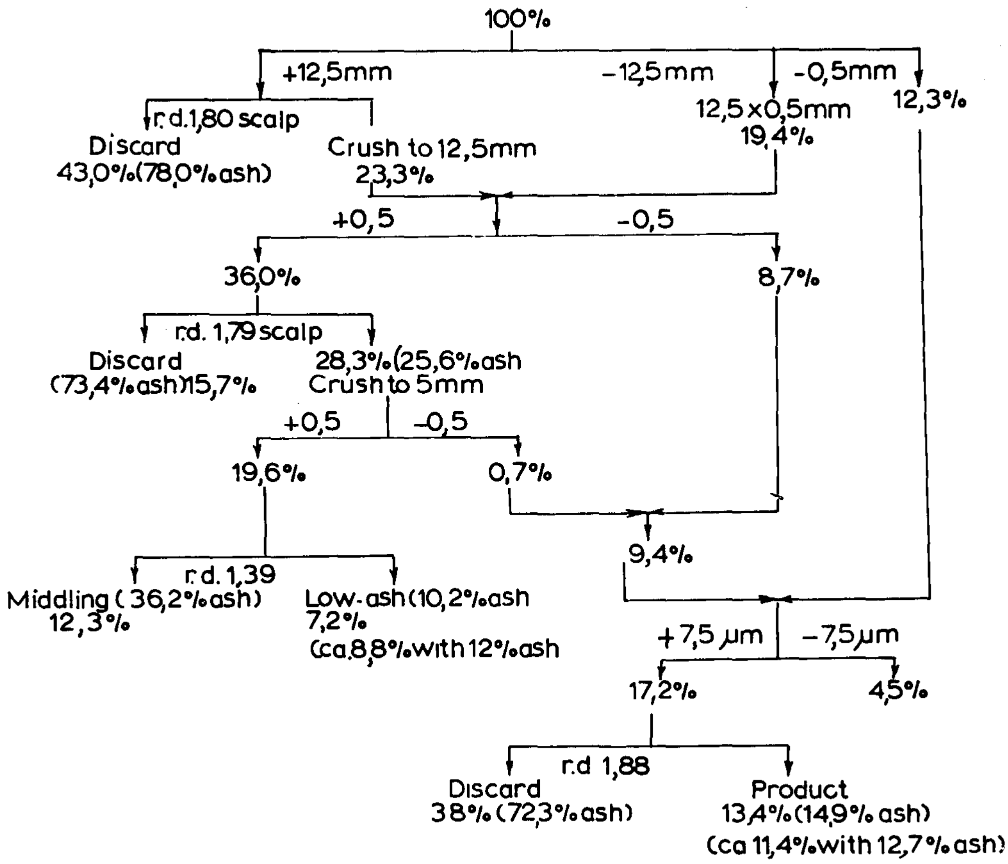

Fig. 5—Material flow and mass balance in pilot-plant tests conducted at the Fuel Research Institute

(Fig. 7) showed that the separation at a relative density of 1,8 was not as simple as predicted originally. The efficiency of separation that was obtained at a very low feed rate was more or less in agreement with that given in the literature for modern jigs, viz. an Epm of approximately 0,08. An increase in the feed rate, however, resulted in a drastic reduction in the efficiency of separation, viz an Epm of 0,53. It must be borne in mind that the jig tests were undertaken by highly skilled operators, which in itself could be a problem in practice.

The conclusion drawn from the results was that the jig washer, in terms of separating at a very high density and with very high discards, is not suited to this application.

RECOVERY OF MINUS 0,5 MM COKING COAL

During the washing of the bulk sample of run-of-mine feed at the Fuel Research Institute, the minus 0,5 mm natural fines and the minus 0,5 mm produced by the crushing of the primary coarse heavy-medium separator floats were intimately mixed and sub-divided into several representative sub-samples for comparative process evaluation.

Froth Flotation

Standard froth flotation was carried out on a sub-sample using power paraffin as collector and Montanol as the frother. This material floated without any trouble, and variables, such as frother concentration and control of the air and stirrer speeds, were not as critical as with some other, more difficult coals.

The yields and efficiencies obtained from the double-stage flotation (Table X) were very good, considering that the feed was unclassified. However, it is expected that the multistage addition of reagents would improve the yields and efficiencies of single-stage flotation circuits. Laboratory work has confirmed that improved yields can be obtained from more difficult coals by the multistage addition of reagents. The results reported here are very much in line with the work reported by Firth *et al.*[3].

Heavy-medium Cyclone Cleaning[4]

A sub-sample of minus 0,5 mm feed was treated by the Fuel Research Institute in a 150 mm dense-medium cyclone plant. The combined minus 0,5 mm fines were classified at 0,075 mm before the dense-medium separa-

TABLE X

FLOTATION YIELDS FOR INCREASING PRODUCT ASH

Product Ash %	Washability %	Single stage, %		Double stage, %	
		Yield	Efficiency	Yield	Efficiency
8	41			30,0	73,2
10	48	10,0	20,8	40,0	83,3
11	52	28,0	53,9	44,0	84,6
12	55	38,5	70,0	47,5	86,4
14	60	48,0	80,0	55,5	92,5
16	65	57,0	72,3	62,0	95,4
18	69	63,5	92,0		

tion. The washability of the fraction between 0,5 mm and 0,075 mm is given in Table XI.

From the washability results the partition density appears to be rather high, and, since there is a differential between the operating density and the partition density in a dense-medium cyclone, it was thought that this type of washer would not be very efficient at relative densities around 1,8. As Fig. 7 shows, efficiencies of more than 98 per cent were achieved.

A direct comparison cannot be made of the total yields of 12 per cent ash coking coal recovered by the two processes (flotation and heavy-medium cyclone separation), since the ash of the feed differed, being 33,9 per cent for the feed to heavy-medium separation and 36 per cent for that to froth flotation. If this difference is taken into account, the yields of the two processes are close, and a choice between the two would have to be based on practical and economic considerations.

It should be borne in mind that a certain amount of desliming had already taken place during the production of the feed samples, which would favour the heavy-medium cyclone separation. A combination of the two processes could be the best solution.

TABLE XI

WASHABILITY OF THE FRACTION BETWEEN −0,5 MM AND 0,075 MM FROM THE 150 MM CYCLONE FEED

Relative density	Cumulative yield %	Cumulative ash %
F1,30	30,9	3,0
1,30 to 1,35	41,5	4,5
1,35 to 1,40	48,4	5,7
1,40 to 1,45	52,9	6,8
1,45 to 1,50	56,5	7,8
1,50 to 1,55	59,1	8,7
1,55 to 1,60	61,4	9,5
1,60 to 1,65	63,4	10,2
1,65 to 1,70	65,3	11,0
1,70 to 1,75	67,1	11,8
1,75 to 1,80	68,8	12,6
S1,80	100	30,8

Water-only Cyclone Separation of the Minus 0,5 mm Fines

Water-only cyclones can be used for high-density separation, especially where there is little material close to the density required.

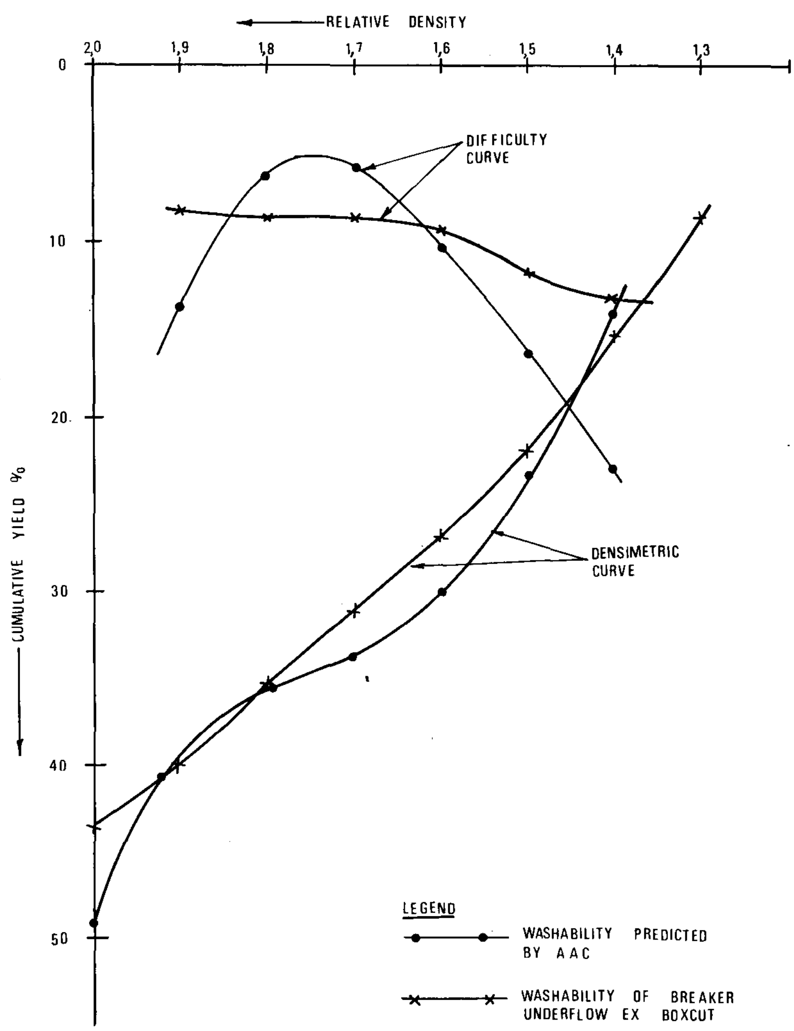

Fig. 6—Washability curve of raw coal (prepared by D. W Horsfall of Anglo American Corporation)

Washability data were supplied to Stamicarbon to enable them to assess the feasibility of using DSM water-only cyclones for the production of 12 per cent ash coking coal. Their study indicated that the possibility does exist, but this prediction has not yet been confirmed by testwork done elsewhere.

Nortons Tividale South Africa (Pty) Ltd conducted a series of tests using the 200 mm Visman compound water cyclone installation at the Fuel Research Institute. The feed sample was a sub-sample of the bulk sample of minus 0,5 mm fines used for froth flotation and heavy-medium cyclone separation.

The results indicate that the Visman compound water cyclone is not suitable for the treatment of the Soutpansberg minus 0,5 mm material. The best result was as follows:

	Overflow	*Underflow*
Mass, %	52,1	47,9
Ash, %	18,3	58,0
Raw ash (calculated), %	37,3	

These results are in line with water-only cyclone testwork conducted at Iscor.

TREATMENT OF MIDDLINGS

The coal in the Soutpansberg coalfield has a high vitrinite content with different proportions of inter-banded mineral material. Petrographic studies have indicated that the high-vitrinite coal can be liberated by crushing to minus 0,21 mm. Washability studies of the middlings crushed to minus 0,21 mm indicated that approximately 3,6 per cent of the run-of-mine coal could be recovered at 12 per cent ash. This yield would make a substantial contribution towards the overall recovery from this mine. Subsequent work indicated that a much finer grind was necessary on certain samples to achieve liberation.

The Fuel Research Institute, during their investigations of the bulk sample, crushed a sample of middling coal to minus 0,8 mm and screened this material at 0,075 mm. The plus 0,075 mm fraction was subjected to detailed float and sink analysis and the washability evaluated. This exercise indicated a liberation of about 18 per cent of the middlings with an ash content of 12 per cent. This represents about 2 per cent of the run-

TABLE XII

RESULTS FROM SOUTPANSBERG BLOCK C MIDDLINGS

Wet-milling time min	Size consist		Yield of clean coal 12 % ash washability
	% +0,074 mm	% −0,038 mm	
1	83,1	—	7
2	—	—	26
4	26,7	48,0	30
8	2,0	75,4	41
16	—	—	41

TABLE XIII

FLOTATION RESULTS

Product	Cumulative yield %	Ash	
		Fractional %	Cumulative %
Concentrate	26,7	12,1	12,1
Intermediate product 2	45,0	25,5	17,5
Intermediate product 1	64,9	42,2	25,1
Tailing	100		

Organic efficiency = 64,6 % at 12,0 % ash.
Coal yield at 12 % ash content = 3,3 % of run-of-mine coal.

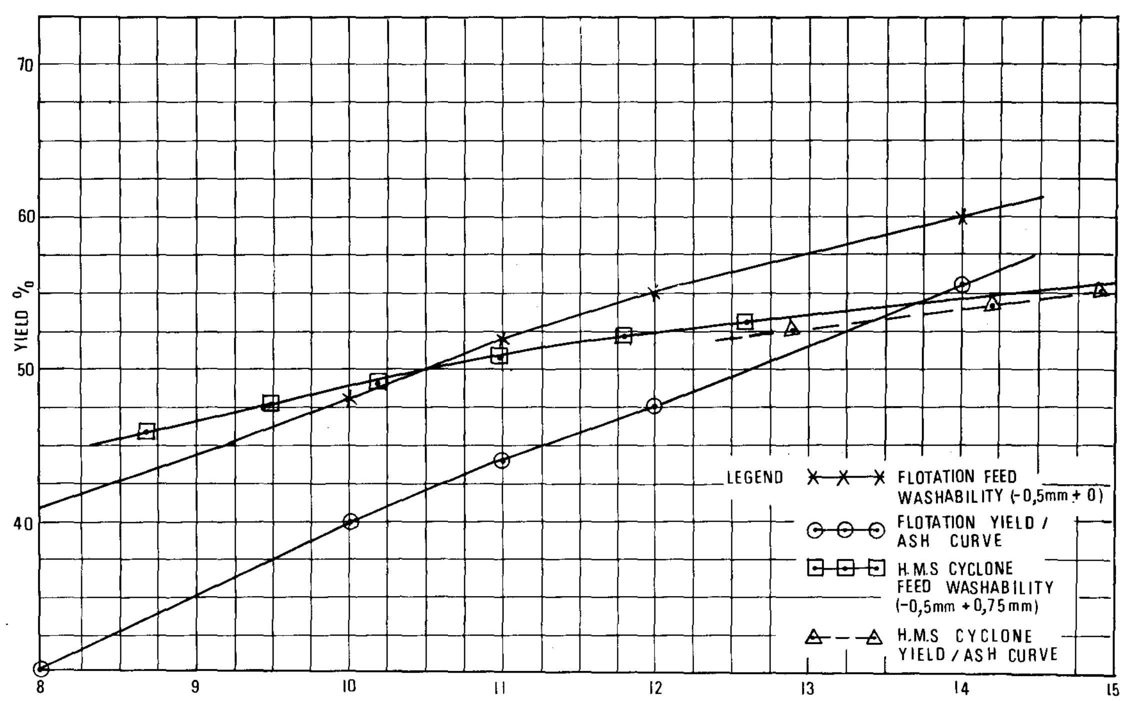

Fig. 7—Family of curves showing the relationships between the yields obtained by flotation and heavy-medium cycloning compared with the theoretical values for minus 0,5 mm material

TABLE XIV

FLOTATION RESULTS

Product	Cumulative yield %	Ash	
		Fractional %	Cumulative %
Concentrate	22,5	10,8	10,8
Intermediate product 2	35,7	22,2	15,0
Intermediate product 1	56,6	36,1	22,8
Tailing	100	49,0	34,1

of-mine coal, compared with 3,6 per cent when crushed to minus 0,21 mm unclassified.

Preliminary froth-flotation studies in the laboratory on the material between 0,21 mm and 0 mm were not successful. Only low yields at a high ash content could be achieved, being 2,56 per cent of the run-of-mine coal at 17,5 per cent ash.

Comminution studies indicated that the liberation was complete in the region of 0,038 mm (Table XII). Dry milling to minus 0,044 mm followed by froth flotation still gave a poor yield at high ash.

In an attempt to alter the response of the coal particles to froth flotation, wet milling to minus 0,038 mm and subsequent flotation were tried. Although this technique had been used successfully in investigations on the flotation of other minerals, the need to float micro-fine coal had not arisen. In the case of the coal from Soutpansberg Block C, it was necessary to recover this fine coal from the middling fraction.

A sample of wet-milled material of approximately 90 per cent minus 0,074 mm was subjected to laboratory froth flotation (Table XIII). The response of this material to three-stage flotation was very good in view of the disappointing results previously achieved with the coarser material.

The aeration rate was 0,41 l/min, and the impeller speed was 1100 r/min. A pulp density of 17 per cent (m/m) was used, and the cell retention time was 25 min. The pulp was not conditioned before flotation. The reagent addition was rather high, being, 1,7 kg of power paraffin and 164 g of Montanol (Hoëchst) per ton.

To investigate the effect of flotation rate on the pulp, the aeration rate was increased to 6 l/min. As a result, the amount of collector could be reduced to 1,24 kg/t, and the quantity of frother had to be drastically reduced to 0,013 kg/t. The results are summarized in Table XIV.

The low ash yield achieved was very gratifying since it has been found that the yields of froth flotation at low ash contents can be disappointing.

FURTHER INVESTIGATIONS

Laboratory filtration tests indicate that micro-fine coal can retain as much as 40 per cent moisture. The customer, in this case the coke ovens, prefers to receive products with a total moisture content of less than 8 per cent. As the moisture content of the minus 0,5 mm fraction in the final product can be as high as 50 per cent, dewatering of the fines has become a new problem area that requires investigation.

Centrifuges can be used to reduce the moisture of the classified minus 0,5 mm product to total moisture contents of as low as 15 per cent. However, the micro-fines will have to be treated differently and processes such as the following are being considered:

(1) the utilization of filter aids such as the addition of oil to the slurry prior to the filters,

(2) filtration in conjunction with briquetting,

(3) drying of the product in suitable furnaces.

Another factor that has to be taken into account is the loss of fines in the coke-making process. This, in conjunction with the lower size limit of coal particles that contribute to the coke-making process, must also be investigated.

CONCLUSION

The coal-preparation route for maximum recovery leads to the production of micro-fine coal that has been discarded up to now. Studies of oil agglomeration have proved that micro-fines can be recovered from slurries, but this method has not yet progressed beyond pilot-plant production units.

The success achieved in the laboratory with the froth flotation of micro-fines is very encouraging, but the problem of moisture reduction requires urgent attention.

The reasons for the improved response of the material to froth flotation as a result of wet milling can only be speculated on. The theories propounded are being tested, and could lead to interesting developments.

ACKNOWLEDGEMENTS

This paper is presented with the permission of Mr M. J. Deats, Divisional General Manager (Mining), South African Iron & Steel Industrial Corporation, Limited. The author expresses his thanks to his colleagues at the pilot plant of the Research and Process Development Department of Iscor for their co-operation and assistance with the experimental work, particularly the sterling work done by Mr P. H. Botha, who was responsible for the flotation circuits and results.

REFERENCES

1. Steyn, J. G. D. South Africa may face a shortage of prime coking coals. *Metals Miner. Process.*, vol. 1, no. 11. Jun. 1979. pp. 6–15.
2. Fuel Research Institute. Report no. 14. Pretoria, the Institute, 1979.
3. Firth, B. A., Swanson, A. R., and Nicol, S. K. 1979. Flotation circuits for poorly floating coals. *Int. J. Miner. Process.*, vol. 5. 1979. pp. 321–334.
4. Fuel Research Institute. Report no. 40. Pretoria, the Institute, 1979.