BENEFICIATION OF WATERBERG COAL

Berrin Eroğlu

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A dissertation submitted to the Faculty of Engineering, University of the Witwatersrand, Johannesburg, in fulfilment of the requirements for the degree of Master of Science in Engineering

Johannesburg, 1992

DECLARATION

I declare that this dissertation is my own, unaided work. It is being submitted for the Degree of Master of Science in Engineering in the University of the Witwatersrand, Johannesburg. It has not been submitted before for any degree or examination in other University.

Signed _____

Dated this 18th day of DEC 1992

ABSTRACT

Modern methods of mechanised mining and the necessity for the utilization of total reserves have caused the inclusion of more and more impurities in run of mine coal. This fact, together with the limited supply of naturally clean coal for gasification, liquefaction and metallurgical purposes, has made some form of beneficiation obligatory at many mines not only in South Africa but also in many other countries.

One of the South African Coalfields, Waterberg, contains the continent's largest reserves (approximately 46% of South African known reserves). At the Grootegeluk Coal Mine, approximately 15 m tons of coal per annum are mined by opencast methods. The coal is characterised by containing a high proportion of reactive macerals. The Waterberg Coalfield is currently supplying coal for coke manufacture and middlings for power generation. This coal could also be used for other markets, as Waterberg coal is low in oxygen, contains up to 30% volatile matter. Because it contains 90% vitrinite, it is suitable for direct liquefaction, and possibly coal-water mixtures. However the yield of coal suitable for coking or liquefaction (approx 10% ash) is only 12%, with another 24% of 35% ash coal, currently used for power generation. These yields render mining generally uneconomical if making a simple product.

The objective of this project is to ascertain whether the yields of washed coal from the Waterberg Coalfield might be increased by using comminution. Thereafter appropriate beneficiation techniques might be employed on different size fractions.

Liberation, float and sink, froth flotation and oil agglomeration processes were examined to identify the best way of treating the coal. Work was carried out on the existing clean coal, middlings and discard fractions. The major objective was to optimise the yield of 10-15% ash coal.The results of the experiment indicate that it is possible to obtain low ash coal from middlings, and middlings from

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discard for power station. The capital and operating costs for improved new plants are calculated by using available factorised data.

The results of experiments on both middlings and discards indicate that yields are significantly higher than those currently obtained, but the cost of obtaining such enhanced yields can be too high for normal commercial application.

DEDICATION

In memory of my father

Mustafa Demir Cinel

1930 - 1979

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1 INTRODUCTION

1.1 Background

The energy requirements of the earth's population are almost incalculable. Since the industrial revolution energy requirements have grown and the earths natural resources are being depleted at a growing rate. We must therefore develop technologies and skills to extend the life of energy sources.

Energy Reserves

The energy sources of the earth have been classified into five groups. namely;

- Solid fuels
- Liquid Fuels
- Natural gas
- Hydro-electricity
- Nuclear electricity

This project deals only with solid fuels (for more information see **appendix A**). The major deposits of coal resources and reserves are concentrated in a number of countries in the northern hemisphere. They are,

- The Soviet Union,
- The United States of America,
- The People's Republic of China, and
- a number of European countries

There are only three countries which have major coal deposits in the southern hemisphere, namely;

- Australia,
- The Republic of South Africa(see **appendix B**), and
- Botswana (World Energy Resources, 1985)
- OPEC

In 1973, the world's leading oil suppliers formed The Organization of Oil Exporting Countries (OPEC). They were able to increase the oil prices to unprecedented levels because of the crisis in the Middle East and the limited

number of oil exporters. Importing countries had no option but to accept the new inflated price. Oil importing countries made the following decisions to maintain their existing and future energy requirements:

- oil dependent technologies were converted to coal-fired technology,
- many industrialised countries began to import coal in order to extend their own reserves,
- the search for alternative fuels was accelerated, involving the more specialised uses of coal in the chemical and conversion fields initiated.

These were necessary because of the energy crisis and the limited number of potential energy exporters around the world. Thus the importance of South Africa and the other coal producers increased rapidly and the world markets for coal were explored.

Coal in South Africa

Today, South Africa is one of the most important steam coal exporting countries in the world, and is a world leader in the use of coal as a chemical feedstock and for synfuel production. South Africa has one of the world's highest dependencies on coal for its energy requirements.

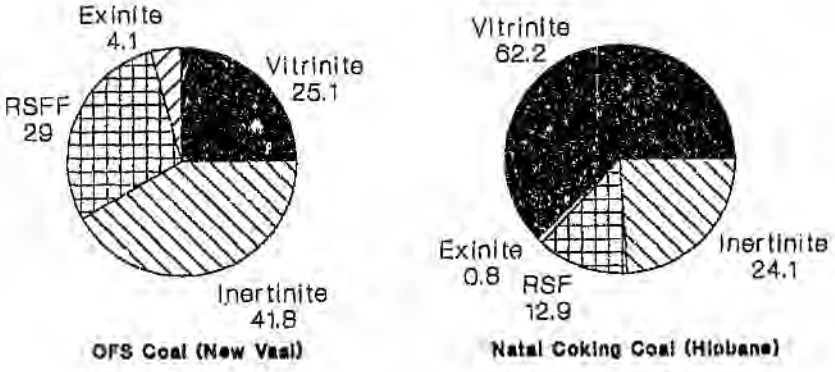
However, in promoting the use of coal, beneficiation technologies to upgrade coal assume growing importance. The special needs of South African coal in this context are explained by discussing the mode of formation.

Geologically coals of the southern hemisphere, were mainly formed in Permian times and those in the northern hemisphere during the Carboniferous period. Comparing the coal-forming conditions of these eras will show the importance of beneficiation for South African coals. When coal was formed, South America, Africa, Australia and India formed one continent called "Gondwanaland"; whilst Europe, North America, Asia formed another called "Laurasia". Laurasian coal swamps experienced lush tropical conditions and a swampy environment; as a

result of these conditions Laurasian coal is characterised by high proportions of vitrinite, a fair proportion of exinite, low inertinite and a small proportion of finely disseminated mineral matter within the coal matrix. Gondwana coal swamps grew in cold to cool temperatures associated with the waning of a massive ice age. Consequently Gondwana coals are characterised by lower proportions of vitrinite and exinite, and much higher proportions of inertinite and mineral matter (washed in from surrounding glaciers) than their Northern counterparts. In addition to these, the earth cover was thinner in Gondwanaland and the rank of the coal is, therefore, generally lower than that of Laurasian coal (Falcon, 1986; Van Der Walt, 1984). The following pie charts show the petrographic analysis of different coals, both in South Africa and overseas.

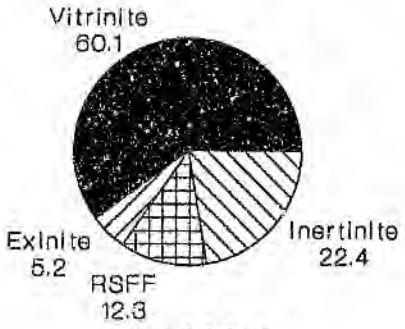
MACERAL COMPOSITION OF SA COALS

OFS coal and Natal coking coal

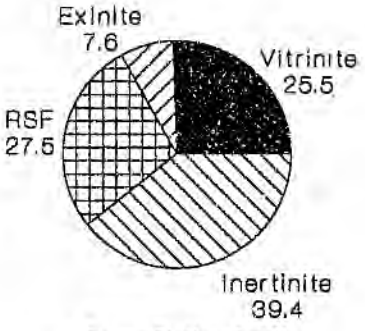


MACERAL COMPOSITION OF SA COALS

Number 2 seam, lac and pse



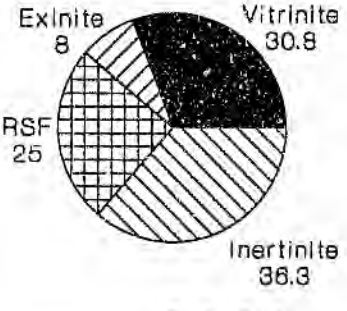
Low Ash Coal (Kleinkopje)



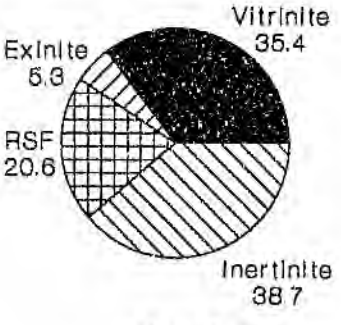
Power Station Smalls (Kleinkopje)

MACERAL COMPOSITION

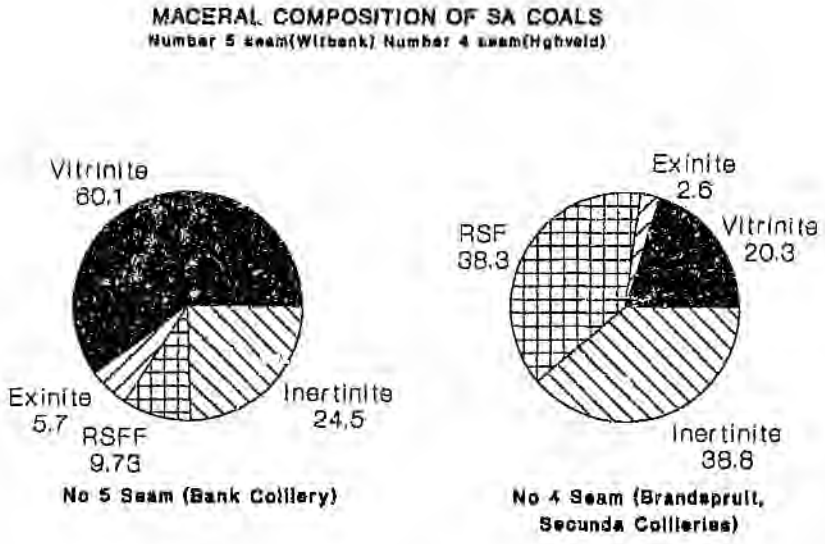
No 2 Seam Power Station Coals



Duvha Coal



Arnot Coal



(Horsfall,1992)

The reserves situation of South Africa also make beneficiation more important,especially to determine the optimum way of using these reserves. Moreover, the better quality reserves were exploited in the past. Using beneficiation techniques is ,therefore, important to enable the optimum use of poor quality reserves and discards to extend the life of reserves in South Africa and to make SA coal acceptable in the international markets (Alber, 1987).

Waterberg Coalfield

South Africa has 18 principal coalfields. One of them is the Waterberg coalfield which contains the continent's largest reserves (approximately 46% of South African in situ known coal reserves. This comprises 28% of South Africa's economically mineable reserves). It covers 730 square miles in the North Western Transvaal. Subsurface data from boreholes provided the information for

determining the coalfield stratigraphy.

The Waterberg coalfield occurs in the Karoo Sequence and forms a part of the Ecca group (De Kun, 1965). It consists of coal-rich zones interspersed with mineral rich zones in the upper regions (which are already being mined) and dull coal in the middle regions.

The upper region of the Ecca group in the Waterberg coalfiel' has an average thickness of 60 m and consists of seven zones(numbered 11--5). Each zone consists of a different number of samples which has various coal and mineral matter layers. It may, therefore, be expected the organic i.e coal component is unusual in South African coals in being generally >85% vitrinite. Thus in going from a low ash of say 10% to a higher ash content of 15% the maceral composition of the coal remains almost unchanged, i.e. the percentage of reactives will remain >85%. The amount of such organic material rises or falls in proportion to the ash content.

At Grootegeluk coal mine, approximately 30 tons of coal per annum are mined by open cast methods. This was achieved in two phases each of about 15 m ton per annum. Modern methods of mechanised mining and the necessity for the utilization of total reserves have caused the inclusion of an increasing amount of impurities in run of mine coal. This, together with the limited supply of naturally clean coal for gasification, liquefaction and metallurgical purposes, has made some form of beneficiation obligatory at many mines. The total coal preparation plant in Grootegeluk, at 6 000 tons per hour , is one of the largest in the world. The present beneficiation plant treating coal used in the firs phase of project yields 12% coal with an ash content of 10 - 11%. 24% middlings is obtained with an ash content of 33 - 35%. The discard of 65% has an ash content of up to 80%. The present beneficiation plant flowsheet is shown in **figure 1.1***. The beneficiation plant for the second phase also treating 15 m tpa does not form a part of this study.

Grootegeluk, currently the only mine in the Waterberg coalfield supplies coal for coke manufacture and middlings for power generation. The coal could also be used for other markets, as Waterberg coal is low in oxygen, contains up to 30% volatile matter, and contains 90% vitrinite. If of low enough ash, the coal has been shown to be highly suitable for direct liquefaction, and, may be useable to produce coal – water mixtures.

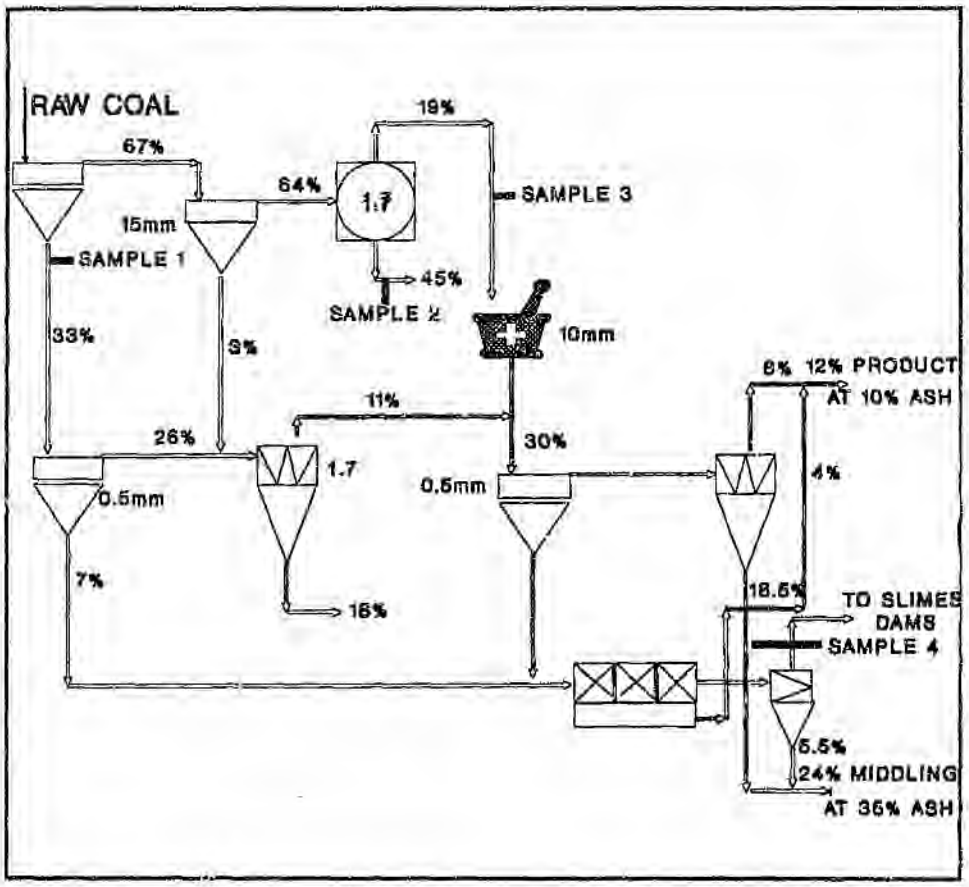


Figure 1.1 The present beneficiation plant

* This flowsheet shows the system of beneficiating Waterberg coal when this project was started. Today, the flowsheet includes spiral plants to treat –1 mm size fraction.

1.2 Objectives

The aims of this project could be summarised as follows;

– to ascertain whether the yields of washed coal from the Waterberg coalfield might be increased by using comminution. This should achieve liberation of the mineral species so that separation of the desired minerals from the coal could be effected. Beneficiation techniques might be employed on size fractions different to those currently employed, which could lead to an increase in the production of low ash coal and middlings, and a reduction in the percentage of discard.

– specifically to ascertain whether the currently discarded fraction can yield additional saleable products and whether this fraction could be sold as power station coal. This treatment could similarly be applied on the currently available middlings product to obtain enhanced yields of high quality products.

– specifically to ascertain whether the coal of ash 10–15%, suitable for direct liquefaction or for coke production, could be produced from the 35% ash middlings fractions currently being sent to Matimba.

The above listed objectives are illustrated by **figure** 1.2.

– to estimate the budget capital and operating costs for the resulting proposed systems by using available factorised data.

For these purposes, liberation, float & sink, froth flotation and oil agglomeration processes were examined to identify the best way of treating the coal. Froth flotation and float & sink processes on coal finer than 0.1 mm in size were tested at Wits University. Oil agglomeration tests on the same size range were performed at the University of Cape Town. Washability tests on coarser sizes were carried out by Richlab.

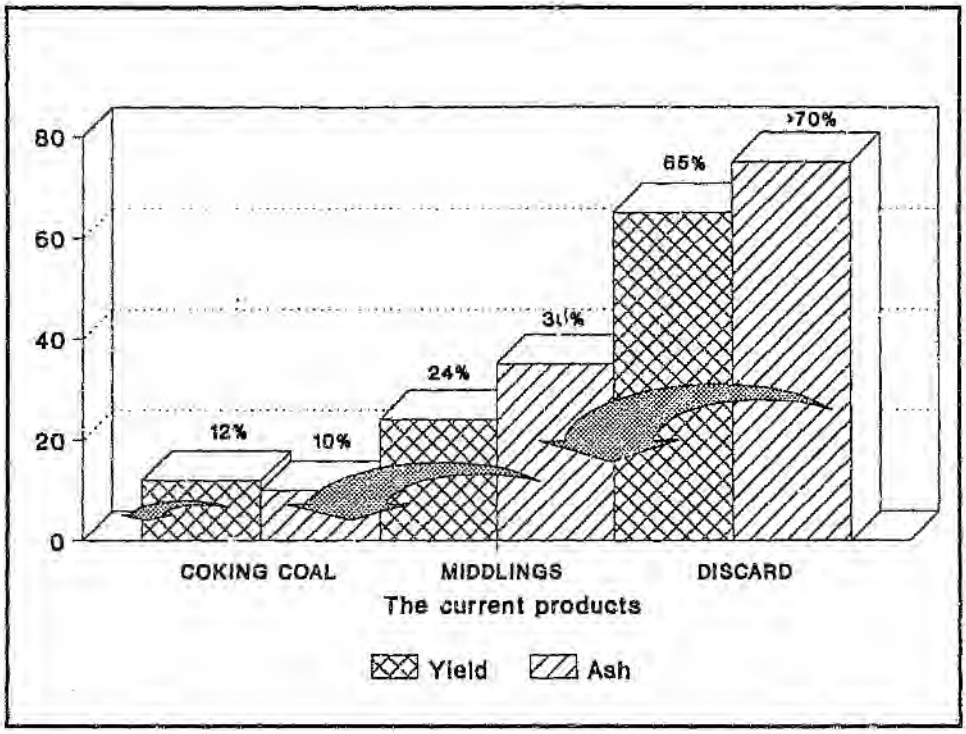


Figure 1.2 The illustration of the aims listed above

2 LIBERATION

All individual coal particles contain various minerals as impurities. Some were intimately mixed with coal during its formation, syngenetic mineral matter were deposited in cleats and epigenetic mineral matter fissures after the coal itself was formed. The minerals present in cleats and fissures may be liberated by comminution and then rejected by various methods of mineral processing.

The objective of liberation is to free the desired minerals so that their separation can be attained. If such an aim is achieved, then not only is energy saved by the reduction of the amount of fines produced, but any subsequent separation stages become easier and cheaper to operate (Wills, 1979). It is therefore necessary to determine the degree of liberation. Mr Horst in 1985, defined the degree of liberation as the percentage of that mineral occurring as free particles in relation to the total quantity of that mineral occurring in both the free and locked forms. This is not an easy definition to apply to coal. Perhaps it should be the degree of dissociation leading to ash levels and yields.

2.1 Reasons For Size Reduction

Size reduction is practised for the following purposes,
- to reduce run of mine lumps in order to make for ease of handling and cleaning.
- to liberate minerals from coal
- to achieve the maximum particle size in the finished product.

The reasons for the increased interest in liberation studies on coal may be summarized as follows;
- South Africa depends almost entirely on coal for its energy requirements. South Africa is also currently one of the most important steam coal exporting countries in the world, and a world leader in the use of coal as a chemical feedstock and synfuel production. Consequently the maximum use must be made of the

remaining reserves of South Africa.

Most of current South African coal production does not need liberation before beneficiation as it is sold unbeneficiated. However this country's expansion into international markets, and the rapid evolution of new technological uses of coal makes the identification of liberation characteristics and improving the efficiency of beneficiation on South African coal very important. It may also be a means of keeping cost under control, but in this instance such an aspect is not involved.

– The increasing quantity of fines in run of mine coal encourages investigation into their beneficiation. These fines are produced by natural degradation and increasingly mechanised mining methods. Most fines are now beneficiated but ultra fines are usually discarded because of their high ash content, even though they also contain a large proportion of valuable low ash material. This low ash material could be recovered by identification of the liberation characteristics.

– The beneficiation of a coal is dependent on its liberation characteristics. The amount of near density material in the feed causes great difficulty in coal processing plants. For instance, if the feed were composed entirely of pure coal at a S.G of 1,3 and shale at a density of 2,7, then the separation would be easily carried out over a wide range of operating densities. If however, the feed consists of appreciable middlings, and much material is present very close to the chosen separating density, then only a small variation in this density will seriously affect the yield and ash content of the product (Wills, 1988). If the mineral matter is liberated, the amount of near density material should decrease with particle size.

Although liberation produces more fines and it is an expensive process, it needs to be studied in the wider context of improving beneficiation.

2.2 Measurement of Coal Liberation

The increased use of modern methods of mechanised mining has given rise to increasing quantities of fines. Fine coal beneficiation has therefore assumed greater importance and prompted the application of various liberation techniques.

The technologies used for the identification of the liberation characteristics of coal are discussed in this section. They can be categorised as follows;
- physical properties
- surface properties
- optical properties

2.2.1 Separation by Density Differences

Float and sink analysis is the separation of coal into a series of relative density fractions in the laboratory using the difference in the relative density to differentiate between pure coal and the impurities associated with it. Float and sink is followed by analysis of the different fractions. Ash content and calorific value are the main parameters determined (Horsfall, 1980).

Float and sink analyses are carried out for the following reasons:
- Determining of washability characteristics to ascertain the most suitable way to beneficiate a coal.
- Establishing the efficiency of separation,

Float and sink analyses of washery products are required to evaluate the efficiency of separation.
- Plant control,

For day to day plant control, float and sink separations are carried out on the products at the required density (Fourie, 1978).

Float and sink analysis is carried out by consecutively immersing the sample of

coal in liquids of different relative density and by recovering either the float or the sink fraction at the relative densities (Strecher, 1974). Depending on the coal being analyzed, one would either start at the highest relative density and work through the different intervals down to lowest relative density, or vice versa. A general principle is that the relative density chosen should enable as much of the sample as possible to be removed in the first stage i.e discard should be initially floated at the highest relative density used, and clean and raw coal at the lowest relative density. For washability the intervals are usually 0.05 over the range 1,30–1,80.

The fluid to be used in float and sink analysis depends on the relative densities required. It may be either an organic liquid or an aqueous solution of inorganic salts, or suspensions of high density solids in water or high density solution. The choice of medium is also affected by the:

– amount of coal. This affects the cost of the analysis,

– size of coal. Fine particles require media of low viscosity

– If particulate material is to be determined, the liquid containing the same element should not be used (South African Bureau of Standards).

Organic Liquids

Relative densities of some commonly used organic liquids,

Name of The Organic Liquid	**Relative Density**
Petroleum Spirit	0.73
White Spirit	0.77
Toluene	0.87
Carbon tetrachloride	1.60
Perchlorethylene	1.61
Bromoform	2.79
Tetrabromomethane	2.96

When relative densities <1,60 are required, the medium may either be diluted with

petroleum spirit, white spirit or toluene. Bromoform or tetrabromoethane may be added to attain a higher density. The vapours of organic liquids may present a serious health and fire hazard. The liquids should only be used in a laboratory well equipped for the removal of fumes.

Inorganic Liquids – Solutions of Salts

Inorganic liquids may be used in the density range 1,3 to 1,8. They are commonly employed for sizes in excess of 0.5 mm, because the viscosity of more concentrated solutions affects the separation for small material.

Zinc chloride is the most commonly used inorganic salt for large scale work. It has several disadvantages i.e. zinc chloride solutions are corrosive, hence care should be taken in the choice of container used in the test, and skin contact should be strictly avoided. A further disadvantage is that the pores of the sample frequently became permeated by the zinc chloride solution. This may introduce mass determination errors.

The time required for effecting complete separation of floats from sinks depends on:

- the size of the particles in the sample,
- the viscosity of the separating medium,
- the number of particles with a relative density close to the density of separation.

Large coal usually separates quickly, a longer time is required to effect separation for smaller particle. (SABS method 939)

2.2.2 Surface Properties

Froth Flotation is especially useful for beneficiating ultra fines (i.e. <0.15 mm); although its use is often extended to sizes >0.15 mm, even up to 1 mm in size. This method utilises the differences in physico-chemical surface properties of particles of various minerals. In this case it utilises the fact that coal is naturally

hydrophobic that is, water-repellent.The phenomenon is measured by the contact angle, illustrated in figure 2.1.

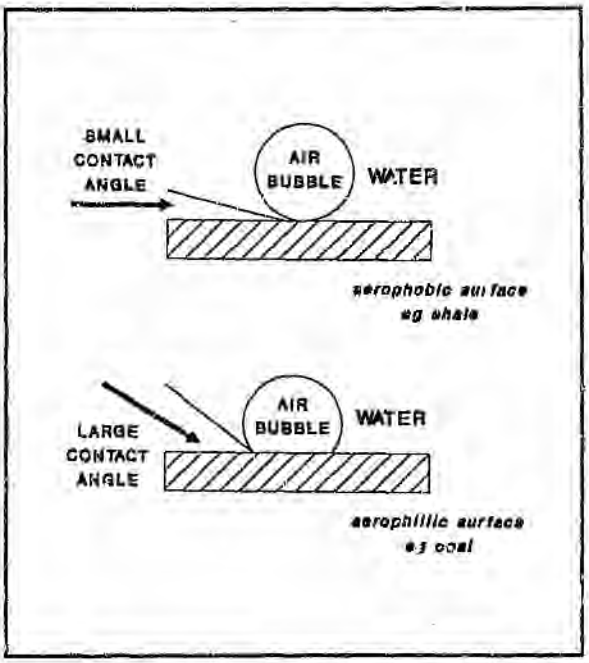


Figure 2.1 Contact Angle

After treatment with reagents, the differences between organic material and inorganic material become apparent. For flotation to take place, an air-bubble must be able to attach itself to a particle, and lift it to the water surface (see figure 2.2). If the particle is small enough, the bubbles bear organic particles to the surface of the suspension, otherwise the adhesion between the particle and the bubble will be less than the particle weight and the bubble will therefore drop its load.

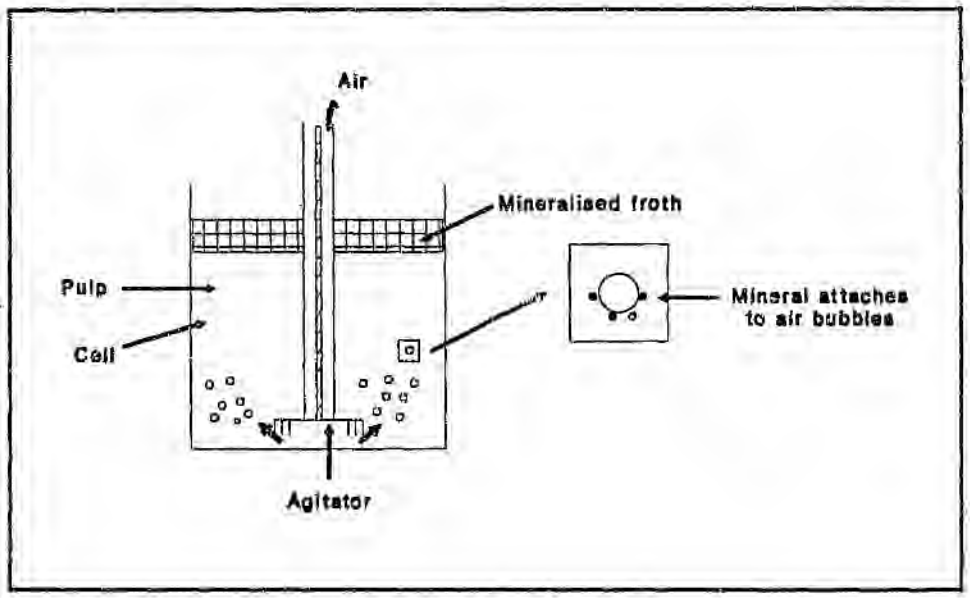


Figure 2.2 Principle of flotation

Reagents employed to promote the linkage between particle and bubble are classified in two types. These are collectors and frothers.

Collectors, are used to render the coal particles more hydrophobic in order to promote contact between the particle and the air–bubble. They are required to be selective, that is encourage attachment if the mineral content of the particle is slight, discourage it if the particle has more than a certain mineral content. Hydrocarbon oils are generally used for coal flotation such as kerosene and fuel oil.

Frothers are used to reduce the surface tension of water. They are usually compounds with hydroxyl, carboxyl, ester or carbonyl groups.

Other parameters affecting the efficiency of the process are:
- feed particle size distribution,
- density,
- petrographic composition,
- degree of surface oxidation,
- pulp density,
- water ionic composition,
- temperature,
- PH,
- cell design,
- agitation,
- aeration,
- pulp level, etc (Horsfall, 1992).

Oil agglomeration is also a surface property dependent process. It is being developed by several organisations including the National Research Council of Canada. The oil agglomeration process has become more attractive because of the reduction in the quality of resources and hence the necessity for grinding to

liberate impurities.

Oil agglomeration(**figure 2.1**) depends on the different behaviour between hydrophobic and hydrophillic materials in oil–water mixtures. Since coal particles are essentially hydrophobic in nature, they can be readily agglomerated by the addition of oil. On the other hand, inorganic ash–forming constituents which have hydrophillicity are rejected and remain in the water.

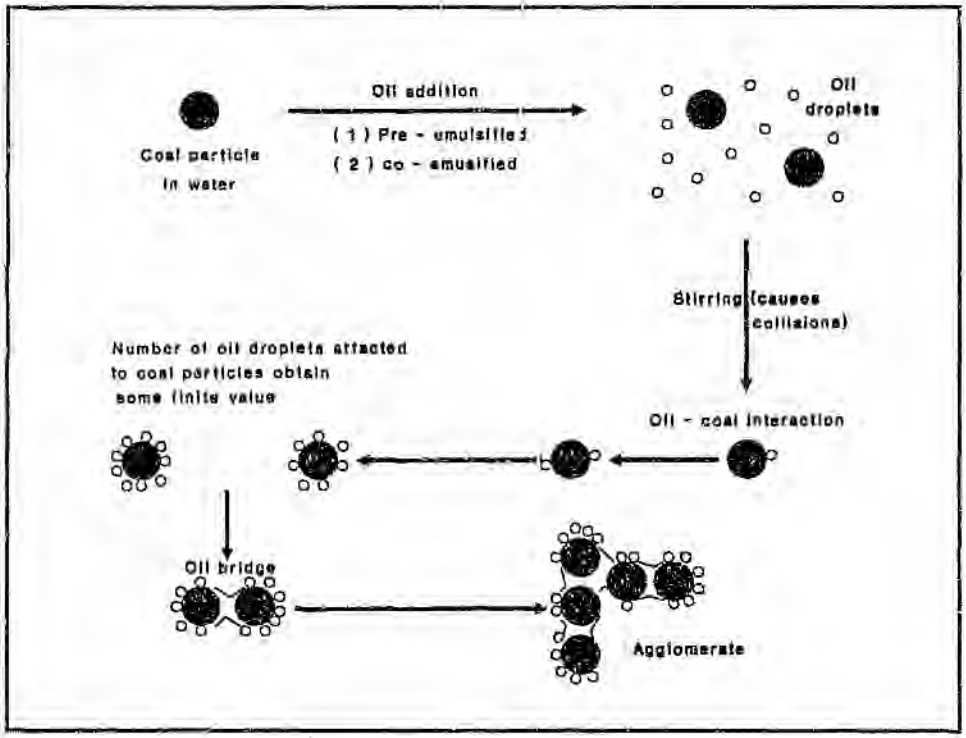


Figure 2.3 Principle of oil agglomeration process

The procedure for oil agglomeration may be summarized as follows;

– the coal and the inorganic minerals are wetted by oil,

– the coal–oil–water system is agitated with a high shear mixer which causes oil droplets and coal particles to collide,

– the oil–coated coal particles agglomerate (Capes, 1980).

A generalized flow diagram with suggested equipment for the coal agglomeration process is shown on **figure 2.2** (Liu, 1982).

The most important factors affecting the separation efficiency of the oil agglomeration are;

- type and amount of the oil used,
- level of duration of mixing,
- surface nature of impurities

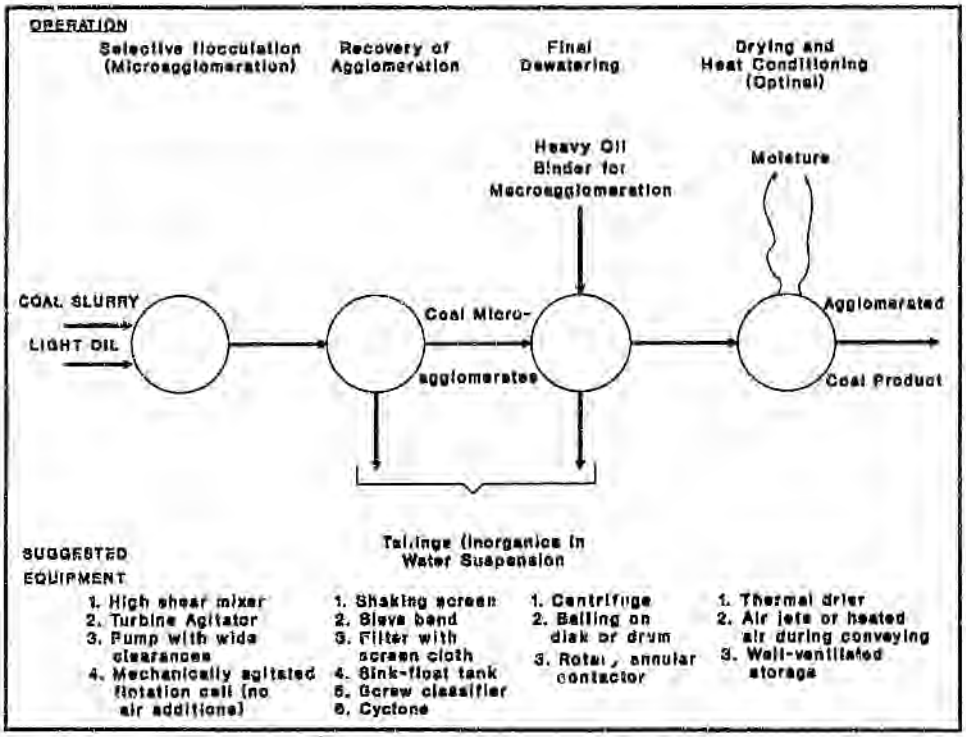


Figure 2.4 Coal agglomeration flowsheet showing alternative processing steps and types of equipment

Type and amount of the oil used

The lighter, refined oils are more selective than the heavy complex oils in recovering the low ash coal product. The heavy oil recovers not only the carbonaceous fraction of the slurry but also the ash fraction. Commercial oils like Shellsol AB and Shellsol K, diesel and furnace oil are relatively cheap and

efficient in producing coal agglomerate. 5% to 30% of oil has been shown to be effective in recovering carbonaceous products. The characteristics of the agglomerate are also dependent on the amount of oil used.

Time of mixing

Approximately 8 minutes are required to complete the agglomeration at 600 rpm in a blender, and 18 minutes are required at the lower speed of 300 rpm.

Surface nature of coal

Oil agglomeration is capable of more efficient coal desulphurization than the other surface dependent methods as it changes the nature of the pyrite surface from hydrophobic to hydrophillic (Tsai, 1982; Mishra 1987).

2.2.3 Optical Methods

Several methods for evaluating coal liberation exploiting optical properties have been documented in the literature recently. Huggins et al (1982) described a method by which minerals in coal can be quantitatively determined by the combination of a Scanning Electron Microscopy-Based Automated Image Analysis (SEM-AIA) and Mossbauer Spectroscopy. This method also facilitates the study of mineral association and could possibly be applied to mineral-maceral associations.

Straszheim et al (1988) used SEM-AIA and energy-dispersive X-ray spectroscopy to characterize mineral matter in sub-bituminous coals. This method is able to show up differences in sample composition with cleaning, but is unable to determine the mode of mineral association i.e. whether the mineral particles are disseminated widely throughout the coal or clustered together. In this respect, the actual degree of dissociation has qualified by usual physical separation process on the comminuted material.

3 COAL USED IN INVESTIGATION

3.1 Reserve and Stratigraphy

The part of the Waterberg Coalfield currently being exploited by the South African Iron and Steel Industrial Corporation Limited (ISCOR) is situated near the town of Ellisras in the North Western Transvaal.

It contains the continent's largest reserves; approximately 46% of South African known in situ reserves. It covers 730 square miles and coal bearing horizons have been identified which reveal a maximum aggregate thickness of 430 ft. The figure of the coal reserves in the Waterberg coalfield is given in **table 3.1**.

Table 3.1 Coal reserves (Raw Coal In Situ *) in the Waterberg coalfield (In megatons)

		Bright Coal	Dull Coal	Total
Opencastable †:	State area ‡	11 095	7 410	18 505
	Other §	12 956	2 814	15 773
	Total	24 054	10 224	34 278
Deep Coal:	State area	30 039	4 839	34 878
	Other	12 178	5 213	17 391
	Total	42 217	10 052	52 269
Total:	State area	41 134	12 249	53 383
	Other	25 137	8 027	33 164
	Total	66 271	20 276	86 547

* *In situ* reserves include all coal irrespective of the thickness of individual layers

† *opencastable* is the area where the Upper Ecca coal can be mined at stripping

ratio of at most 0.98 bank cubic metres (BCM) per ton of raw coal

‡ *State area* is the area where the state has reserved mineral rights

§ *"Other"* refers to areas where mineral rights are held by the private sector.

Coalfield stratigraphy was established by borehole exploration. The Waterberg coalfield occurs in the Karoo Sequence. Subdivision of the Karoo Sequence is given in **table 3.2**.

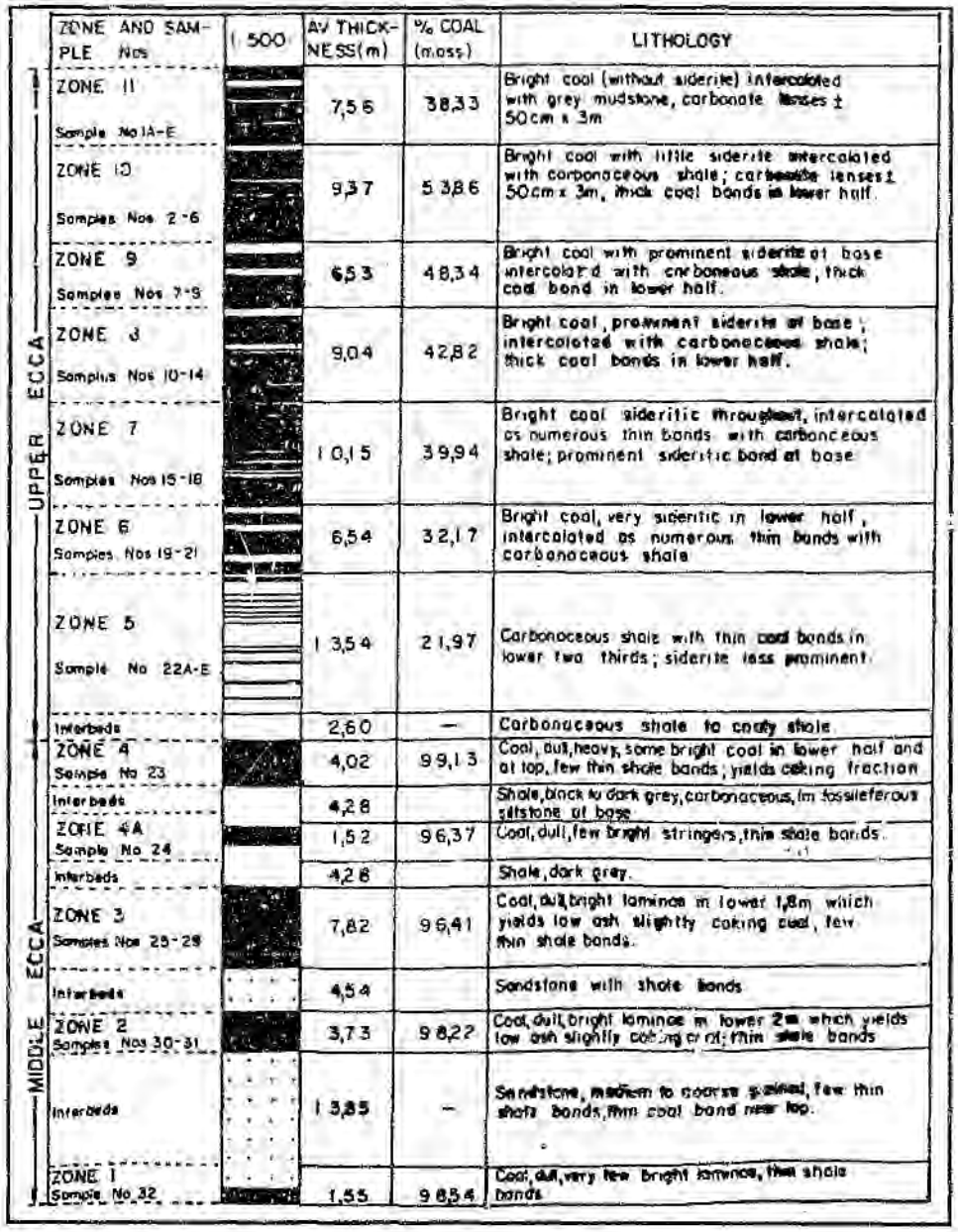
Table 3.2 Subdivision of the karoo sequence

GROUP	FORMATION	REPRESENTATIVE ROCK	AV.THICK(m)
STORMBERG	DRAKENSBERG	Lava, purplish to red, amygdaloidal	95 m
	CAVE SST.	Sandstone, fine grained, white, yellow brown, reddish	80 m
	RED BEDS	Mudstone, red to chocolate brown, clayey	90 m
	MOLTENO	Sandstone, white, med to coarse grained, scattered pebbles	15 m
BEAUFORT		Mudstone, purple and greenish grey, alternating at top, light grey at base	90 m
ECCA	UPPER ECCA	Intercalated shale and bright coal	60 m
	MIDDLE ECCA	Sandstone and grit, intercalated carb, shale, siltstone, few thick coal seams mainly dull	55 m
	LOWER ECCA	Shale and sandstone, grit in lower portions	150 m
DWYKA		Tillite	3 m

The Waterberg Coalfield forms a part of the Ecca group and consists of coal–rich zones interspersed with mineral rich zones in the upper regions (which are already mined) and dull coal in the middle regions.

The upper region of the Ecca group in the Waterberg Coalfield has an average thickness of 60 m and consists of seven zones(numbered 11–5). Each zone consists of a different number of various coal and mineral matter layers. A typical

borehole showing the zones in upper and middle Ecca formation of Grootegeluk coal mine is given in **figure 3.1**.



	ZONE AND SAMPLE Nos	1 500	AV THICKNESS(m)	% COAL (mass)	LITHOLOGY
UPPER ECCA	ZONE 11 Sample No 1A-E		7,56	38,33	Bright coal (without siderite) intercalated with grey mudstone, carbonate lenses ± 50cm x 3m
	ZONE 10 Samples Nos 2-6		9,37	53,86	Bright coal with little siderite intercalated with carbonaceous shale; carbonate lenses ± 50cm x 3m, thick coal bands in lower half.
	ZONE 9 Samples Nos 7-9		6,53	48,34	Bright coal with prominent siderite at base intercalated with carbonaceous shale, thick coal band in lower half.
	ZONE 8 Samples Nos 10-14		9,04	42,82	Bright coal, prominent siderite at base; intercalated with carbonaceous shale; thick coal bands in lower half.
	ZONE 7 Samples Nos 15-18		10,15	39,94	Bright coal sideritic throughout, intercalated as numerous thin bands with carbonaceous shale; prominent siderite band at base
	ZONE 6 Samples Nos 19-21		6,54	32,17	Bright coal, very sideritic in lower half, intercalated as numerous thin bands with carbonaceous shale
	ZONE 5 Sample No 22A-E		13,54	21,97	Carbonaceous shale with thin coal bands in lower two thirds; siderite less prominent.
	Interbeds		2,60	—	Carbonaceous shale to coaly shale
	ZONE 4 Sample No 23		4,02	99,13	Coal, dull, heavy, some bright coal in lower half and at top, few thin shale bands; yields coking fraction
	Interbeds		4,28		Shale, black to dark grey, carbonaceous, fossiliferous siltstone at base
	ZONE 4A Sample No 24		1,52	96,37	Coal, dull, few bright stringers, thin shale bands.
	Interbeds		4,28		Shale, dark grey.
	ZONE 3 Samples Nos 25-29		7,82	96,41	Coal, dull, bright laminae in lower 1,8m which yields low ash slightly coking coal, few thin shale bands.
	Interbeds		4,54		Sandstone with shale bands
MIDDLE ECCA	ZONE 2 Samples Nos 30-31		3,73	98,22	Coal, dull, bright laminae in lower 2m which yields low ash slightly coking coal; thin shale bands
	Interbeds		13,85	—	Sandstone, medium to coarse grained, few thin shale bands, thin coal band near top.
	ZONE 1 Sample No 32		1,55	98,54	Coal, dull, very few bright laminae, thin shale bands

Figure 3.1 Grootegeluk coal mine coal zones

Zones in the upper region are opencast mined in four benches with an upper bench including some of zone 11 that is overburden. The four coal benches are

composed as follows;

Bench Number	2	3	4	5
Bench Zones	11 -10	9-8	7-6	5

All zones contain bright coal, zone 5 also contains a high phosphorus coal.

The middle region of the Ecca group selected for mining attains a thickness of 55 m and consists of four zones. It contains typical Eastern Transvaal dull coal which can be directly mined, crushed and screened to be supplied to Eskom without beneficiation as it is clean coal and it consists of non carboniferous material (shale and sand stone) occurring between the coal seams. The non carboniferous material from benches will be directly stockpiled back in the opencast pit.

3.2 Characteristics

The following analyses are carried out at the Grootegeluk Coal mine for quality control.

Proximate and Ultimate Analyses

Each coal sample of each zone is subjected to float and sink analysis from 1,3 to 2,2 in steps of 0.1. All the shale samples are also used for float and sink analysis from 1,5 to 2,2 in order to determine the amount of coal interspersed.

Proximate and ultimate analyses are carried out on each density fraction of float and sink analysis on each sample of zone.

These results are combined to calculate the proximate and ultimate analysis for each zone.

The resulting proximate and ultimate analyses are only given for the upper Ecca group. The analyses of coking coal and power station coal in each zone are given below;

Coal Analyses on Coking Coal

Zone Number	Yield * %	Ash %	VM	ROGA
11	10.8	12.7	36.2	64
10	14.0	11.2	36.6	65
9	13.2	11.4	36.1	65
8	12.8	10.4	36.1	65
7	21.2	10.6	36.4	67
6	23.4	9.7	36.7	65
5	15.0	10.0	36.0	57

* The yield is calculated from float and sink analyses.

The table given above reveals that the yield of high quality coal (<12% ash) in Upper Ecca regions ranges from 10.8% to 23.4%. It is only possible to get the higher yields from zones 7 and 6. As seen from the borehole data in fig 3.1(page 18), the mass of coal in zones 7 and 6 is not high enough to make a significant difference on overall yield, therefore on average only 12% of coal can be produced as coking coal. This table also shows that the ash content of each zone shows that the ash content of each zone ranges from 9.6% to 12.7%. There is no significant change in the volatile matter of each zone, which averages 36%. The Roga index shows each zone has bright coal, and except for zone 5 the Roga values are similar.

Proximate Analysis on Middlings (Steam Coal)

Zone Number	H_2O %	CV %	Ash* %	VM %	FC %	Total S. %	Abr. Index	HGI
11	2.6	23.5	28.1	30.6	39.3	1.6	189	51
10	2.5	22.6	30.6	30.9	38.7	1.2	228	52
9	2.4	23.7	27.6	30.3	39.8	1.1	172	51
8	2.2	22.5	30.5	30.7	38.9	0.8	206	52
7	2.1	23.2	28.1	30.4	39.8	0.9	154	50
6	2.1	23.7	26.7	30.1	40.5	1.0	141	49
5	2.1	23.5	27.1	29.2	41.8	0.8	146	51

The combination ash composition for each zone is given below;

Z.N	SiO_2 %	Al_2O_3 %	Fe_2O_3 %	MgO %	CaO %	Na_2O %	K_2O %	TiO_2 %	P_2O_5 %	SO_3 %
11	70.1	14.3	5.5	1.1	2.7	0.2	1.3	0.6	0.1	3.2
10	70.7	18.6	3.5	0.9	1.4	0.2	1.2	0.8	0.1	1.6
9	64.7	17.2	9.5	0.9	2.1	0.2	0.9	0.7	0.1	2.5
8	64.9	19.8	7.1	1.0	2.0	0.1	0.8	0.8	0.1	2.3
7	61.1	20.9	7.5	1.0	1.9	0.1	0.7	0.9	0.1	2.4
6	51.0	22.6	9.8	1.7	6.1	0.1	0.5	1.1	0.2	6.5
5	50.5	25.7	3.8	1.8	7.3	0.2	0.7	1.4	1.0	5.9

Petrographic Analysis

Zone Number	Vitrinite	Exinite	Active Semifusinite	Inertinite
11	89.5	0.4	1.2	3.1
10	87.0	1.0	1.9	5.0
9	87.5	1.0	1.4	4.6
8	88.1	1.1	1.5	4.7
7	85.8	1.4	2.0	6.0
6	86.6	1.5	2.1	5.4
5	82.6	1.4	3.5	7.6

(Dreyer, 1991)

3.3 Present Beneficiation Plant

There are two coal preparation plants at Grootegeluk. One of them was only recently commissioned (1991). According to Mr PL Gouws in a paper published in 1976, the prime objective of the first coal preparation plant in Waterberg was to supply some 1,8 m tons of blended coking coal to the ISCOR's steel works. In addition to metallurgical coal, some 2,0 m tons of steam raising coal would also be produced.

Primary breaking and screening streams are employed for the reduction of the ROM material to −150 mm for the liberation of coal. Each stream consists of a receiving bin with a 900 mm square opening grizzly, apron feeder, rotary breaker, conveyors, screens and storage bunkers. Three streams are normally in operation.

−150 mm product from the rotary breaker is dry screened in one stream on 15 mm aperture screens. The 150x15 mm and −15 mm products are fed to storage bunkers for beneficiation in the dense medium Teska vessels plants and the primary dense medium cyclones respectively.

150x15 mm size fraction is fed to the Teska Vessel for separation at a relative density of 1,7. The sinks material(>75% ash) is disposed of as final waste via a conveyor system. The 25% ash floats material is crushed to -10 mm in order to liberate coking coal. This is separated from the steam raising coal in the secondary Dense Medium Cyclone and froth flotation plants.

In the primary cyclone plants, the 15x0.5 mm raw coal is washed at 1,7 RD. The 150x15 mm floats, crushed to -10 mm joins the 15x0.5 mm floats at 1,7. The combined 1,7 floats are rescreened at 0.5 mm and the mixture of -10x0.5 mm and 15x0.5 mm material are fed to secondary cyclone plant to be washed at 1,4 RD. Sinks from the 1,4 wash are middlings, the quality being given below. All the minus 0.5 mm fines are submitted to froth flotation; the clean coal joining the metallurgical coal, and the tailings being added to the middlings. The process of beneficiating Waterberg coal described above is as it was when this project was started. It has since been modified. Now -1 mm size fraction is treated in the spiral plant.

The estimated unit yields (ROM basis) for current production give the following overall yields :

	ASH	**YIELD**
Clean Coal	10% - 11%	12%
Middlings	33% - 35%	24%
Discard	> 70%	65%

The second plant now adds power station coal at an approximate rate of four million tons/year. This has doubled to production of power station coal.

The expansion project (single stage beneficiation plant) has provided the necessary facilities for the delivery of sufficient power station coal to meet the requirements of the mammoth Matimba dry-cooled power stations, in terms of the Iscor-Eskom

coal delivery agreement (Mining Mirror,1990).

3.3.1 Properties of The Clean Coal and Middlings

Waterberg coal is characterised by containing a high proportion of reactive macerals. The beneficiation plant products have the following typical analyses:

PROXIMATE (AIR - DRY)

	CV	H_2O	ASH	VM	FC	TOTAL	ABR	HGI
	MJ/kg	%	%	%	%	S%	Index	
Coking Coal	28,8	3,5	10,3	34,5	51,7	0.85	100	53
Steam Coal	20,2	2,5	34,9	27,4	35,2	1,27	440	48

ASH FUSION TEMPERATURE (°C , REDUCING ATMOSPHERE)

	DT	HT	FT
Coking Coal	1 390	+1 400	+1 400
Steam Coal	1 390	+1 400	+1 400

ULTIMATE (DAF)

	C	H	N	Org.S	O	VM	ASH
	%	%	%	%	%	%	%(Ad)
Coking Coal	80.8	5.4	1.5	0.7	11.6	40.0	10.3
Steam Coal	78.2	5.5	1.5	0.6	14.1	43.8	34.9

PETROGRAPHIC ANALYSES ON DMMF BASIS

	VITRINITE	EXINITE	RSF	INERTINITE
Coking Coal	81.3	3.8	9.1	5.8
Steam Coal	75.2	5.2	7.2	12.4

ASH ANALYSIS

	WASHED COAL	MIDDLINGS
SiO_2	69.9	69.3
Al_2O_3	17.8	18.3
Fe_2O_3	4.9	4.7
P_2O_3	0.1	0.1
TiO_2	1.7	0.8
CaO	2.2	3.3
MgO	0.8	0.8
K_2O	1.3	1.4
Na_2O	0.2	0.1
SO_3	1.3	1.2

CL in coal : 0.01%

3.4 Collection and Preparation of Samples

3.4.1 Sample Collection

When the samples were received, the aim was to get a "typical" sample of the bench. This would give some information about general behaviour of the benches. A "typical" sample can be defined as one having average characteristics of a seam but not necessarily representative of the seam. ISCOR undertook to prepare such samples. Because of limited handling facilities at WITS plant samples were taken as this gave handle–able particle sizes.

Three batches of samples were received from the three working benches from Grootgeluk coal mine and sent to the pilot plant in Pretoria West. All samples were received in plastic bags tied with cotton rope.

After discussion with Mr HC Voges (Manager, Metallurgical Services, Iscor), the four plant products (namely raw coal, Teska product, middlings, Teska discard) of each bench were accepted.

The distribution of the work carried out on the samples is given in **figure 3.2**. The four plant products of each bench were collected from Pretoria and sent to Richlab for comminution and treatment. Sub samples of comminuted products were sent to Wits for liberation studies and treatment. After further discussion it was decided to use the sample derived from the top bench (No.2) for the test work at WITS, because of the similarity of float and sink and petrographic analyses of each.

This bench is composed of zone 11 and zone 10. These zones together represent a thickness of 16.9 m and comprise 33% of the coal mined from Grootegeluk coal mine. 12% coking coal at an ash content of 12% and also middlings with 30–35% ash content can be obtained. They average 88% vitrinite.

The samples were not specifically treated to minimise the degree of oxidation, as they had already been oxidised when they had been collected. Even if a minimal degree of oxidation had occurred, this would not make any significant difference to the conclusions. This project still proved that a higher yield of higher quality coal than currently produced can be achieved. If it were possible to obtain completely unoxidized samples, it is likely that an even better quality product could be achieved.

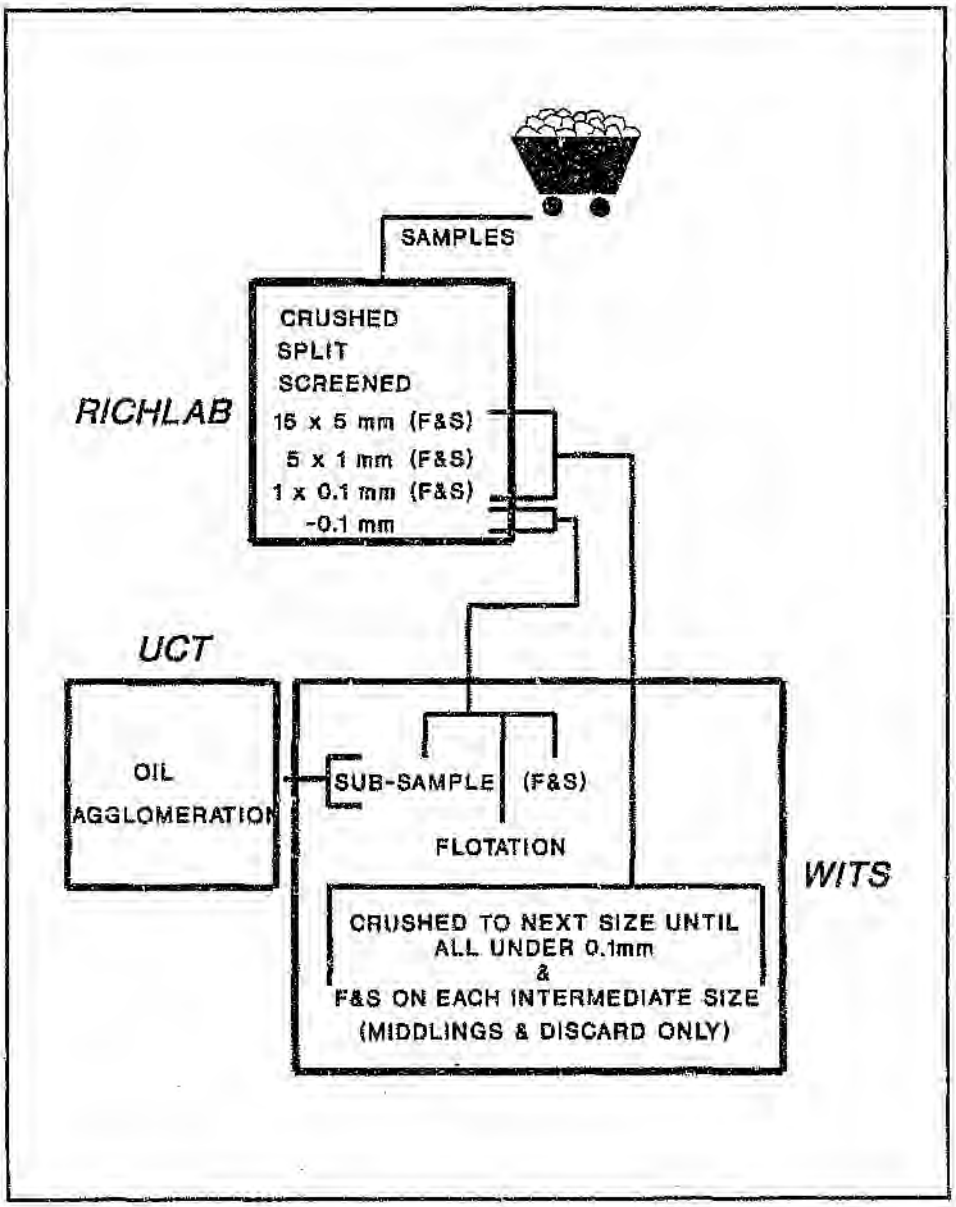


Figure 3.2 The distribution of work carried out on samples

3.4.2 Sample Crushing and Screening

The samples were all crushed to -15 mm using a jaw crusher at Richlab. Crushed samples were then screened into the following size fractions (15x5 mm, 5x1 mm, 1x0.1 mm, -0.1 mm). Sub-samples of -15 mm material were sent to Wits

University and progressively crushed to –5 mm, –1 mm, –0.1 mm. At Wits a jaw crusher was used to crush to –5 mm, a roll crusher to crush to –0.1 mm, and a hand mill to crush –1 mm material to –0.1 mm. The size analyses on the products from each plant are given below.

TOP BENCH SAMPLE

Table 3.3 Size analyses on middlings

TOP SIZES(mm)					
–15		**–5**		**–1**	
15x5	45.1%	5x1	70.6%	1x0.1	82.5%
5x1	43%	1x0.1	26.9%	0.1x0	17.5%
1x0.1	11.1%	0.1x0	2.5%		
0.1x0	0.8%				

Table 3.4 Size analyses on discard

TOP SIZES(mm)					
–15		–5		–1	
15x5	60.9%	5x1	76.5%	1x0.1	86.6%
5x1	29.0%	1x0.1	20.1%	–0.1	13.4%
1x0.1	8.5%	–0.1	3.4%		
–0.1	1.6%				

Table 3.5 Size analyses on raw coal

TOP SIZE (mm)	
-15	
15x5	37.3%
5x1	40.0%
1x0.1	17.8%
-0.1	4.7%

Table 3.6 Size analyses on Teska product

TOP SIZE (mm)	
-15	
15x5	42.5%
5x1	42.6%
1x0.1	12.5%
-0.1	2.4%

The three larger sizes were used for float & sink analysis; -0.1 mm ultra-fines were used for float & sink using the centrifuge method, froth flotation tests and oil agglomeration tests.

3.5 Sample Treated and The Techniques Employed

3.5.1 Float and Sink Analysis

Float & sink analyses on the original sizes screened and of −15 mm top size samples were carried out by Richlab. The recrushed materials were treated by float & sink analysis at Wits.

The Mineral Processing Laboratory at the Department of Metallurgy and Materials Engineering in Wits is equipped to carry out F & S separations on sizes below 1 mm. A beaker was used for F & S on +1 mm size fraction.

Two different methods of float and sink are used in this project. The differences between the two techniques is shown in **figure 3.3**.

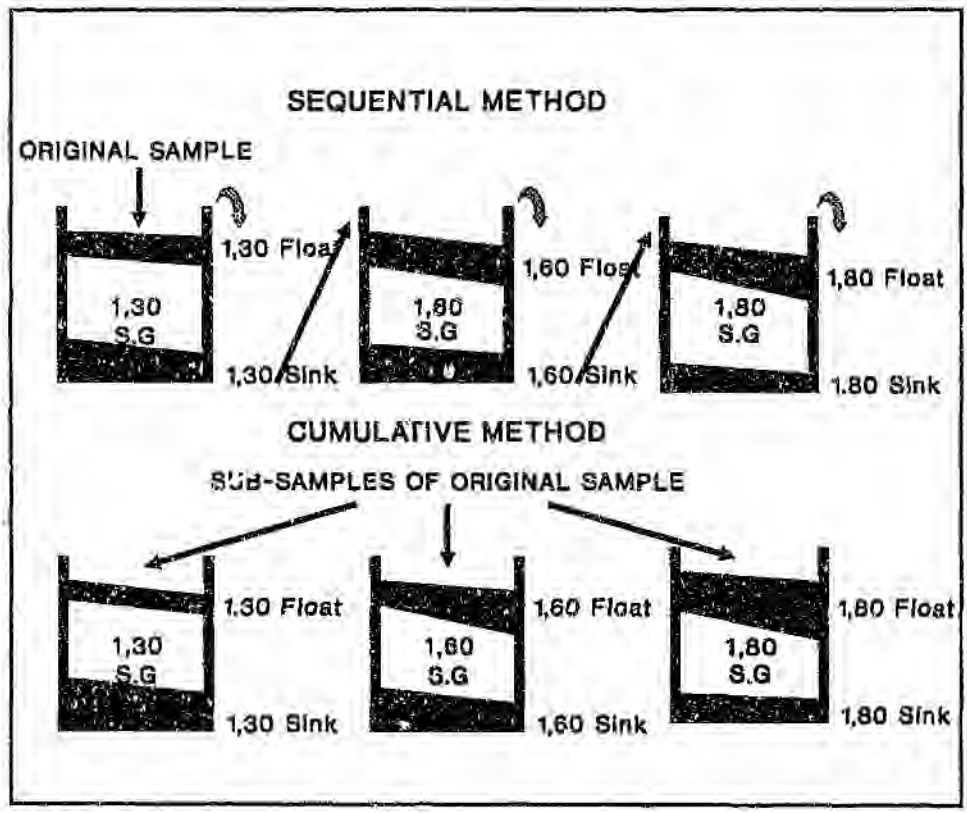


Figure 3.3 The differences between sequential and cumulative float and sink

The methods shown above can be summarized as follows;
The general principle of the washability by the sequential method is that the relative density chosen should enable as much of the sample as possible to be removed in the first stage. Discard should be initially floated at the highest relative density used and clean or raw coal at the lowest relative density.

Initially the whole test sample was separated at the lowest relative density, and then all the float material was dried in air, weighed and, as required, prepared for analysis.

The sink material was well drained, care being taken not to alter the relative density of the succeeding liquids by introducing incompletely drained material from the previous liquid, and then introduced (a little at a time if necessary) into the medium of next highest relative density. The float material was again dried, weighed and, as required prepared for analysis. This process was repeated until separation in the medium of highest relative density had been carried out. The sink material from this final separation was dried, weighed and prepared for analysis as required.

The method described above way used in the first two size ranges and is called the "sequential method" as this gives far superior results to the "cumulative method". Previously the separation was commenced at the lowest S.G (around 1,3) and advanced upwards in steps of 0,1.

In all experiments Certigrav was used as the medium. Depending on the mixtures employed, Certigrav can have a specific gravity from 1,2 - 2,5.

Washability by the cumulative method requires careful subdivision of the sample (i.e, for a eight-point separation, eight subsamples are required). Each subsample is than introduced into the appropriate specific gravity, the float material and the final sink products are weighed and the percent ash for the resultant products

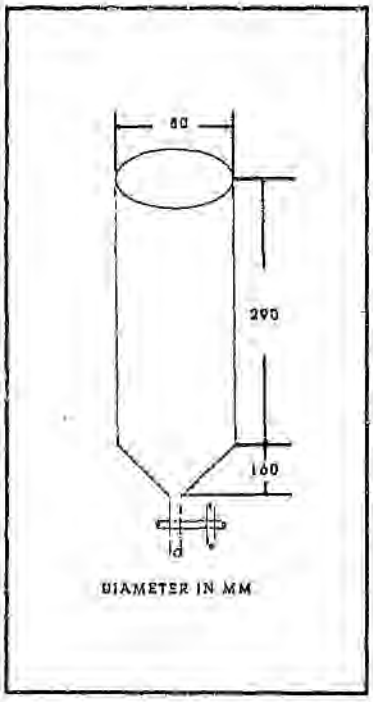
analyzed. The weight percent and ash content at a particular specific gravity fraction can then be calculated.

3.5.1.1 Float and Sink at Coarse Sizes

A 5 kg sample of 5x1 mm size fractions was obtained by riffling for this experiment.

A 5000 ml beaker was used as float & sink vessel. The beaker was half filled with Certigrav of the desired specific gravity. The sample was slowly introduced in several steps. The sample was stirred with a glass rod and left to settle. After settling floats were removed with a scoop . The liquid was poured through the screen and the sinks left to drain. The S.G of the liquid was re-measured by hydrometer immediately after the completion of separation.

3.5.1.2 Float and Sink at fine sizes



A 1,5 kg sample of 1x0.1 mm size fractions was obtained by riffling for this experiment.The larger column was used for float & sink analysis. The column(**figure 3.4**) was half filled with Certigrav of the required S.G and the sample slowly introduced from the top. The sample was agitated with a glass rod and allowed to settle. After separation the sinks were discharged into a funnel and filtered. In order to avoid compaction and blockage, sinks were continuously removed. Whatever remained in suspension was handled as floats. The S.G of the liquid was measured immediately after the completion of separation.

Figure 3.4 The columns used for float and sink analysis

3.5.1.3 Float and Sink at Ultrafine Sizes

An accurately weighed sample (1 gram) of coal was transferred to a test tube, 75% full of the required gravity solution, and stirred with a glass rod until all the coal was moistened. The tubes were balanced, then centrifuged for one hour at 2500 rpm. After one hour they were taken out and left overnight to effect a clear separation. Floats were then removed by means of the arrangement as shown in **figure 3.5**, filtered , dried and weighed. Sinks were washed out from the tubes, filtered , dried and weighed. The same procedure was applied for specific gravities from 1,3 to 1,8.

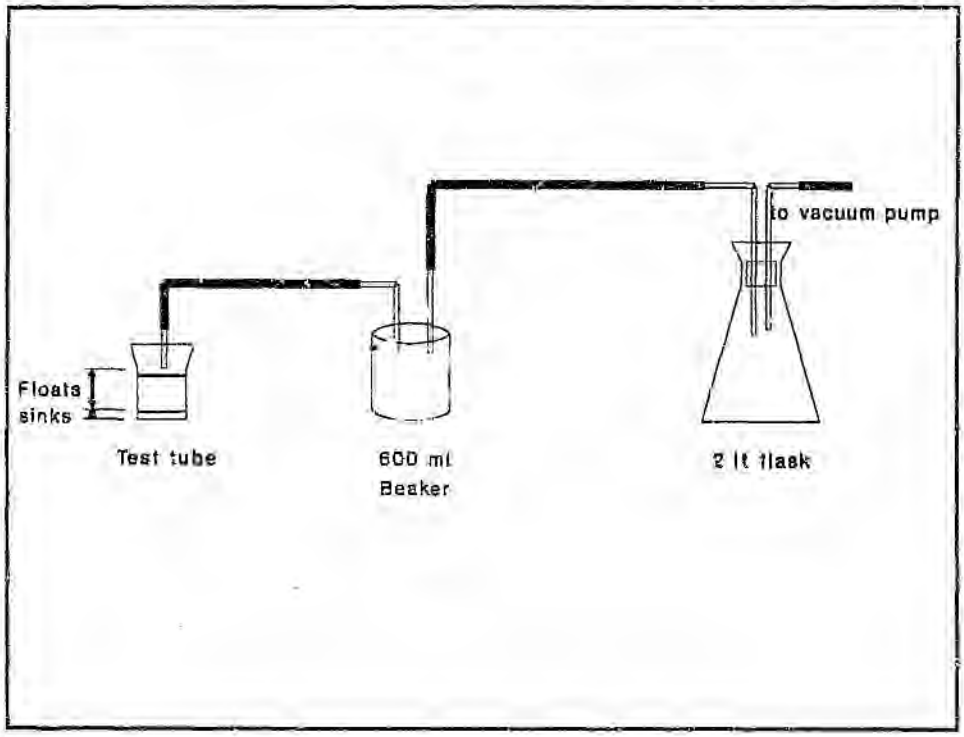


Figure 3.5 The arrangement used to remove floats from sinks

3.5.2 Froth Flotation

The laboratory flotation tests on -0.1 mm were all carried out at Wits in a 1 lt

laboratory Denver cell. The 1 lt cell was chosen as only a small amount of coal was available in the -0.1 mm size. The pulp level was controlled manually. Froth concentrates were removed by scraping. Reagents were diluted with water to 1% concentration and emulsified in a high speed industrial blender. The cell rotor speed was maintained at 1500 rpm and only rotor-induced air was added (Denney, 1983).

The froth flotation method " tree " was developed in Australia (**figure 3.6**). The procedure of this method permits a number of froth products to obtained (such as rough and clean products). The purpose of this extended procedure is to provide information similar to that provided by the float & sink curve.

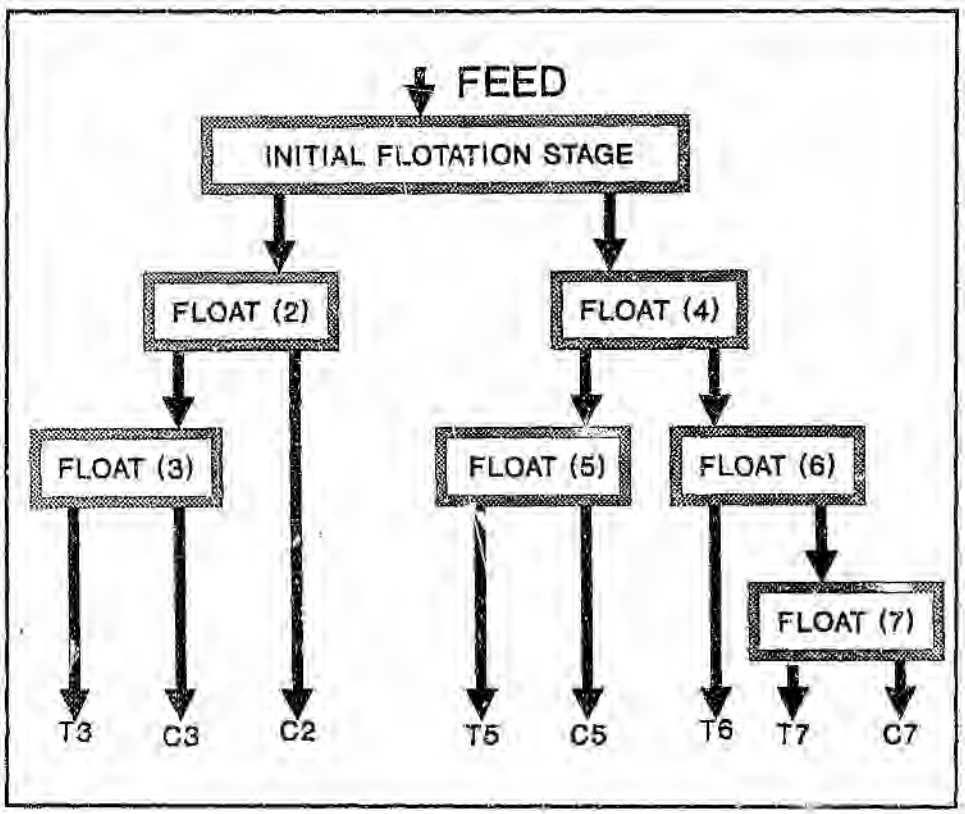


Figure 3.6 Method of tree

This process has 8 separation stages (Australian Standard) (**figure 3.6**). In the

initial separation stage, no collector was used and the froth was removed until it became barren. When the froth product was removed, the tailings commodity was retained in the cell. In the second flotation stage, the pulp in the cell was conditioned with collector (Shellsol A). The same applied to subsequent stages. The froth product was again removed and the same procedure was applied to the tailings retained in the cell. These tailings were scavenged twice. The concentrate set aside in the initial separation stage was also refloated twice. At the end of the experiment, eight final products were obtained and arranged according to their ash content.

3.5.3 Oil Agglomeration

The oil agglomeration tests were carried out by UCT. The equipment used consists of a 250 ml beaker, a mechanical stirrer and a spatula inserted into the beaker to serve as a baffle(**figure 3.7**). A 106 μm sieve was used to separate the agglomerate from unwanted materials.

The content of the oil used for this process ranged from highly selective aliphatic to less selective aromatic oils and mixtures of the previous two hydrocarbons. A list of the oil used is given below;

- Shellsol AB (95% aromatic hydrocarbons)
- Shellsol K (95% aliphatic hydrocarbons)
- Shellsol AB/K ((50:50) by mass)

An accurately weighed sample (about 7,5 gram) was wetted with a minimal amount of water to form a thick paste and transferred into the oil agglomeration vessel. Sufficient water was added to make up 10% solid content. The coal-water mixture was agitated at the speed of 1660 rpm for approximately 5 minutes to ensure complete wetting of the coal.

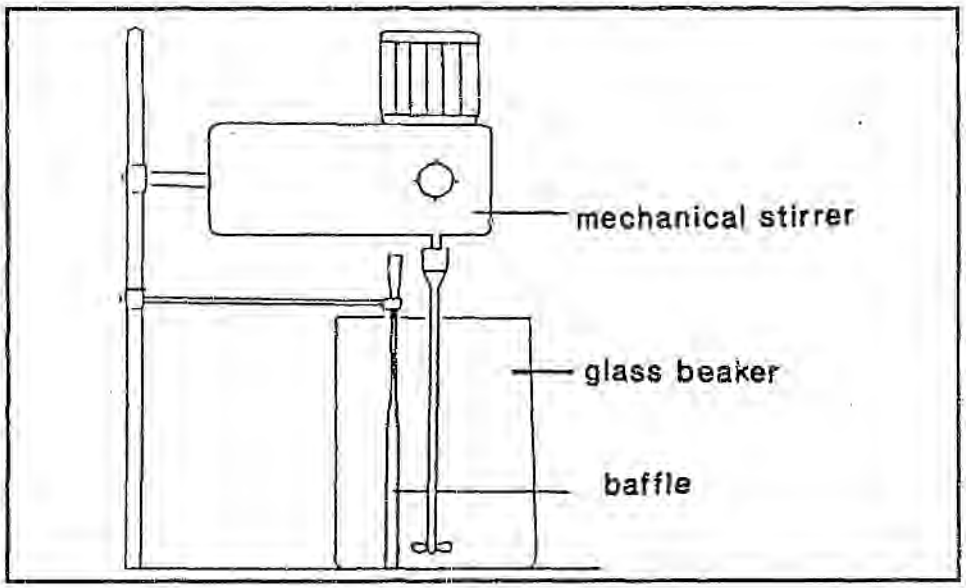


Figure 3.7 Oil agglomeration equipment

Between 10% and 35% (m/m) reagent was added to the pulp to make an oil bridge. The oil agglomeration time was visually determined as the time taken from addition of bridging oil to the formation of agglomerates. When the agglomeration was completed, the agitation was stopped and the agglomerates were separated by screening the mixture through the 106 μm test sieve.

The bridging oil was removed from the coal surfaces by washing with acetone. The agglomerates and unwanted material fractions were dried in a microwave oven and then weighed and analyzed for ash content.

3.5.4 Repeatability and Reproducibility

For the precision of the results of the experiments two terms are defined by the South African Bureau Standards as follows;

- Repeatability(formerly known as "within laboratory tolerance"); and
- Reproducibility(formerly known as "between laboratory tolerance").

Repeatability is the extent to which the same operator, using the same equipment, can expect his results to agree, if he continually re-analyses sub-samples of the same sample of prepared coal.

Reproducibility is the extent to which another operator, given a sub sample of the same original sample of coal, but working in a different laboratory with different, but standard equipment, can be expected to agree with the first operator.

Repeatability experiments were only carried out on a few samples, because of the limited amount of sample available. The results below will give some idea of the precision of the results. Repeatability was not performed for ash analysis, because the two samples of each density separation were required to be combined for ash analysis due to the yield.

Sample Name: Teska Discard

Density	Experiment 1 Yield		Experiment 2 Yield		Error
	Float %	Sink %	Float %	Sink %	%
1.3	1	99	2	98	1
1.4	6	94	6	94	0
1.5	14	86	13	87	1
1.6	17	83	18	82	1
1.7	20	80	22	78	2
1.8	21	79	20	80	1

Sample Name: Teska product

Density	Experiment 1 Yield		Experiment 2 Yield		Error %
	Float %	Sink %	Float %	Sink %	
1.3	18	82	19	81	1
1.4	50	50	51	49	1
1.5	66	34	71	29	5
1.6	67	33	72	28	5
1.7	80	20	80	20	0
1.8	84	16	85	15	1

Sample Name: Middlings

Density	Experiment 1 Yield		Experiment 2 Yield		Error %
	Float %	Sink %	Float %	Sink %	
1.3	11	89	10	90	1
1.4	37	63	36	64	1
1.5	47	53	51	49	4
1.6	58	42	59	41	1
1.7	63	37	62	38	1
1.8	69	31	67	33	2

Sample Name: Raw Coal

Density	Experiment 1 Yield		Experiment 2 Yield		Error
	Float %	Sink %	Float %	Sink %	%
1.3	1	99	1	99	0
1.4	30	70	29	71	1
1.5	41	59	45	55	4
1.6	47	53	43	57	4
1.7	53	47	55	45	2
1.8	64	36	62	38	2

3.5.5 Ash Determination

Ash content analyses of the samples were performed using a muffle furnace according to SABS methods. When insufficient material was obtained several fractions were combined.

Standard method (SABS 926)(see **Appendix B**) was used for ash determination.

3.5.6 Simulation

Simulation techniques are used to determine the optimum beneficiation process. Carrying out actual washing tests requires large tonnages of coal. Now that an adequate data base of washing unit performance is available, simulation techniques can be applied to washability results from much smaller samples. Simulation is faster, less expensive and more accurate than experimental techniques.

There are several models which are available in simulation packages. To identify the simulation of the operation, units separating particles on the basis of their relative density differences are used.

"Zitwash", developed by Mr Zalman Zitron was used to simulate the results of washability studies on the middlings and discard which were crushed down to 0.1 mm size fraction. Zitwash uses Epm in conjunction with washability data to simulate and compare the recovery efficiency of different units.

4 RESULTS AND DISCUSSION

As previously discussed, four plant products were used to identify the optimum beneficiation routes for Waterberg coal in this project. All four products were crushed to −15 mm and then screened into the four sizes mentioned in the chapter 3.4. The methods used to beneficiate each size fraction of each sample were given in chapter 3.5. The simple regression analysis is carried out (mean particle size vs yield and mean particle size vs ash) for the prediction of the results. The quality of the regressions can be judged by several criteria. One of them is R^2, the multiple correlation coefficient, given in this section. It is defined as the sum of squares due to the regression divided by the total corrected sum of squares. It should be emphasised that the quality of the regression depends on the quality of the original data. All regression analyses are given in **Appendix C**. In this section the results of experiments carried out on each product are given separately.

4.1 Presentation of Results

RAW COAL

All samples were crushed to a nominal −15 mm at the Grootegeluk plant: this was confirmed by screen analysis. The results of the screen analysis and the way of producing the sizes used to provide washability curves are reported in **fig 4.1**.

The particle size distribution in fig 4.1 shows that 77,3% of raw coal at 43,2% ash content was accumulated in 15x1 mm size fraction. The remainder had an ash content of 33,6% and was finer than 1 mm.

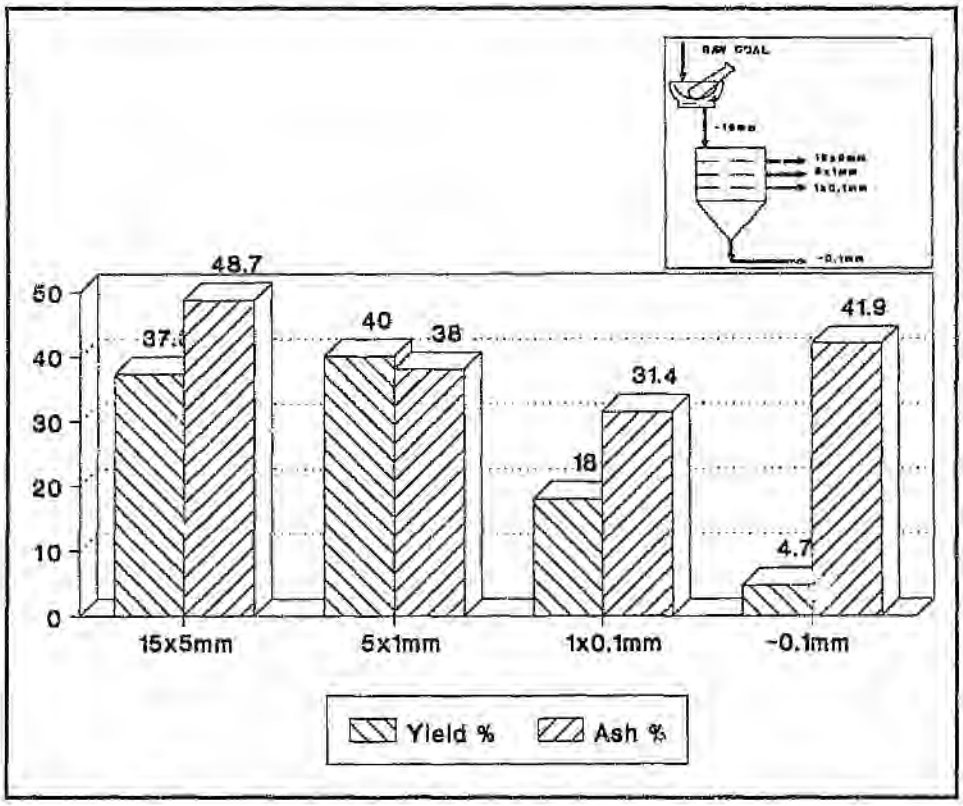


Figure 4.1 Particle size distribution of raw coal

Each size fraction was beneficiated using the methods mentioned in chapter 3.5. The washability data on 15x5, 5x1, 1x0.1, -0.1 mm size fractions of raw coal are given in table D.1, D.2, D.3, D.4, of **Appendix D**. The results of surface property dependent methods applied to –0.1 mm size fraction are given in table D.5, D.6 of Appendix D. The effect of washing on different size fractions of raw coal are represented with washability curves, in other words, these curves are based upon fractions from the same sample. **Fig 4.2** and **Fig 4.3** show the washability curves on size fractions of raw coal. Cumulative ash curves are given in fig 4.2 and densimetric curves are given in fig 4.3.

The trend of these curves in fig 4.2 shows that the effect of liberation starts being discernible on the 5x1 mm size fraction. From the 5x1 mm size fraction, washed coal can be obtained with an ash content of around 14% at 1,7 relative density.

The washability curves on 1x0.1 mm of raw coal show the significance of liberation. It is possible to obtain low ash coal with 67.5% yield at relative density of 1,7.

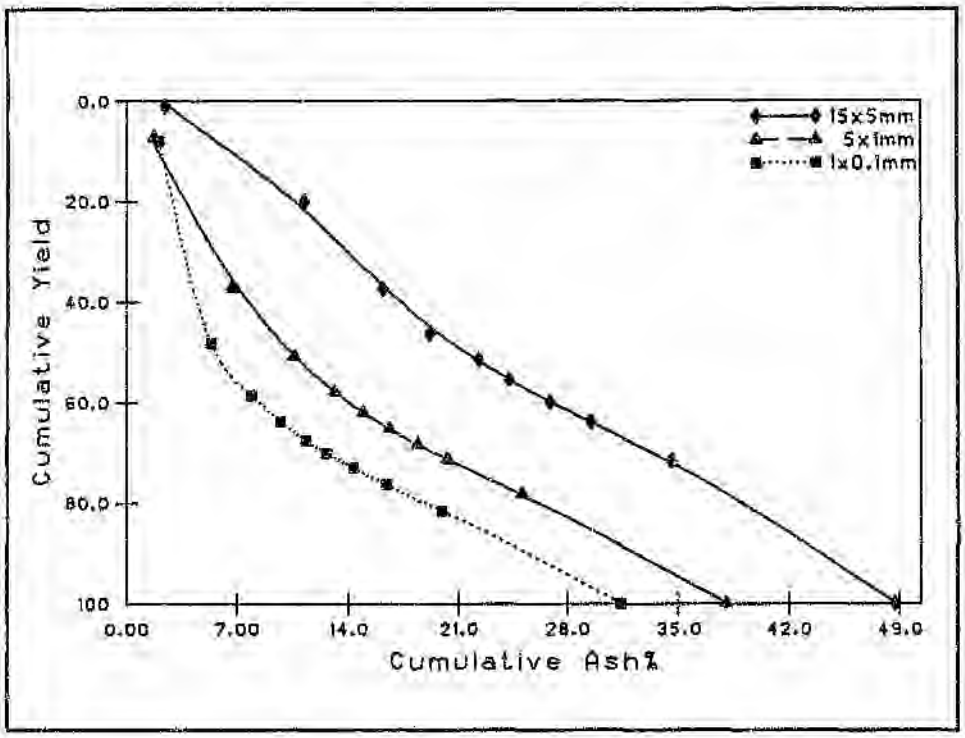


Figure 4.2 Cumulative ash curves of raw coal

The cumulative ash curves on -0.1 mm do not show the same trend of increasing the yield and decreasing the ash as demonstrated on the coarser sizes. The comparison between the ash content of each size fraction (see fig 4.1) explains this extreme situation.

All the data given in figures such as mean particle size vs yield, and mean particle size vs ash content at constant density are applied to simple regression to determine the correlation between these two variables with confidence. A linear model below is derived from the data for two variables.

$$Y = a + bx$$

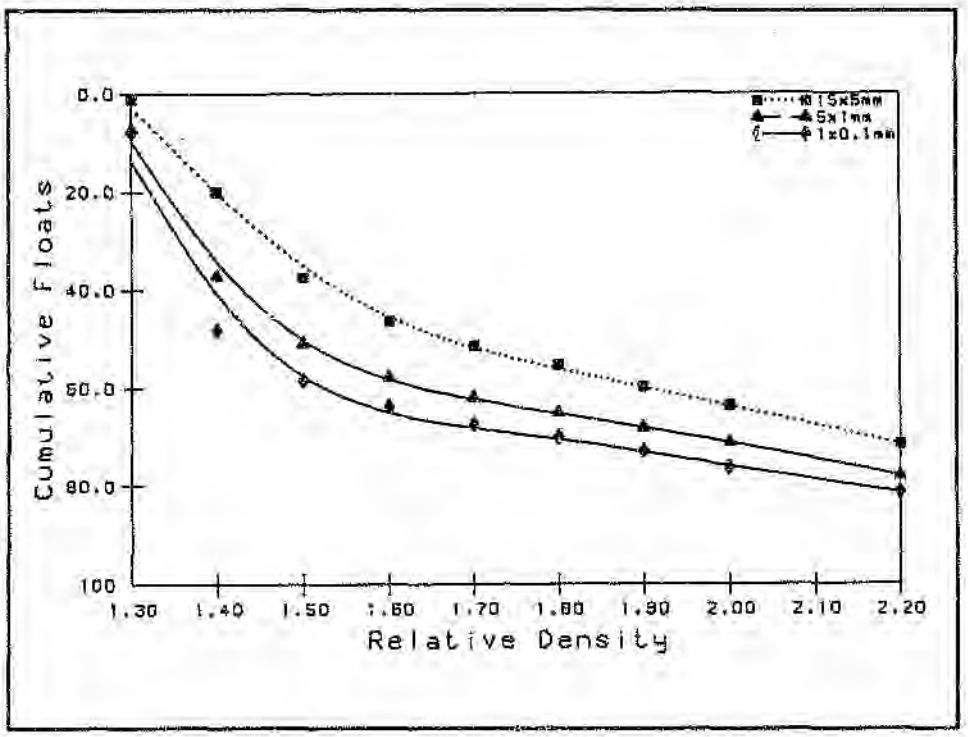


Figure 4.3 Densimetric curves of raw coal

The washability data calculated from the model at the usual plant density of 1,7 and R^2 are given in **table 4.1** so that the quality of regressions can be judged and compared.

Table 4.1 The effect of washing on raw coal and R^2 value

Size (mm)	Mean Size	R.D	Ash %	Yield %	R^2 (Mps vs Ash)	R^2 (Mps vs Yield)
15x5	8.7	1.7	22.6	38.5	96.14	31.0
5x1	2.2	1.7	14.7	60.5		
1x0.1	0.3	1.7	12.4	67.0		
-0.1	0.05	1.7	12.1	67.8		

Table 4.1 shows that there is a very good correlation between mean particle size and ash. However mean particle size and yield do not correlate well.As seen from the data calculated by linear model in table 4.1 the increase in yield and reduction in ash indicates that liberation becomes significant <5 mm. This table also shows that it is possible to obtain 59.7% coal at 15% ash content by single-stage washing from the 15x1 mm size fraction of raw coal. The remainder of this size (50.9%) has an ash content of 40.9%. As mentioned before -0.1 mm size fraction was also subjected to froth flotation and oil agglomeration tests.

Table 4.2 Comparison between the recovery of the methods used on -0.1 mm

Methods	Yield %	Ash %
Froth Flotation	22.0	11.2
Oil agglomeration	51.9	9.1

Table D.5 and D.6 of Appendix D show the results of surface dependent method on Raw Coal. As seen in the table above, regression analyses were not carried out on the results of the froth flotation and the oil agglomeration test, because of insufficient data. The table D.6 shows it is possible to obtain 51.9% coal at an ash content of 9.1% by adding 15% oil and agitating for 30 minutes. Likewise it is demonstrated that we can obtain 37.2% coal at an ash content of 6.1% with the same oil consumption in 10 minutes. These results show that the increase in time affects the increase in yield and increase in ash. The comparison between the oil agglomeration results and the results given in table 4.2 indicates that oil agglomeration is the best treatment on this particular size fraction. However it is expensive. The superiority of oil agglomeration can be explained by understanding its principle. In the oil agglomeration process, oil drops only coat the water-repelling particles which then adhere to each other and form larger clumps. No material other than coal-containing material is agglomerated. Likewise in froth flotation only organic particles adhere to the air bubble and they are lifted up.

When two coal particles are lifted up by the air-bubble, they may also lift inorganic particles between them. This is known as entrainment.

TESKA PRODUCT

The same crushing and screening procedure applied to raw coal is also applied to the Teska product. **Fig 4.4** shows the particle size distribution and ash percentage of each size from the Teska product. As seen from the histogram 85.1% or Teska product is coarser than 1 mm and 14.9% of it is finer than 1 mm. The <1 mm size fraction has a lower ash content than the size fraction coarser than 1 mm.

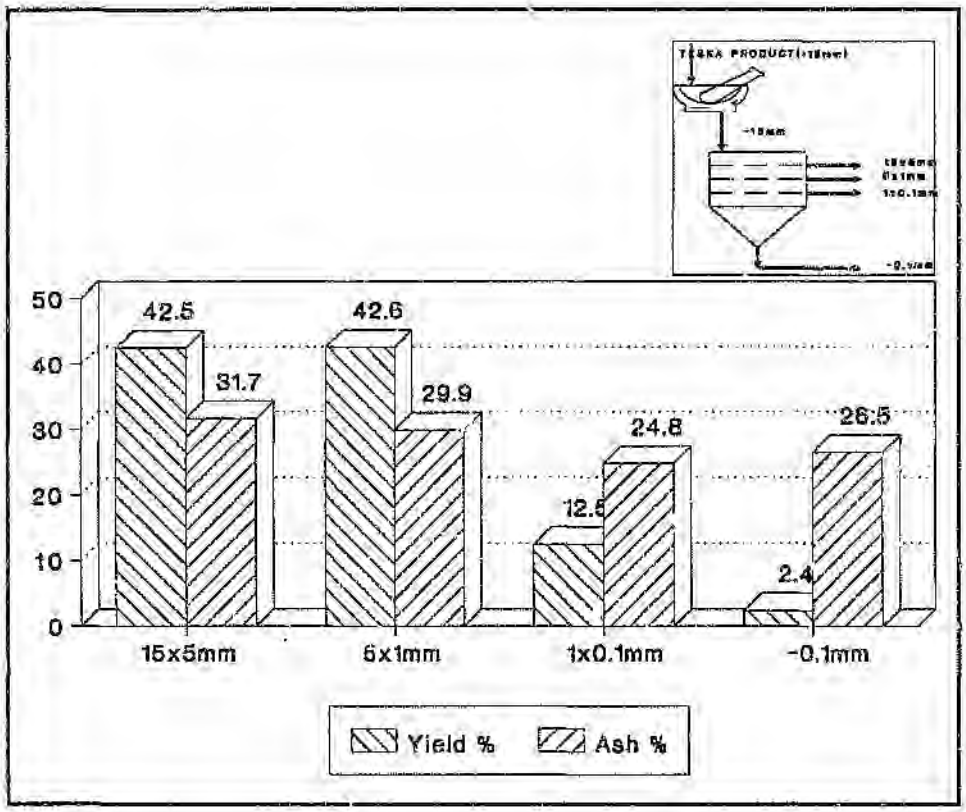


Figure 4.4 Particle size distribution of Teska product

The washability data on 15x5, 5x1, 1x0.1,-0.1 mm size fractions of the Teska product are given in table E1, E.2, E.3, E.4 of **Appendix E.** The results using surface properties dependent methods on -0.1 mm size fraction of the Teska

product are given in Table E.5, E.6 of Appendix E.

In this section the effect of washing on the Teska product is represented by washability curves in **fig 4.5** and **4.6**. Fig 4.5 shows the cumulative ash curves on the size fractions of the Teska product. The curve on 15x5 mm size fraction shows that the lowest ash content produced is 14% in this size fraction.

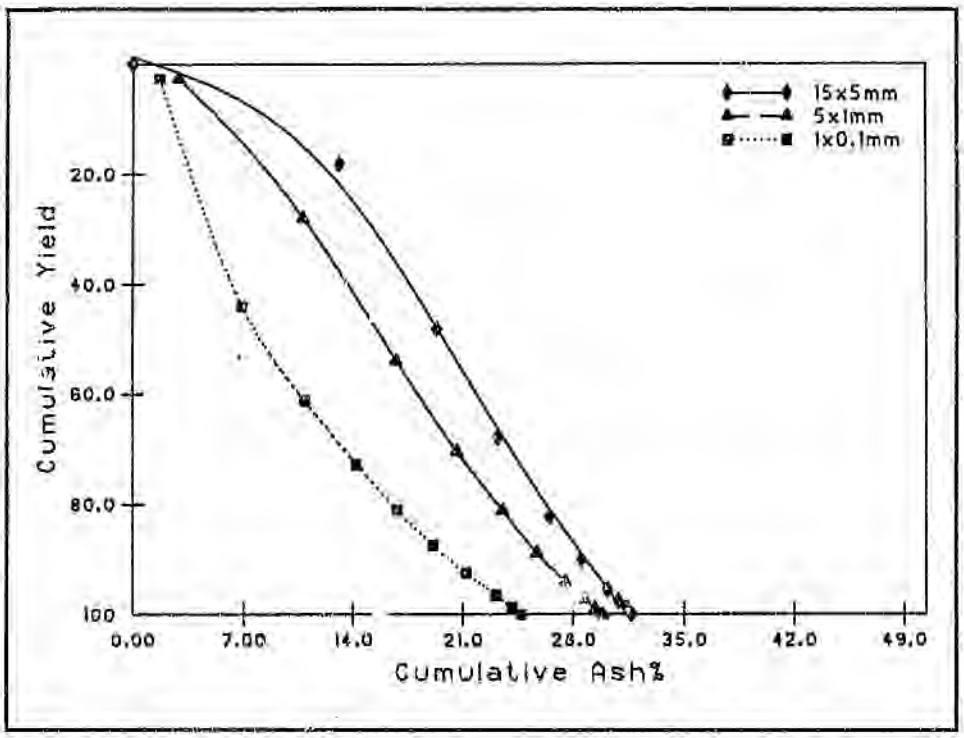


Figure 4.5 Cumulative ash curves on Teska product

The 5x1 mm cumulative floats curve indicate that liberation makes a significant difference to the ash content of the product. In this size fraction, it is possible to obtain 27.9% coal at an ash content of 10.8% at the usual plant density of 1,4. Fig 4.5 and 4.6 show the washability curves on 1x0.1 mm of the Teska product and reveal the possibility of obtaining the coal at an ash content of less than 10%. As seen in the curves, it is possible to obtain approximately 7% ash coal with 44% yield at the same density.The washability data on -0.1 mm in **fig 4.10** shows the

same trend of increasing the yield. These data show that the ash content of the product from -0.1 mm remains the same but the yield increases.

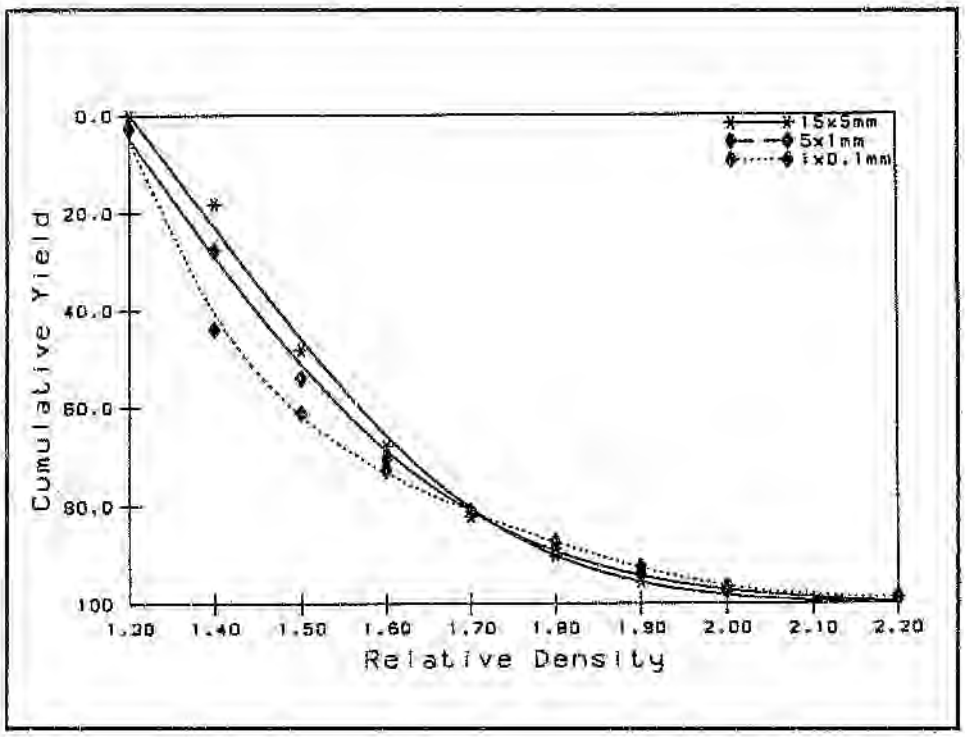


Figure 4.6 Densimetric curves on Teska product

The simple regression analysis is carried on mean particle size vs yield and also mean particle size vs ash content. From regression, an exponential model for both yield and mps vs ash are derived.

$Y = \exp(a + bx)$ Exponential Model

The washability data calculated from models and R_2 at the usual plant density of 1,4 are given in the **table 4.3.**

These results indicate that there is a very good correlation between two variables and also liberation becomes significant below 5 mm size fraction. Crushing to -5 mm will decrease the ash. It is obvious that producing coal of low ash content

with define yield is only possible if all the coal is crushed to below 15 mm. This is in fact current Iscor practice.The results of the Teska Discard confirmed that there is no point in crushing to -15 mm before the first stage.

-0.1 mm size of the Teska product was also subjected to froth flotation and oil agglomeration tests. The results obtained from these methods are given in **table 4.4**.

Table 4.3 The effect of washing on Teska product and R^2 value

Size (mm)	Mean Size	R.D	Ash %	Yield %	R^2 (Mps vs Ash)	R^2 (Mps vs Yield)
15x5	8.7	1.4	11.2	26.9	88.5	97.9
5x1	2.2	1.4	10.3	30.8		
1x0.1	0.3	1.4	9.1	36.6		
-0.1	0.05	1.4	8.0	41.7		

Table 4.4 The comparison between the recovery of methods used on -0.1 mm

Methods	Yield %	Ash %
Froth Flotation	70.8	13.5
Oil Agglomeration	48.6	8.0

Table 4.4 indicates that oil agglomeration yields better results. The results using surface property dependent methods (froth flotation and oil agglomeration) show that producing coal at a low ash content and with high yield is possible. Therefore crushing the Teska float to -5 mm would increase the yield and reduce the ash

appreciably. The results on 1x0.1 mm and -0.1 mm size fractions show that it is possible to obtain the coal at an ash content of less than 10%.

MIDDLINGS

One of samples treated was the middlings was the main area of the study, the aim being to produce low ash coal. To accomplish this the sample was crushed and screened, and subjected to float and sink analysis and surface property dependent methods. The work carried out on middlings varied slightly from the previous analyses. It involved crushing the intermediate sizes ever more finely. First it was crushed to -15 mm then screened into four size fractions (15x5, 5x1, 1x0.1, -0.1). Each size fraction was crushed to the next fine size and screened. This was repeated until all the samples were crushed to -0.1 mm. The procedure of screening and crushing is given in fig 4.7.

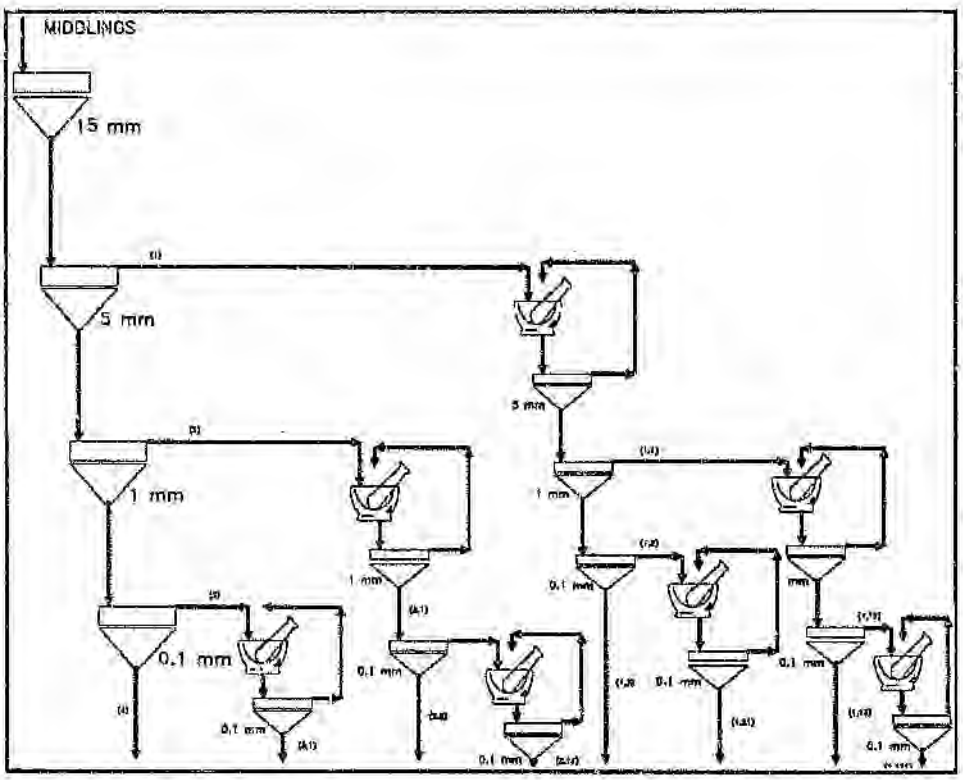


Figure 4.7 Procedure of screening on middlings

Each individual size fraction was subjected to the float and sink analysis.The washability data on all the sizes, including intermediate sizes, are given in the tables from F.1 to F.15 of Appendix F. The results using surface property dependent methods on the -0.1 mm size fraction of middlings are given in table F.16 ,F.17 of Appendix F. Fig 4.8 shows the particle and ash distribution on sub-sizes of 15 mm top size. It is only given on one top size as the others would be similar. **Fig 4.8** shows that 88.5% of middlings at 33.2% ash content are over 1 mm. 11.8% coal at an ash content of 41.7% is under 1 mm.

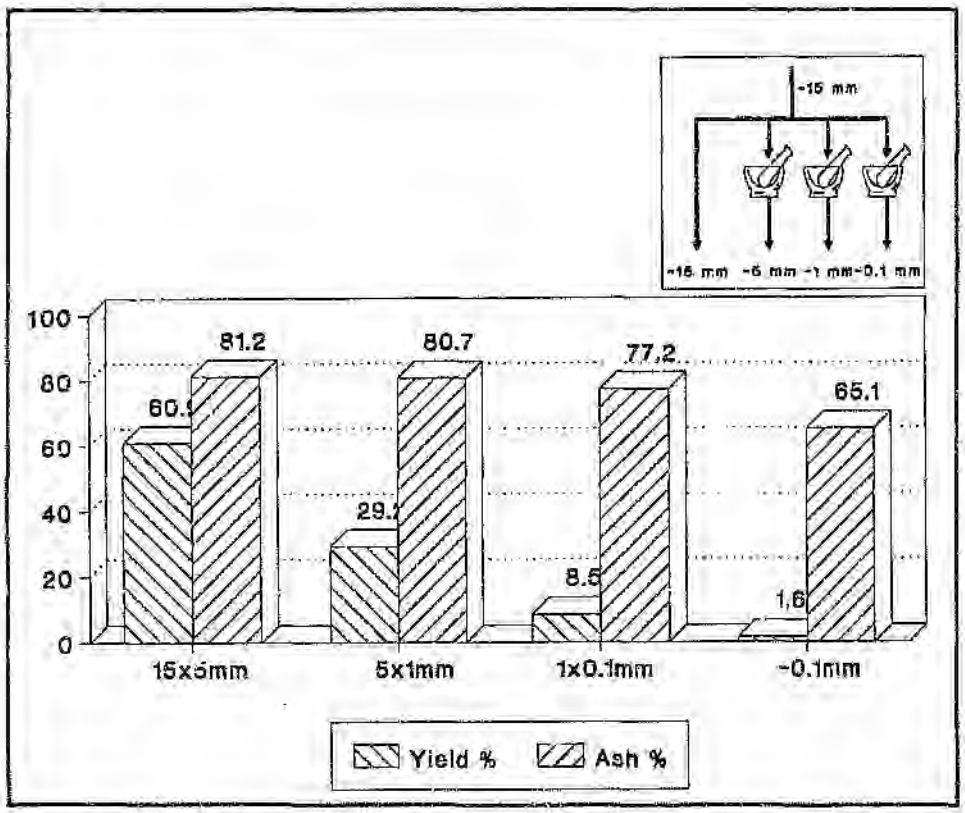


Figure 4.8 Particle distribution of middlings

The washability data given in Appendix F is combined to determine the liberation behaviour of four different top sizes (15x0, 5x0, 1x0, -0.1x0). The washability characteristics of middlings given in **fig 4.9** and **fig. 4.10** show the effect of washing on top sizes of middlings.

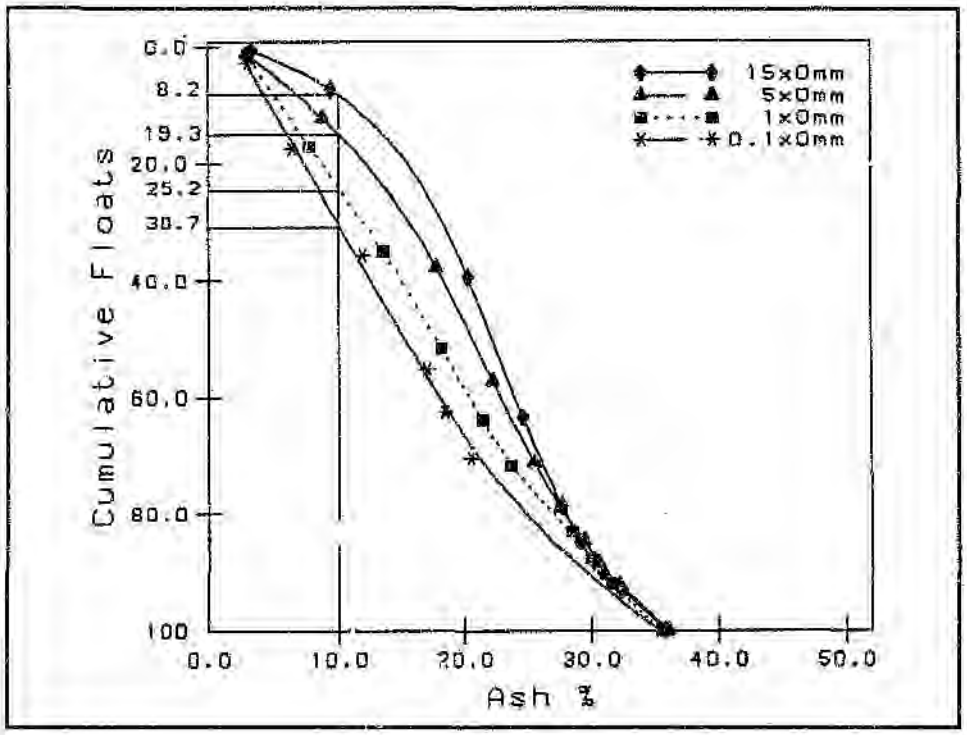


Figure 4.9 Liberation characteristics of middlings

First curve shows the cumulative floats on 15x0 mm top size of middlings. This curve shows that it is possible to get 50% of coal at 22% ash from 15x0 mm. This indicates that the effect of liberation on this size is not significant. The 5x0 mm curve gives a similar impression to 15x0 mm curve indicating that the liberation here is not significant. The washability curves on 1x0 mm size of middlings, as seen from the curves, indicated that the effect of liberation becomes noticeable on this top size. It is possible to obtain 17.5% ash coal with 50% yield.The washability curves on the finest top size of middlings show the trend of decreasing ash and increasing yield with diminishing size.

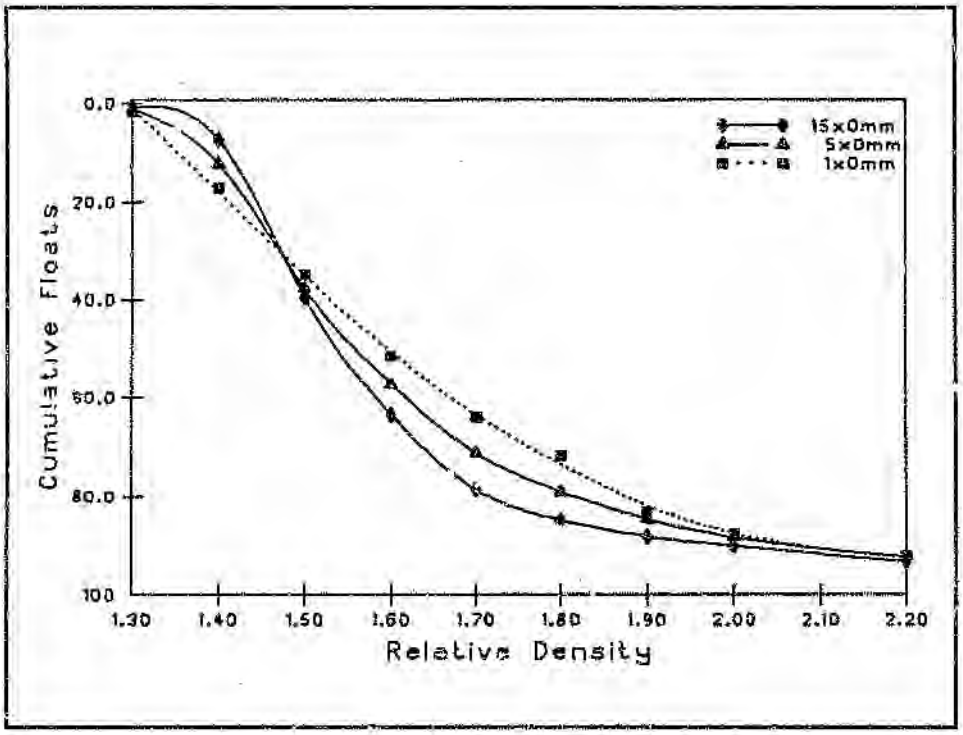


Figure 4.10 Densimetric curve on middlings

The washability results of 15 mm top size of middlings are applied to simple regression. Regression analysis is not carried on the finer top sizes, because of insufficient intermediate size fraction. An exponential model for mean particle size vs yield and a linear model for mps vs ash are derived for the coarsest top size of middlings.

$Y = \exp(a + bx)$ Exponential Model

$Y = a + bx$ Linear Model

The washability data calculated from models and R^2 in **table 4.5**.

Table 4.5 The effect of washing at 1.4 R.D on intermediate sizes of 15 mm and R^2 value

Size (mm)	Mean Size	R.D	Ash %	Yield %	R^2 (Mps vs Ash)	R^2 (Mps vs Yield)
15x5	8.7	1.4	16.8	9.0	74.2	97.2
5x1	2.2	1.4	9.5	13.4		
1x0.1	0.3	1.4	7.4	19.6		
-0.1	0.05	1.4	7.1	25.3		

The table above gives some idea of the effect of liberation. It also shows that mean particle size, yield and ash correlate each other quite well. From the table, it is clear that increase in yield and decrease in ash content is possible with diminishing size.

It is significant that the 15x0 top size gave about 8.2% of 10% ash; whilst 1x0 mm material gave 25.2% of 10% ash. The results of surface dependent methods of a -0.1 mm are given in **table 4.6**.

Table 4.6 The comparison between the recovery of the methods used on -0.1 mm

Method	Ash %	Yield %
Froth flotation	17.0	54.5
Oil agglomeration	9.0	56.8

The results on the -0.1 mm size fraction by surface property dependent methods show that the oil agglomeration process gives the best results for the ultrafine fraction.

TESKA DISCARD

The work carried out on the discard was similar to that on middlings. The discard

was also crushed down to -0.1 mm but each intermediate fraction was not subjected to the washability studies. It is assumed to behave similarly to the intermediate size fraction of -15 mm.

The ash and particle distribution on sub-size of 15 mm top size are given by histogram in **fig 4.11.** It shows that 90.1% of it at 81.0% ash is accumulated in 15x1 mm. 10.4% of it at 73.1% ash is under 1 mm.

The washability data on the all sizes are given in table G.1, G.2, G.3., G.4 of **Appendix G.** The results using surface dependent methods are given in table G.5, G.6. **Fig 4.12 and 4.13** show the washability curves on discard. The 15x5 mm size fraction enables us to obtain 40% ash coal but only 4% yield. As seen from this figure the effect of liberation is quite poor.

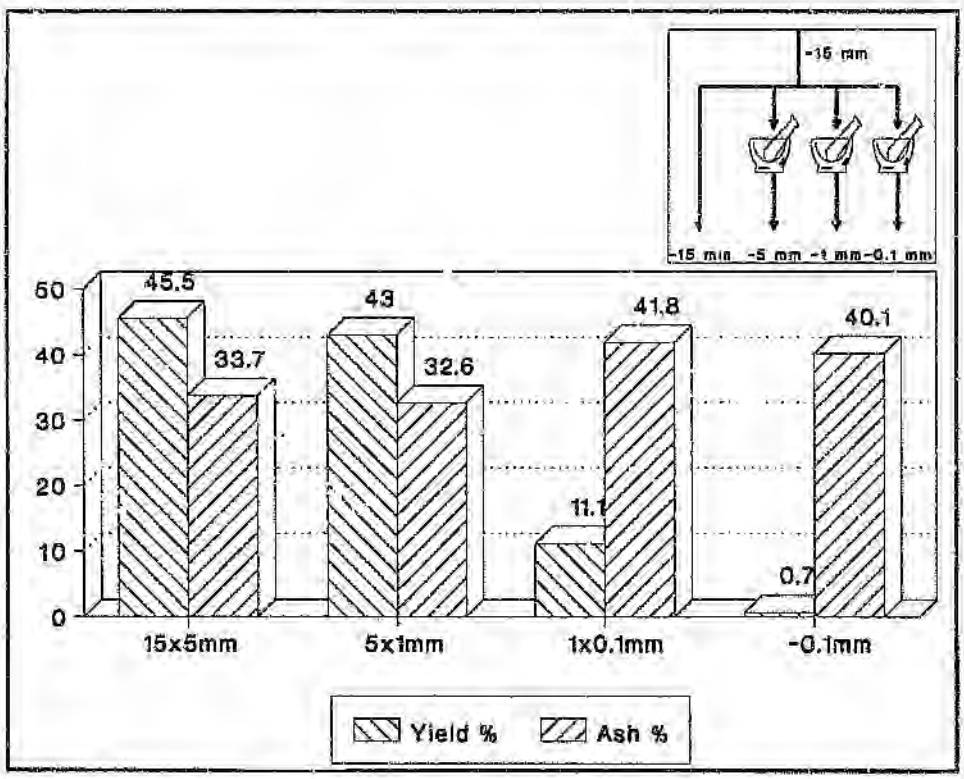


Figure 4.11 Particle size distribution of discard

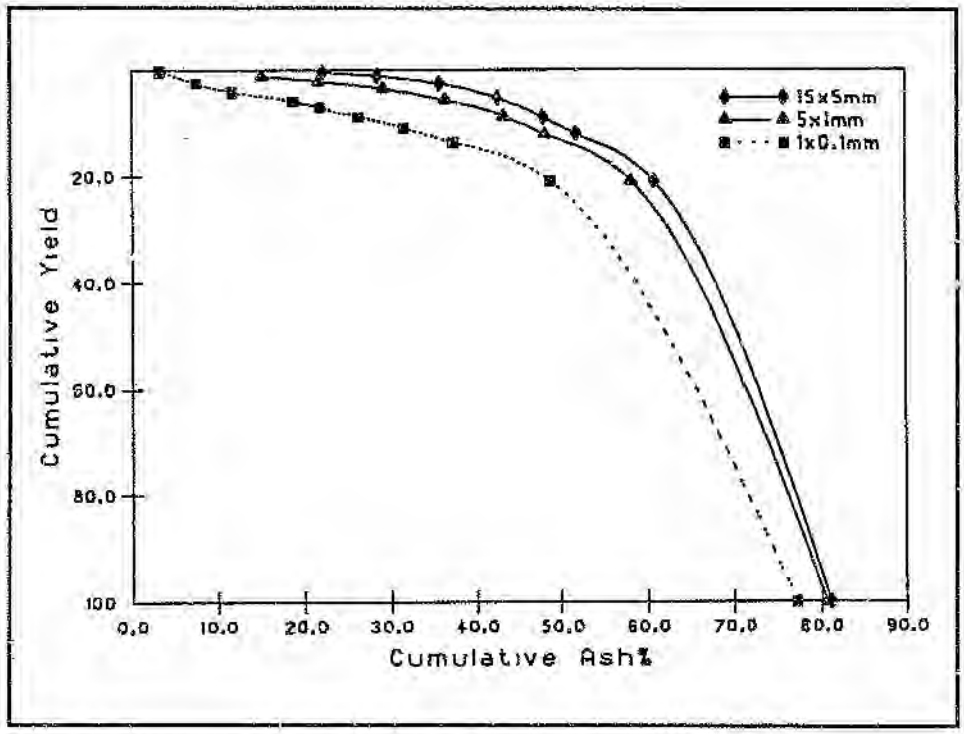


Figure 4.12 Cumulative ash curves on discard

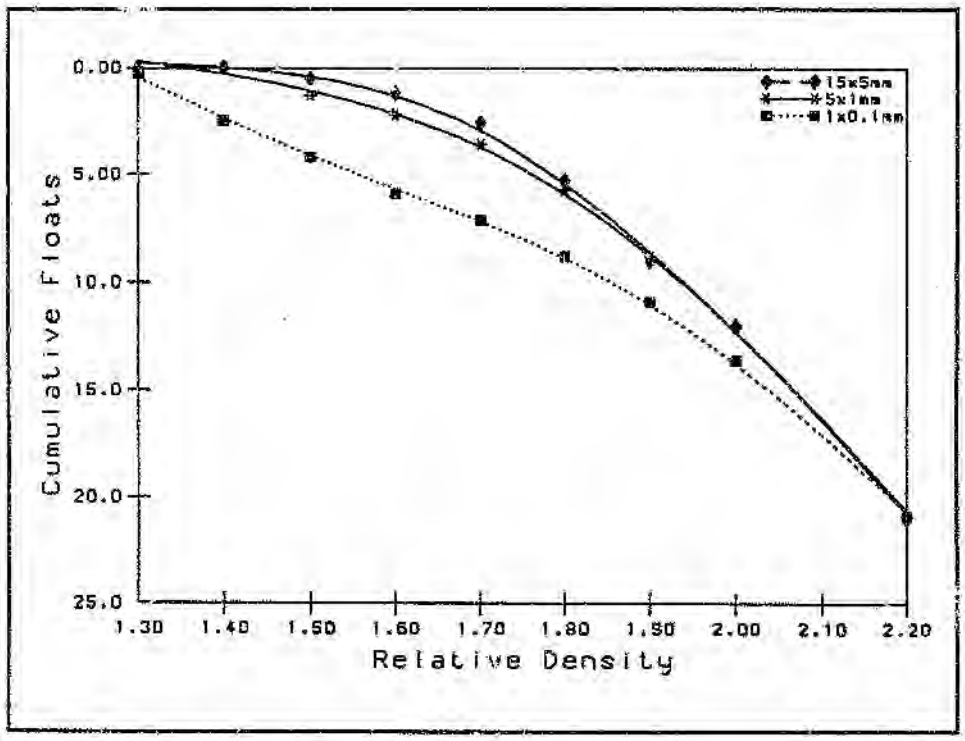


Figure 4.13 Densimetric curve on discard

The washability curves on the second coarse size fraction of discard show that it is possible to obtain 7% of coal at an ash content of 30%. This indicates that there is not much increase in liberation in 5x1 mm size fraction. The 1x0.1 mm curves indicate that at this size it is possible to obtain 12% power station coal at 35% ash from the discard. From the washability data, it is possible to obtain 35% ash coal with 45% yield from -0.1 mm of discard.

The results of regressions on the washability data from Teska discard are given in **table 4.7**. A reciprocal and a multiplicative models are derived from the laboratory data.

$1/Y = a + bx$ Reciprocal Model

$Y = ax^{b}$ Multiplicative Model

Table 4.7 The results of regression at 1.8 R.D on intermediate sizes of 15 mm and R^2 value

Size (mm)	Mean Size	R.D	Ash %	Yield %	R^2 (Mps vs Ash)	R^2 (Mps vs Yield)
15x5	8.7	1.8	42.5	5.4	97.6	100.0
5x1	2.2	1.8	35.8	5.7		
1x0.1	0.3	1.8	27.8	8.7		
0.1	0.05	1.8	22.2	26.1		

Table 4.7 shows the very good correlations between variables. As seen in the above table, it is possible to obtain power station coal with a reasonable yield only if the discard is crushed to below 1 mm.

The results of surface dependent methods on -0.1 mm of discard are given in **table 4.8.**

Table 4.8 The comparison between the recovery of methods used on -0.1 mm

Method	Ash %	Yield %
Froth Flotation	32.5	27.8
Oil Agglomeration	20.7	29.3

The table above and the table in Appendix G shows that it is possible to obtain 29.3% coal at 20.7% ash using 20% oil, while 25.7% coal at 15.6% ash is obtained using 10% oil.The results of the froth flotation and oil agglomeration both show that the recovery of power station coal from discard is similar.

4.2 Simulation and Cost Estimation of Real Washing

Why Simulation ?

All the float and sink data presented so far is for virtually perfect separation. If high grade coal is to be recovered from say middlings, the effect of real washing must be examined. If sufficient coal is available this may be achieved by carried out an actual washing test in say a pilot plant. Such work is beyond the capabilities of WITS research facilities. However, using computer based techniques, SIMULATION of washing may be carried out. Based on comparisons of simulated and actual washing tests, such simulation relates closely to real washing conditions. Essentially simulation consists of allying the washing characteristics of a washing unit defined by the partition curve with this coal washability.

Derivation of Flowsheet

In carrying out simulation it is necessary to work to a specific flowsheet. This was derived from the analytical data presented in the previous section, showing the degree of liberation achieved at various top sizes. The simulation then indicates the most suitable process to exploit such liberation. Simulation

techniques permit several progress to be compared in terms of recovery. Such comparison may then be limited to fractional estimates of the capital and operating costs.

Units Simulated

-15 mm fractions

15x0.1 mm size fraction could be washed in either a dense medium cyclone or on a fine coal jig. Therefore simulation was carried out at the Epm expected from these plants.

The simulation data are given in tables H.1 to H.6 of Appendix H for middlings and discard.

The simulation data showed that near-density material ranged from 28% to 50%. The range of near-density material to operate a jig should be under 10%. Jig treatment was discarded for this size fraction, in favour of a dense medium cyclone.

-5 mm fractions

5x0.1 of 5 mm top size was treated similarly.

-1 mm fractions

The 1x0.1 mm fraction of the 1 mm top size was simulated for a spiral plant and also for a dense medium cyclone. Simulation data showed that the material could not be treated in a spiral because of the high range of near-density material and high ash content of product. This size should also be treated by DMC.

-0.1 mm fractions

-0.1 mm size fraction of 15, 5, 1 top sizes were treated either by froth flotation or oil agglomeration. Here actual lab data was used. The amount of sample available was adequate for such tests. Moreover, the results obtained are close to results

obtained from full scale plant, as the test procedures are similar to full scale operation.

Several

It is not claimed that any of the flowsheets adopted are the best and especially that any of them, in the current economic climate, is capable of being used commercially.

Cost Estimation

The objective is to use cost effective processes at all stages, there being a hierarchy of costing essentially dictated by particle size, difficulty of washing, and unit performance. Processes using density differences are used to the maximum possible extent, and the more costly surface-property dependent processes only as necessary. An approximate mathematical relationship between processing cost and particle size is given below,

$$\text{Processing Cost} \propto 1/\sqrt{mps}$$

With these terms of reference, the flowsheets employ one or two stage crushing dependent on the top size, dense medium cyclones and oil agglomeration or froth flotation to produce low ash coal from middlings fractions and additional power station coal from washery discard.

Method of Costing

Available factorised data was used to calculate the capital and operating cost of a possible new plant.

The capital and operating costs data for the units used to beneficiate the middlings at three top sizes are given in **table 4.18.**

The following equations are used to calculate other costs.

Maintenance, Labour and Supplies = 0.15 x Capital cost

Operating Supplies = 0.15 x Maintenance Labour and Supplies (Ruhmer, 1987)

The interest rate of 25% and pay back period from 2 to 5 years were chosen to calculate the price of low ash coal from middlings and middlings from discard. The capital and operating costs were calculated using the above data for the flowsheets which were to produce a better quality product from middlings and discard. Calculation of costs is given in Appendix I.

Table 4.9 Factorized data used to calculate the capital and operating cost of possible new plant (Voges, 1989; Du Preez, 1989)

UNIT	SIZE RANGE	CAPITAL COST R/t/h	OPERATING COST R/t
Crushing	-5 mm from 15 mm	2 300	0.35
Grinding	-1 mm from 15 mm	16 460	0.50
Grinding	-0.1 mm from 15 mm	55 800	1
Grinding	-0.1 mm from 5 mm	55 800	1
Grinding	-0.1 mm from 1 mm	50 000	1
Classification(Hydro Cyclone)		4 700	*
Separation(Dense Medium Cyclone)		54 000	1.7
Filtration	20-0.5 mm 0.5-0.05 mm	1 050 116 000	0.09 0.44
Froth Flotation	-0.1 mm	72 000	2.39
Oil Agglomeration	-0.1 mm	36 000	7.17
Thickener		53 200	*

* The operating cost for classification and tails thickener is assumed to be R3m

The results of simulation and costing are discussed separately on middlings and discard.

4.3 Cost Estimation

4.3.1 Middlings

The production of low ash coal from middlings with a top size of 15 mm and 5 mm is shown in **figure 4.14** and **4.15**. The envisaged treatment employs one crushing stage for the treatment of 15 mm top size (see fig 4.14) and two crushing stages for the treatment of 5 mm and 1 mm top size. As seen from the flowsheets, crushed material is treated in a dense medium cyclone. The cyclone floats may be used as metallurgical coal. The cyclone sinks are ground to -0.1 mm then treated by oil agglomeration or froth flotation to produce low ash coal from middlings. **Fig 4.15 and 4.16** show that there are two stages in these flowsheets. According to the flowsheets, the middlings is crushed to -5 mm and treated in a dense medium cyclone. The cyclone floats may be used as metallurgical coal. The cyclone sinks are ground to -0.1 mm, then treated by oil agglomeration or froth flotation.

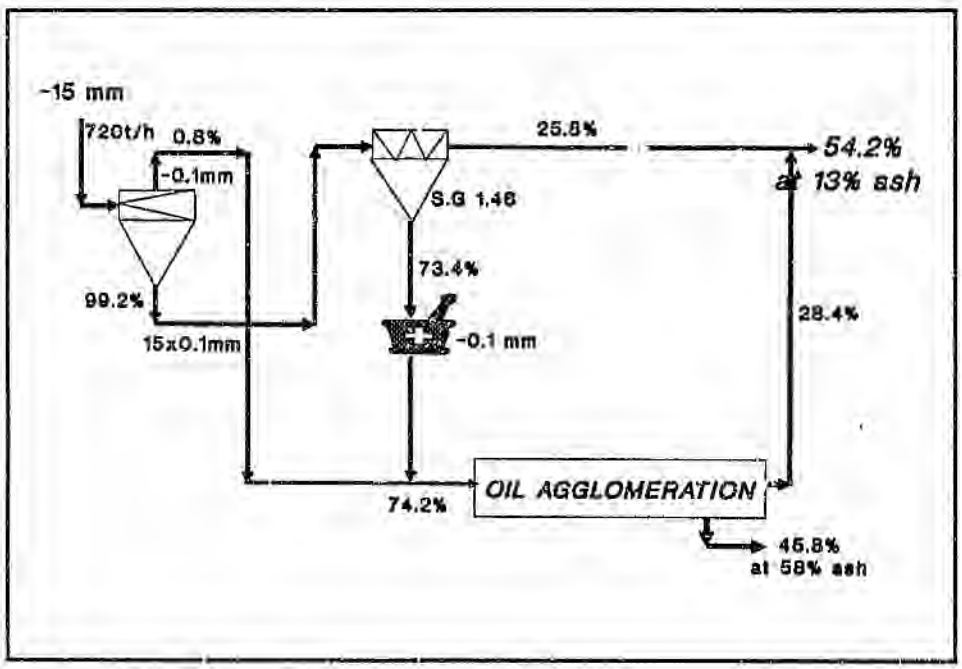


Figure 4.14 Production of low ash coal from middlings (15 mm top size)

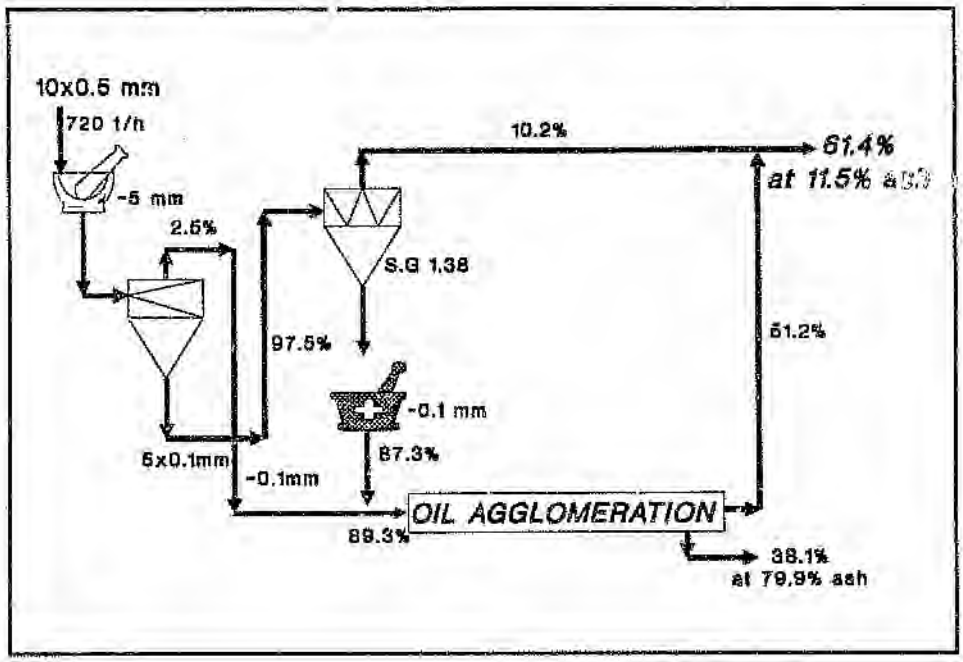


Figure 4.15 Production of low ash coal from middlings (5 mm top size)

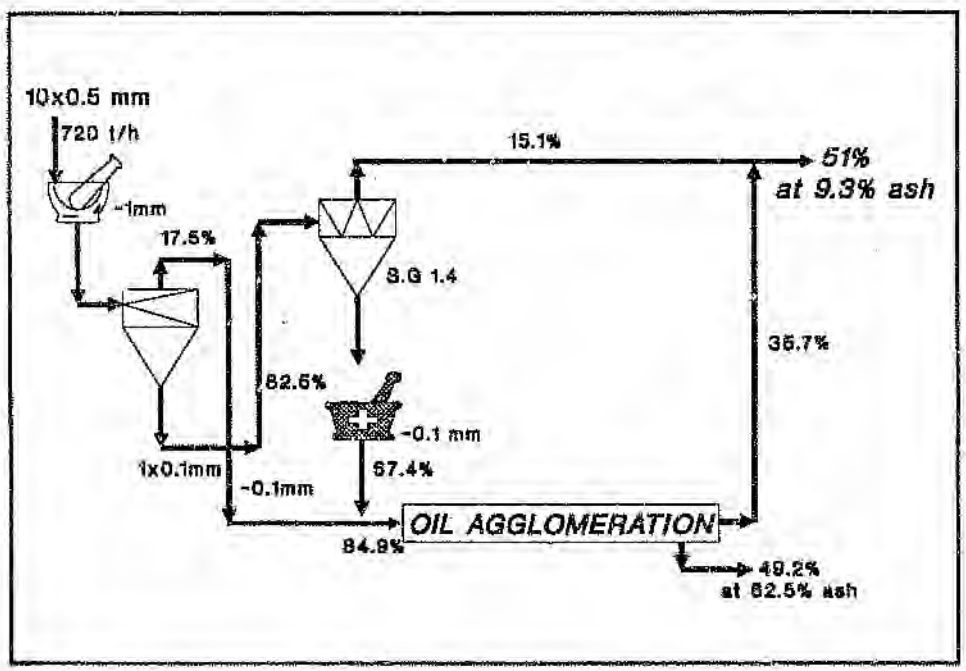


Figure 4.16 Production of low ash coal from middlings (1 mm top size)

The product yields for different top sizes treated as shown in the schematic flowsheets, are given in **figure 4.17**. Using oil agglomeration on the -0.1 mm fraction obtained from the three top sizes (15,5,1 mm) of middlings, fig 4.17 demonstrates that it is possible to get 54% coal at an ash content of 13% from the -0.1 mm fraction of middlings of 15 mm top size. Similarly, from the 5 mm top size we can obtain 61% coal at an ash content of 11,5% in the -0.1 mm fraction. The lowest ash percentage is obtained on -1 mm top size. The second chart for froth flotation shows that the product ash content ranges from 15 to 17 percent.

The capital and operating costs to produce low ash coal from different top sizes of middlings are calculated as described above and given in **table 4.14**.

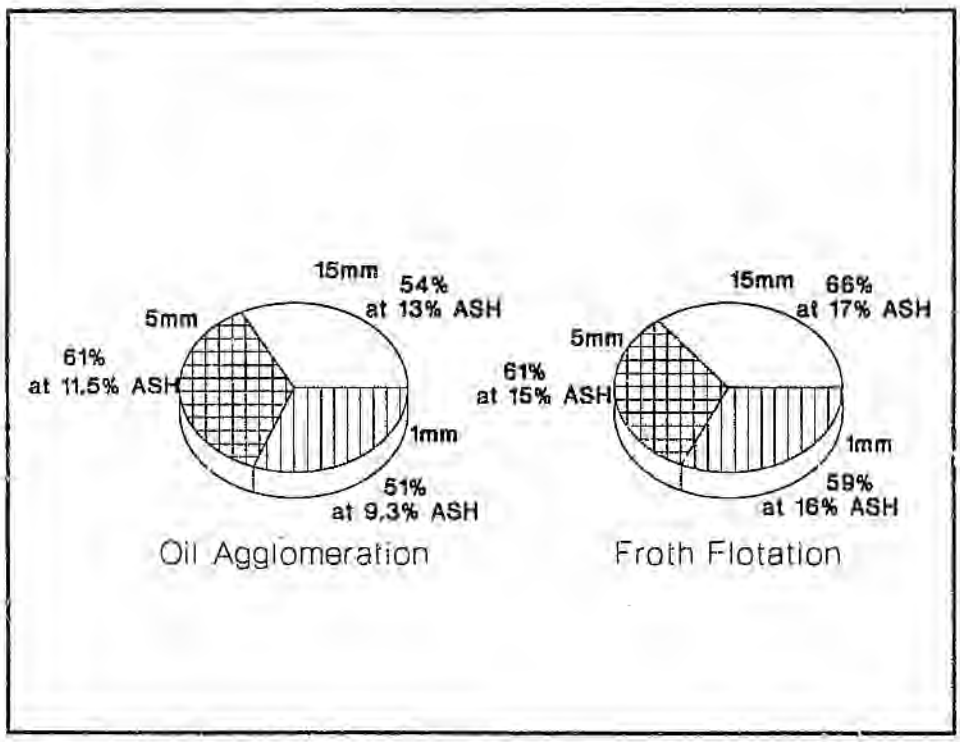


Figure 4.17 The results of retreatment of middlings; top size; method of fine cleaning; overall yield percent and overall ash percent

Table 4.14 The capital and operating cost of producing low ash coal from middlings

	BENEFICIATION TECHNIQUES			
	DMC & FROTH FLOTATION		DMC & OIL AGGLOMERATION	
TOP SIZES(mm)	Operating Cost Rm	Capital Cost Rm	Operating Cost Rm	Capital Cost Rm
15	53.8	194.31	178.8	64.4
5	225.6	60.5	200.27	74.2
1	224.6	61.04	205.6	69.9

The price of low ash coal produced from middlings with pay back period are given in table 4.15.

As seen in table 4.15, producing low ash coal from middlings by oil agglomeration is more expensive than by froth flotation. However, although the cost of froth flotation is much lower than that of oil agglomeration, the quality of the oil agglomeration product is always much better than that of the froth flotation product. If the coal is used subsequently for direct liquefaction the effect of oil cost on beneficiation may be materially diminished.

Table 4.11 The price of low ash coal produced from middlings

TOP SIZES(mm)						
	15		5		1	
Period to pay back (year)	Price R/t *1	Price R/t *2	Price R/t *1	Price R/t *2	Price R/t *1	Price R/t *2
2	72	90	94	90	90	109
2.5	63	80	82	80	78	96
3	56	73	74	73	70	88
3.5	52	68	68	68	65	81
4	49	64	63	64	51	77
4.5						73
5						70

*1 Price of Coal produced by froth flotation

*2 Price of Coal produced by oil agglomeration

4.3.2 Discard

The method of producing additional power station coal from discard is similar to the way used to produce low ash coal from middlings. The proposed flowsheet for 15 mm top size is given in **figure 4.18. Figure 4.19** and **figure 4.20** show the proposed flowsheet for 5 and 1 mm top sizes.

The flowsheets given in figure 4.16 ,4.17 and 4.18 are similar. They all employ two-stage crushing, dense medium cyclones and oil agglomeration or froth flotation to produce additional power station coal from the washery discards.

As seen in the flowsheets, crushed material is treated in dense medium cyclones. The cyclone floats are ground to -0.1 mm then treated by either oil agglomeration or froth flotation. The cyclone sinks are removed as discard.

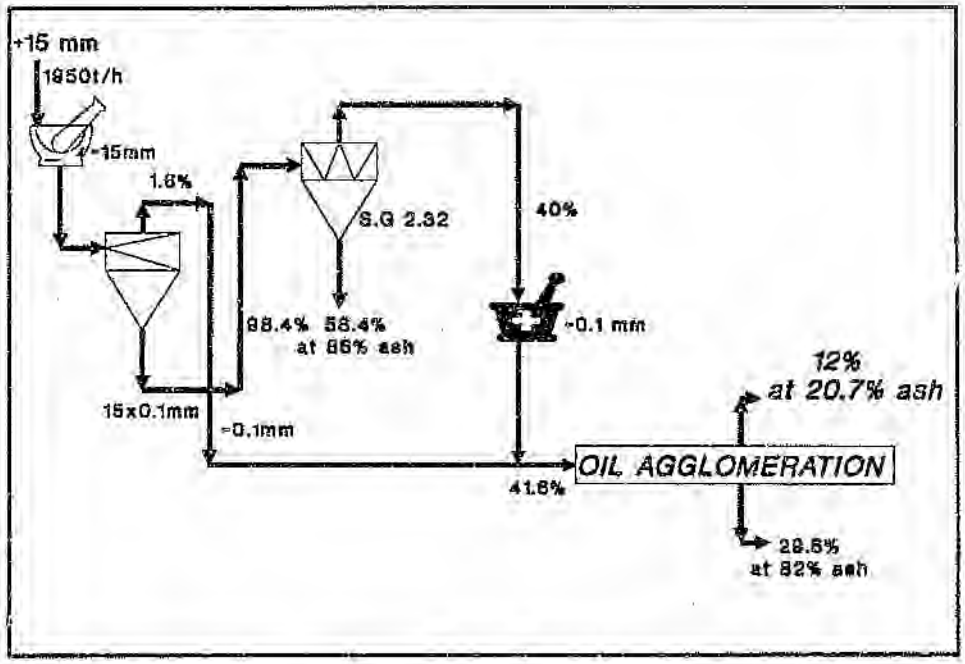


Figure 4.18 Production of middlings from discard (15 mm top size)

The product yields obtained on different top sizes of discard at the same ash content are shown in this pie charts given in **figure 4.21.** They show that it is possible to get 21% coal from 15 mm top size, 20% from 1 mm top size at an ash content of 21% by using oil agglomeration on -0.1 mm size fraction of top sizes. According to the second chart, froth flotation yields 11% coal from 15 mm top size, 12% from 5 mm top size, 20% from 1 mm top size at 33% ash content.

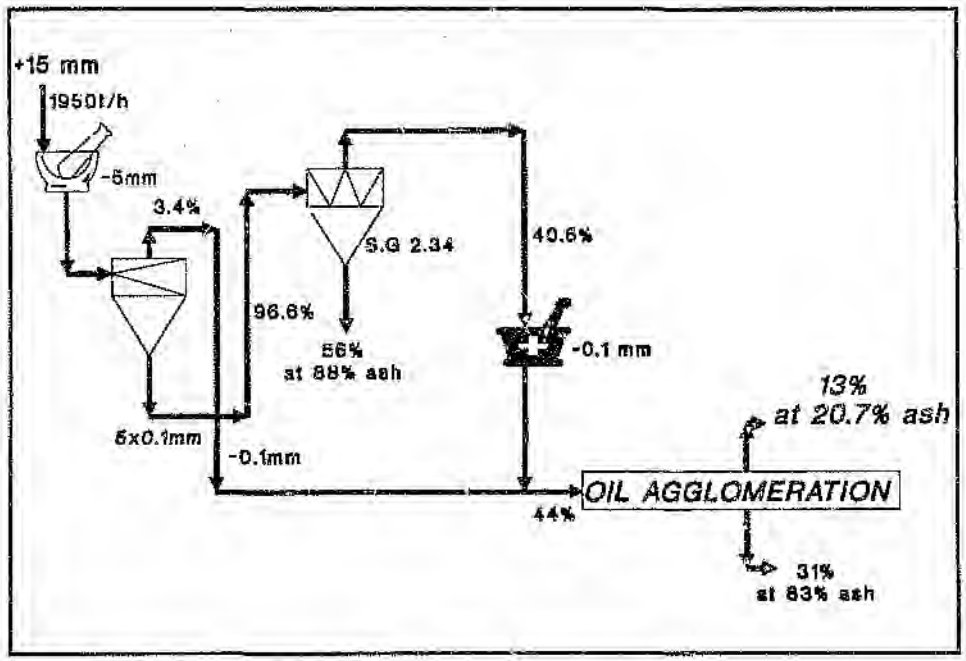


Figure 4.19 Production of middlings from discard (5 mm top size)

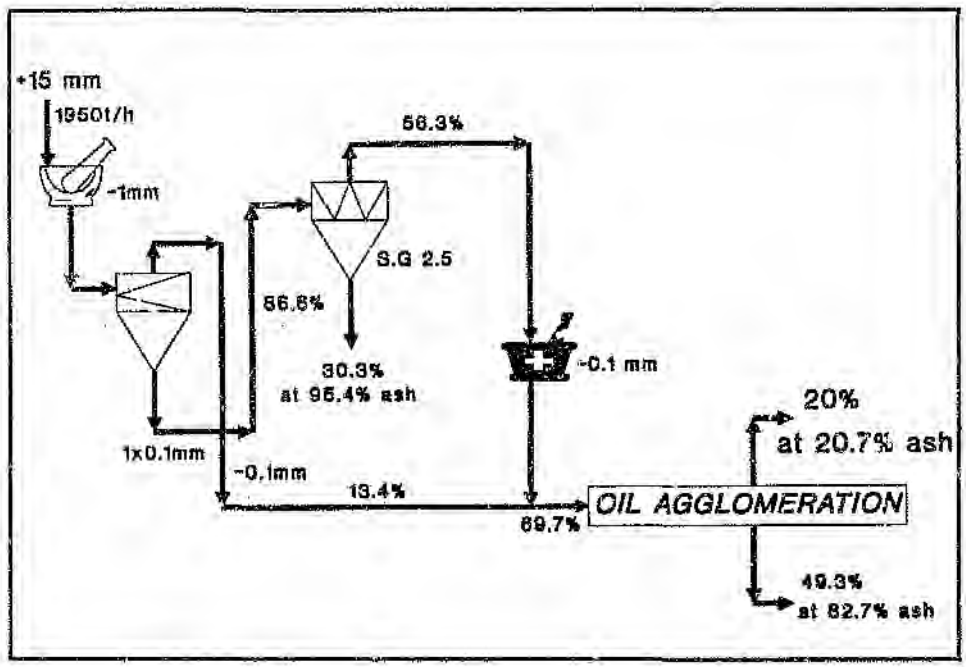


Figure 4.20 Production of middlings from discard (1 mm top size)

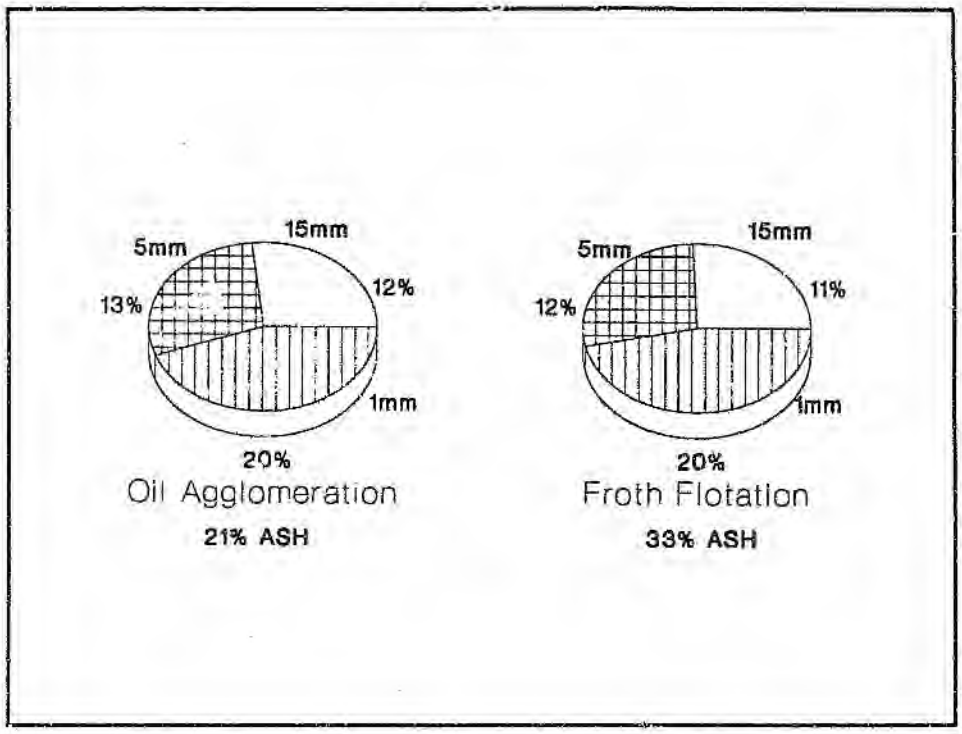


Figure 4.21 The results of retreatment of discard; top size; method of fine cleaning; overall yield percent and overall ash percent

Consequently, these results indicate that selective separation on -0.1 mm of middlings and discard is possible with oil agglomeration.

The same method is used to calculate the cost of producing middlings from discard. The capital and operating costs and the price of middlings from discard are given in **table 4.20** and **table 4.21.**

Table 4.12 The capital and operating cost to produce middlings from discard

	BENEFICIATION TECHNIQUES			
	DMC & FROTH FLOTATION		DMC & OIL AGGLOMERATION	
TOP SIZES(mm)	Operating Cost Rm	Capital Cost Rm	Operating Cost Rm	Capital Cost Rm
15	101.88	362.25	117.16	331.10
5	113.75	374.50	121.17	340.83
1	138.60	540.80	164.40	468.40

Table 4.13 The price of middlings produced from discard

	TOP SIZES(mm)					
	15		5		1	
Period to pay back (year)	Price R/t *1	Price R/t *2	Price R/t *1	Price R/t *2	Price R/t *1	Price R/t *2
2	329.5	282.7	306.4	273.9	256.5	243.322
2.5	303.9	266.0	280.9	253.3	233.5	4.9
3	270.7	253.7	265.9	241.6	219.8	214.4
3.5	261.0	246.8	257.3	235.9	211.5	208.5
4	255.5	243.3	252.4	231.9	206.6	205.5
4.5	252.7	242.3	250.2	230.8	204.0	204.6
5	251.9	242.9	249.9	231.5	203.1	205.1

*1 Price of Coal produced by froth flotation

*2 Price of Coal produced by oil agglomeration

4.4 Moisture Levels in The Products

The proposed equipment employed in the flowsheets adopted are listed Appendix I. As shown in the appendices a basket centrifuge is used to dewater small sizes beneficiated by dense medium cyclone and a horizontal belt filter for -0.1 mm ultra fines dewatering. According to reference 9, while the surface moisture of the product from the basket centrifuge is expected to be 5%-10%, the surface moisture of a belt filter product ranges from 8% to 35%. The surface moisture of the product is largely dependent on the particle size distribution and tonnage throughput. Consequently the figures given in ref.9 have been modified to the top sizes used in this project. The modified figures are given in **table 4.14.**

Table 4.14 Modification of moisture/size data given in ref.9 to the top sizes used in this project

Dewatering Equipment	Particle Size Range (mm)	Modified Particle Size Range (mm)	Surface Moisture Content (%)	Modified Surface Moisture Content(%)
Basket Cent.	20-0.5	15-5	5-10	6
		5-1		9
		1-0.5		10
Belt Filter	0.5-0.05	-0.1*	8-35	14***
		-0.1**		32

* -0.1 mm treated by oil agglomeration

** -0.1 mm treated by froth flotation

*** After suitable natural drainage the agglomerated product can reach a moisture of 16% without centrifuging (see reference 9).

The data given in table 4.14 are used to calculate the surface moisture content of products produced of different top sizes. The moisture content of products from

either oil agglomeration or froth flotation are discussed in section 5.

5 CONCLUSION

In this project the aim was to identify the liberation characteristics of Waterberg coal. The literature survey showed that Waterberg Coal was suitable for density based processes as well as surface property dependent methods. Thus, Waterberg coal is amenable to treatment by both froth flotation and oil agglomeration.

As previously discussed, the samples used in this project were oxidized. Despite oxidation, the results showed that liberation followed by froth flotation and oil agglomeration increased the yield of lower ash products from the middlings currently produced, and could also yield additional saleable products from the currently discarded fraction. If it were possible to obtain a less oxidized sample, even better beneficiation results could be expected after liberation.

The identification of the liberation characteristics of the Waterberg coal shows that a beneficiated fraction of the discard could be of power station coal quality. This would reduce the percentage of discard, but the cost of such is probably unacceptably high. Coal with a low ash content and reasonable yield, could also be produced by liberation and separated from middlings and used as a feedstock for direct liquefaction or carbonization. Such a product could be obtained at cost probably comparable with current production costs in the Waterberg.

The liberation studies indicated that lower ash products are liberated on crushing middlings to -1 mm. If crushed to -75 µm, it might be possible to obtain coal with an ash content of 5%, at approximately 30% yield. This might find application as a direct liquefaction feedstock, or coal/water, coal/oil mixtures.

Liberation studies were similarly carried out on the discard.The results of the liberation studies were submitted to computerised washing simulation. Three flowsheets, which included comminution, oil agglomeration and froth flotation, were designed for each top size of middlings and discard.

All results on -0.1 mm size fraction of top sizes showed the superiority of oil agglomeration.

The overall results obtained from the flowsheets may be summarised as follows; the overall potential increase in yield are given in **figure 5.1** and **5.2.**

In figure 5.1, it is demonstrated that the yield of low ash coal is increased to 27% at 13% ash content, plus 12.7% middlings at 33% ash produced from discard using froth flotation. In figure 5.2, the yield of coking coal is increased from 12% to 24%, the ash being reduced to 9.6%; plus 13% power station coal of 21% ash being produced from discard by oil agglomeration.

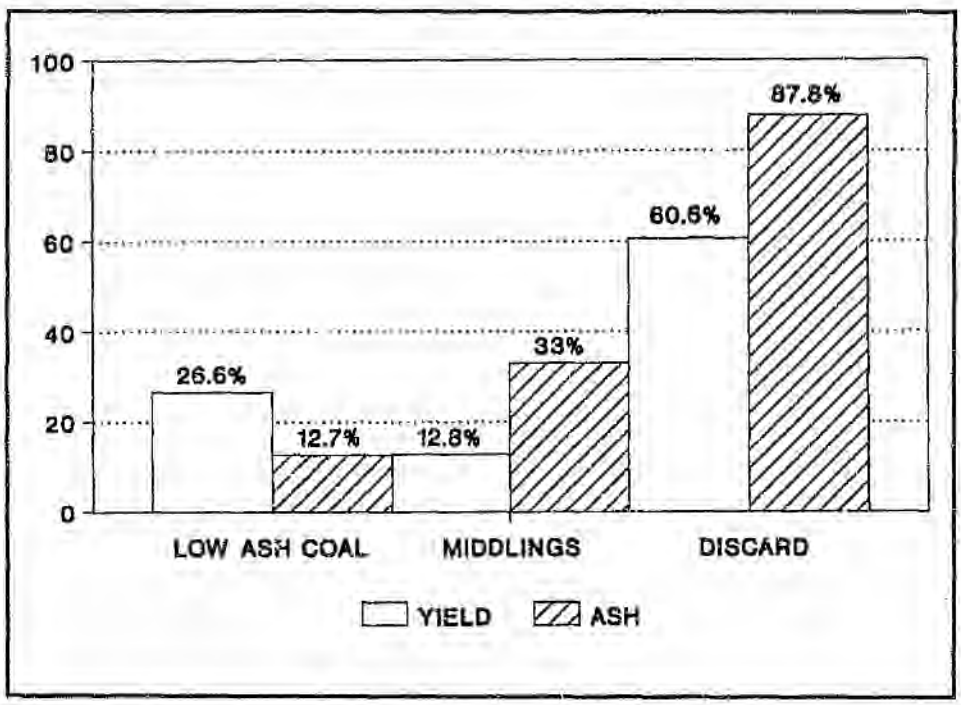


Figure 5.1 Overall improvement in yield using comminution,+dmc,+froth flotation

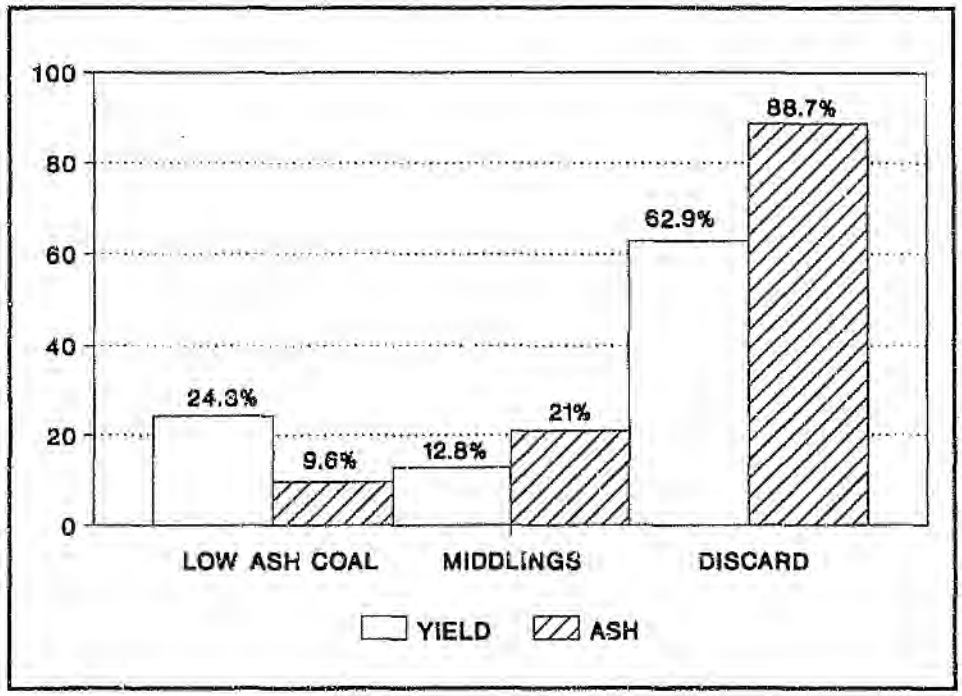


Figure 5.2 Overall improvement in yield using comminution,+dmc,+oil agglomeration

The data given section 4.4 are used to calculate the surface moisture content of products produced of different top sizes. The surface moisture content of products from either oil agglomeration or froth flotation are given in **table 5.1 and 5.2.**

Table 5.1 Surface moisture content of products produced using DMC and oil agglomeration

Top Size (mm)	DMC and Oil Agglomeration Combination to Produce LAC from Middlings			DMC and Oil Agglomeration Combination to Produce Middlings From Discard		
	Unit	Moisture (%)	Cum. Moisture (%)	Unit	Moisture (%)	Cum. Moisture (%)
15	Basket Cent.	6	10	Belt Filter	14	14
	Belt Filter.	14				
5	Basket Cent.	9	13	Belt Filter	14	14
	Belt Filter	14				
1	Basket Cent.	10	13	Belt Filter	14	14
	Belt Filter	14				

The current moisture levels in products from the Grootegeluk preparation plant are estimated to be as follows;

Product	Moisture Content (%)
Coking coal	9.5-10.5
Middlings	11-11.5

Table 5.2 Surface moisture content of product produced using DMC and froth Flotation

Top Size (mm)	DMC and Froth Flotation Combination to Produce LAC from Middlings			DMC and Froth Flotation Combination to Produce Middlings From Discard		
	Unit	Moisture (%)	Cum. Moisture (%)	Unit	Moisture (%)	Cum. Moisture (%)
15	Basket Cent.	6	22	Belt Filter	32	32
	Belt Filter.	14				
5	Basket Cent.	9	28	Belt Filter	32	32
	Belt Filter	14				
1	Basket Cent.	10	26	Belt Filter	32	32
	Belt Filter	14				

Using available factorised data, the cost of each flowsheet was estimated. The results of the cost analysis is summarised in **table 5.3** and **5.4**.

Table 5.3 Production costs of enhanced yields from middlings

SIZE mm	TYPE of PROCESS	TONS of COAL PRODUCED mt/annum	ASH %	COST R/ton
-15	(1)	2.4	17	49
	(2)	1.94	13	64
-5	(1)	2.1	15.7	63
	(2)	2.2	11.5	64
-1	(1)	2.2	15.3	51
	(2)	1.8	9.3	70

(1) D.M.C + Comminution + Froth Flotation
(2) D.M.C + Comminution + Oil Agglomeration

Table 5.4 Production costs of enhanced yields from discard

SIZE mm	TYPE of PROCESS	TONS of COAL PRODUCED mt/annum	ASH %	COST R/ton
-15	(1)	1.1	32.5	251.9
	(2)	1.2	20.7	242.9
-5	(1)	1.2	32.5	249.9
	(2)	1.3	20.7	231.5
-1	(1)	2.2	15.3	203.1
	(2)	1.8	9.3	205.1

(1) D.M.C + Comminution + Froth Flotation
(2) D.M.C + Comminution + Oil Agglomeration

The studies showed that the run-of-mine yield of 10% ash coal may be increased from present level of 12% to 24%; and that 20% of 21% ash power station coal may be produced from the present discard fraction. The production cost of these enhanced yields are of the order of R70/ton and R205/ton respectively.

While the experimental results on the finest sizes (-75 μm or -0,1 mm) indicate that oil agglomeration is the best treatment for Waterberg fine coal, the budget-level economic evaluations indicate production costs for such enhanced yields are too high for normal commercial application.

However, if direct liquefaction of Waterberg coal is ever considered, the data produced in this report shows that enhanced yields may be obtained by processes which could themselves be incorporated in the direct liquefaction process. In this way the prime elements of the beneficiation cost (comminution to <100 micrometres and oil agglomeration) could be included in the overall conversion process. This may turn out to be quite effective economically, but detailed studies would be required to prove this.

Origin of Coal

Coal may be defined as being fundamentally composed of fossilised plant debris, interspersed with a smaller amount of inorganic matter which has undergone progressive physical and chemical alteration during genesis.

In the first stage of coalification, organic plant remains have undergone various degrees of decomposition, and are transformed into peat. The peat beds after burial become metamorphosed under the influence of pressure, temperature and time, and transformed through the progressive stages of brown coal(lignite), sub-bituminous and bituminous coal to anthracite, and meta-anthracite(**figure A.1**).

The degree of coalification is termed as rank. There are many factors which affect the composition of coal and its rank. These factor include (Francis, 1954):
- the mode of accumulation and burial of the plant debris forming the deposits,
- the structure of the coal forming plants,
- the age of the deposits and their geographical distribution,
- the chemical composition of the coal forming debris and its resistance to decay,
- the nature and intensity of the plant decaying agencies,
- the geological history of the residual products of decay of the plant debris forming the deposits.

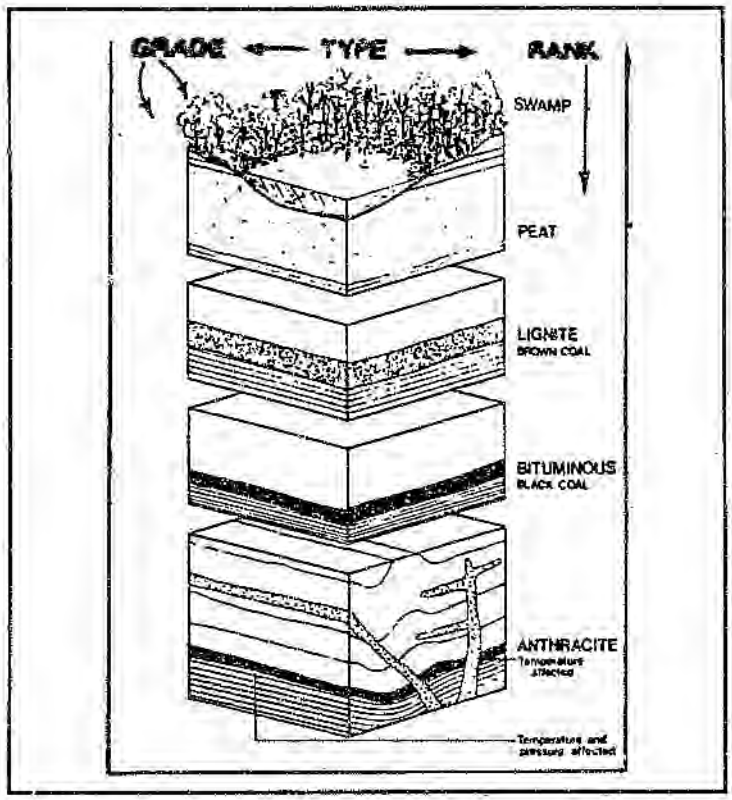


Figure A.1 The formation of coal

Composition of Coal

Constitution of coal may be described in two ways microscopically and macroscopically.

Microscopic constituents of coal

Microscopically, organic constituents of coal are termed macerals. They are derived from maceration of plant matter during the early stage of peat accumulation and early coalification.

Macerals

Macerals are classified into three major groups and many individual macerals(**table A.1**) on the basis of source material, morphologic and genetic considerations (Meyers, 1982)

Vitrinite Maceral Group

Vitrinite is the most abundant and most important maceral in coals, formed from woody and cortical tissues. The density of vitrinite varies between 1,3g/ml in bituminous coal and 1,8g/ml in anthracite with increasing rank. It is largely responsible for the coking properties of coal, and also the most suitable feedstock for conversion process of pyrolysis. In low rank coals, vitrinite is readily oxidised, frequently with a high liability towards spontaneous combustion, and also can be hydrogenated and liquefied.

Table A.1 Classification of macerals

Group	Maceral
Vitrinite	Telinite, collinite, "pseudovitrinite", vitrodetrinite
Liptinite (Exinite)	Sporinite, cutinite, resinite, alginite, suberinite, liptodetrinite
Inertinite	Micrinite, macrinite, semifusinite, fusinite, sclerotinite, inertodetrinite

Exinite Maceral Group

The macerals of the exinite group are derived from resinous and waxy material of

plants, such as resins,cuticles, spore and pollen exines, and oil of vegetable origin. Exinite group macerals range in density between 1,18 and 1,25g/ml with increasing rank. Exinite by itself is non-coking, but it helps the coking properties of vitrinite, and combusts easily and rapidly, thereafter leaving very little carbon residue to maintain the flame. The rate of potential oxidation is not well known in exinite, although it has been suggested that exinite may be more rapidly oxidized than vitrinite. Particularly in low rank coal exinite is suitable for hydrogenation and the pyrolysis process.

Inertinite Maceral Group

Inertinite represents a group of macerals derived from plant material other than woody tissues in the peat stage of coal formation. The inertinite group macerals range in density between 1,35 and 1,7g/ml, changing very little, if at all with increasing rank. In direct contrast to vitrinite and exinite, the inertinite macerals are non-combustible and non oxidable. Furthermore it is not suitable for hydrogenation, pyrolysis and coking processes.

Mineral Matter

The constituents of coal minerals can be classified into six groups:

- Clays
- Carbonates
- Sulphates
- Oxides
- Chlorides
- Sulphates

Microlithotypes

The combination in various associations of the organic and inorganic constituents of coal are termed microlithotypes(**figure A.2**). They have two properties in order to distinguish them from macerals.

- They are greater than 50μm in width.

– They may be composed of pure macerals or varying proportions of different macerals.

Physically, they have hardness and resistance to breakage.

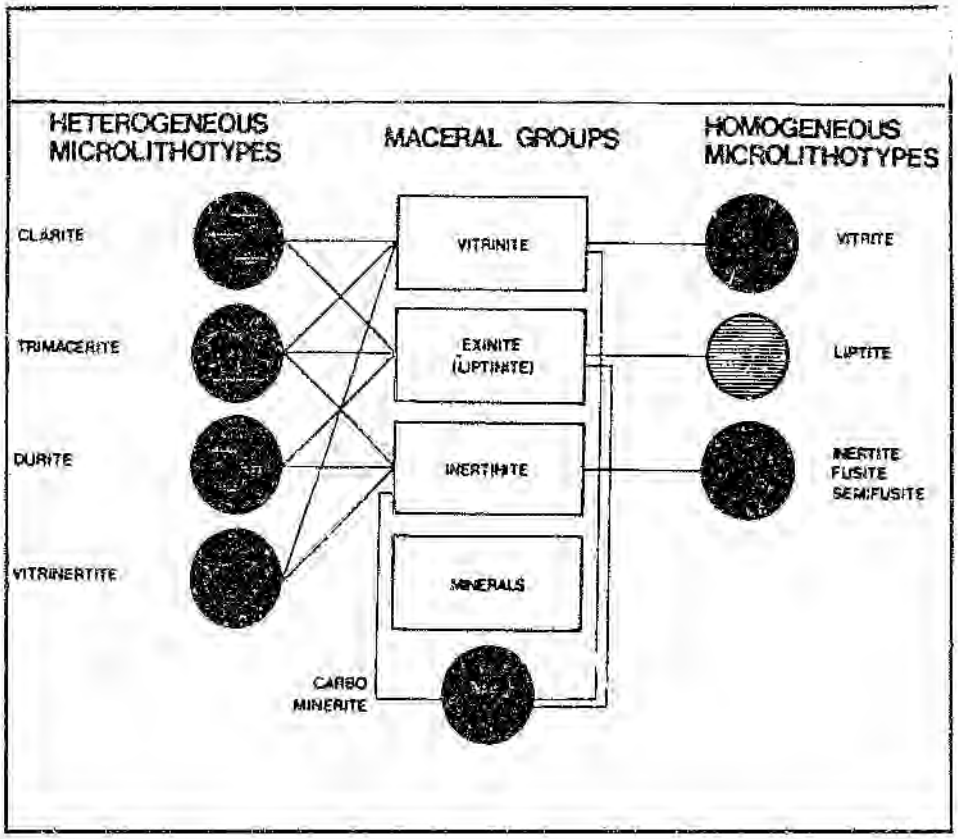


Figure A.2 The association of macerals into microlithotypes

– Macroscopic constituents of coal

Macroscopically identifiable constituents of coal are called lithotypes are formed by a succession of microlithotypes. Lithotypes may be classified into sapropelic and humic coals. Sapropelic coals are formed from the accumulation of algae and fine wind or water–borne detrital organic matter. The accumulation of abundant humic matter with in the original peat swamp forms humic coals, as opposed to

the open-water origins of the sapropelic coals (Falcon, 1986).

Characterizisation of Coal

The general analyses performed on coal are:

- Proximate analysis, which entails
  ash content
  moisture content
  volatile matter content
- Calorific value
- Swelling number
- Sulphur content
- Ultimate analysis
- Hardgrove index
- Abrasiveness
- Ash fusion temperature

Proximate analysis is used to determine moisture, ash,volatile matter, fixed carbon contents, while ultimate analysis is used to determine the elements present in the organic matter which are carbon, hydrogen, nitrogen, sulphur and oxygen.

Proximate Analysis

Moisture Content (SABS Methods 924, 925)

The inherent moisture retained in the pores of the coal is known as the moisture content of coal. Before this test is carried out the sample should be exposed for at least 30 min to allow the moisture content to attain equilibrium with the laboratory atmosphere. About one gram of coal passed a 212 μm test sieve is weighed into a dish and heated in a vacuum-oven at a temperature of 105–110°C until it's mass remains constant(1 hour is normally sufficient).

After 1 hour, the following steps are carried out;

- take the dish out of the oven and cover it with its lid,
- allow it to cool on a metal plate for 10min and transfer it to the desiccator,
- allow it to cool in the desiccator, and then weigh to determine the loss in mass.

The percentage loss in mass is expressed as the moisture content.

Ash Content (SABS Method 926)

The mass of the residue remaining after combustion of the mineral matter present in the coal is known as ash. About 1–2 grams sample is weighed in a dish. This is placed in a cold muffle furnace ventilated by air, heated up to a temperature of 815 ±10°C and maintained at this temperature until constant mass is attained. The dish is taken out and cooled, first in open air and then in a desiccator, and the mass of residue in the dish is determined.

Volatile Matter (SABS Method 927)

About 1 gram of sample is weighed into a crucible with a well fitting lid. It is heated for seven minutes at 900 ±10°C out of contact with air in the furnace. The crucible is then removed, cooled in a desiccator, and weighed again. The moisture content is determined at the same time. The volatile matter content of coal is the loss in weight, expressed as a percentage of the weight of the coal used, less the moisture content.

Fixed Carbon (SABS Method 928)

This property is calculated by subtracting the sum of moisture, ash and volatile matter contents from 100%.

Calorific Value (SABS Method 929)

This is the heating value of coal> About 1 gram of weighed coal sample is burned in a calorimetric bomb filled with oxygen under pressure. The calorimetric bomb is placed into the calorimeter vessel which holds a specified amount of water. The gross calorific value is calculated from the temperature rise of the water in the

calorimeter vessel. This is the way to determine the gross calorific value. The coal user is generally more interested in the net calorific value(see SABS Method 929 for the procedure).

Swelling Number (SABS Methods 933, 934)

This method is used to determine coking properties. To determine the swelling index 1 gram sample is weighed in a suitable crucible and heated with the crucible covered in a special electric furnace (i.e. out of contact with air) at a temperature of 820 ±5°C. After two and a half minutes the crucible is removed and the solid residue is compared with the outlines of a set of standard profiles. If the solid residue is a powder the swelling number is zero.The standard profiles are shown in **figure A.3**.

Sulphur (SABS Eschca Method 930)

Approximately 1 gram sample is weighed accurately and mixed intimately with 2.5 gram of the Eschca mixture (two parts magnesium oxide to one part sodium carbonate) in a suitable vessel. This mixture is then transferred into the crucible, which is placed in a cold furnace ventilated by air, heated to 800°C, and kept at this temperature until oxidation is complete. All combustible materials are removed and convert the sulphur present in the sample to sulphate by oxidation. The sulphate is then extracted and determined gravimetrically by precipitation with barium chloride.

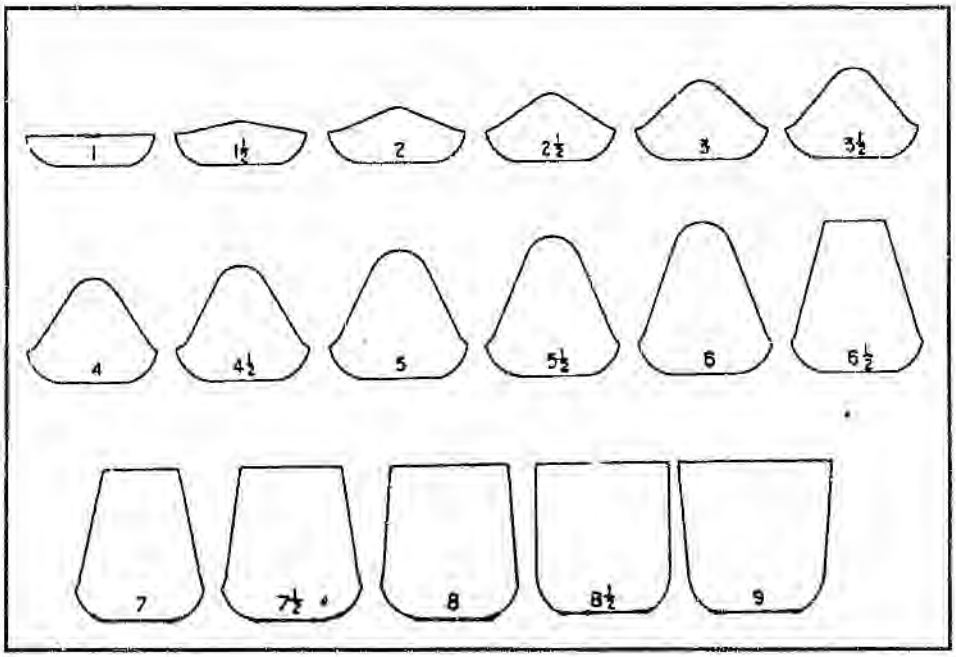


Figure A.3 Swelling number standard profiles

Ultimate Analysis (ASTM Method, D 3176–84)

This is the analysis of the coal for elements present in the organic matter, but only the determination of carbon and hydrogen will be briefly described here.

The determination of carbon and hydrogen is made by burning the 0.2gr weighed sample of –250μm in oxygen in a combustion tube at evaluated temperatures, and absorbing the water and carbon dioxide formed in suitable reagents. The percentage of carbon and hydrogen are determined from the mass of the coal and these two products.

Hardgrove Index (ASTM Method, D 409–85)

This method is used to determine the ease of pulverisation of coals in comparison

with coal chosen as standard. About 50 ±0.01 gram sample is weighed from the 4.75 mm by 0.6 mm size deducted material and placed in a Hardgrove ring–ball machine and turned for 60 revolutions under standard load. The mass of minus 200 mesh and the plus 200 mesh are determined separately. The grindability is determined by subtracting the weight retained on the 200 mesh sieve from the test sample weight.

Abrasiveness Index (Draft BSS no 1016 part 19)

During the grinding operation, the coal which contains much sandstone or pyrite may cause considerable wear on the mill. The abrasiveness index measures this tendency. For each test 4 kg of –4.75 mm coal is used. The coal is placed into the pot, and the stirrer rotated for 12 000 revolutions at 1 500 rpm. The wearing plates are weighed before and after the test and the loss of mass is the abrasiveness index.

Ash Fusion Temperature (SABS Method 932)

A suitable amount of coal must first be ashed to give enough ash for the test. The ash is finely ground and pasted with water. The paste is shaped into a pyramid, cylinder or cube and dried. This test piece is heated at about 800°C under standard conditions, kept under continuous observation, and the temperatures at which characteristic changes of shape occur are recorded. The following temperatures are recorded;

– Deformation Temperature (DT): Where the first signs of rounding of the tip or edges of the test piece occur.

– Hemisphere Temperature (HT): where the ash has formed approximately a hemisphere.

– Flow Temperature (FT): where the ash has been spread out over the supporting tile in a layer, so that the height is a third of the hemisphere height.

Appendix B Coal in South Africa

Characteristics

South Africa's major coal deposits occur in the Vryheid Formation of the Karoo Sequence. They have varied patterns and different qualities, because of the geological history of coal development in South Africa. It is necessary to characterize South African coals in order to ensure optimum evolution, marketing and selective utilisation more efficiently.

There are many systems which are represented by using proximate and ultimate analyses, and they are used to characterize and classify coal deposits in northern hemisphere. The chemical analyses therefore formed the basis of a simple classification of coal. In the past decade many coals, and in particular southern hemisphere coals, were found to behave anomalously in such systems.

In South Africa the coals are much more variable in type. They have some properties which do not allow us to accept these systems for northern hemisphere coal as they stand,

- South African coals are inertinite-rich,
- There is a large proportion of organic matter, transitional in origin and form, between vitrinite and inertinite. Vitrinite therefore should not be used alone to define the type boundaries.
- The liptic category has been amended in order to incorporate coal of transitional category be termed humo-sapopelic, and that it should include subcategories of exinite-rich, clarites, trimacerites and durites(Falcon, 1986).

The differences between northern hemisphere and southern hemisphere, as described above, requires the adaptation of these two universal classifications to South African coal which is illustrated by **figure B.1**.

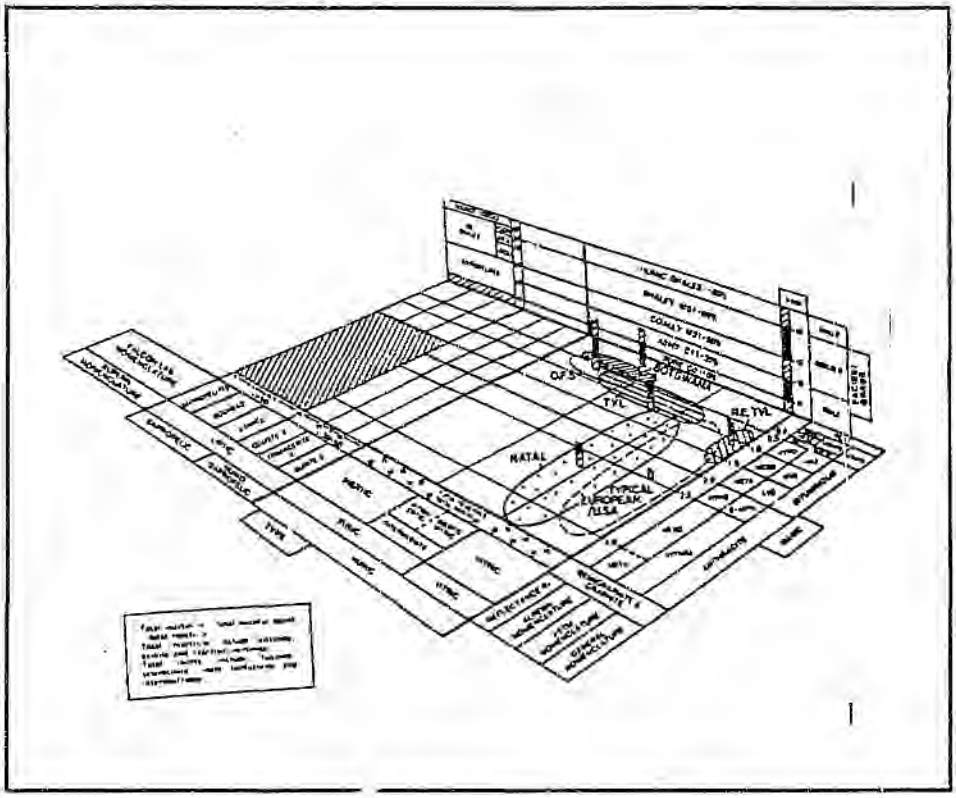


Figure B.1 The adaptation from northern hemisphere coal to South African coal based upon the "universal classification"

Reserves

South Africa has 18 principal coalfields. They are spread over an area of 700 km from north to south and 500 km from west to east (Mehliss, 1988).
The location of each coal field is shown in **figure B.2.**

The two latest reserve figures of South African coal are reported by Petrick(1975) and De Jager(1982). De Jager decided to use a different basis from that used by Petrick. This therefore leads to differences between the two figures. Recoverable coal per coal field according to Petrick and De Jager is given by **table B.1.**

Figure B.2 Coalfields of South Africa

The results of the Petrick commission basically indicated that South Africa should use its coal reserves carefully. This makes the identification of liberation characteristics on South African coal more important so that the optimum way of using these reserves can be determined.

Table B.1 Recoverable Coal Per Coalfield According to Petrick(1975) and De Jager(1982) (In Megatons)

	RAW BITUMINOUS		SALEABLE HIGH-GRADE WASHED BITUMINOUS		METALLURGICAL WASHED	
	PETRICK	DE JAGER	PETRICK	DE JAGER	PETRICK	DE JAGER
WITBANK	8 464	12 461	3 800	6 456	169	400
HIGHVELD	9 913	10 979	4 443	1 618	-	230
SOUTH RAND	750	730	53	-	-	-
SPRINGBOK FLATS	-	1 700	-	1 700	-	150
WATERBERG UPPER	2 389	18 000	727	4 983	319	SUBSTANTIAL
LOWER	237	202	-	107	-	107
LIMPOPO	160	725	-	267	-	267
SOUTPANSBERG	-	-	-	-	-	-
WESTERN AREAS	167	510	70	318	39	102
UTRECHT	491	472	162	314	88	168
KLIP RIVER	93	71	83	55	54	39
VRYHEID	1 460	538	1 222	2 460	37	16
EASTERN TRANSVAAL	-	-	-	-	-	-
KWAZULU	-	-	-	-	-	-
KANGWANE	1 029	2 233	8	-	-	-
SASOLBURG-VAREENIGING	239	4 920	-	-	-	-
OFS(REMAINDER)	-	-	-	-	-	-
OLD SPRINGFIELD	25 369	57 541	10 566	18 060	706	1 469

Products

About 48.7 percent of South Africa's coal is mined underground by board and pillar methods(almost entirely mechanised), some 7,8 percent each by longwalling, and pillar recovery(stooping), and 35,7 percent by open–pit methods.

Based on 1990 statistics, South Africa's coal producers may be grouped into five classes, as shown on **figure B.3**.

The production from the various coalfields, the processing, products and destination of sales are summarised as a flowsheet on **figure B.4**.

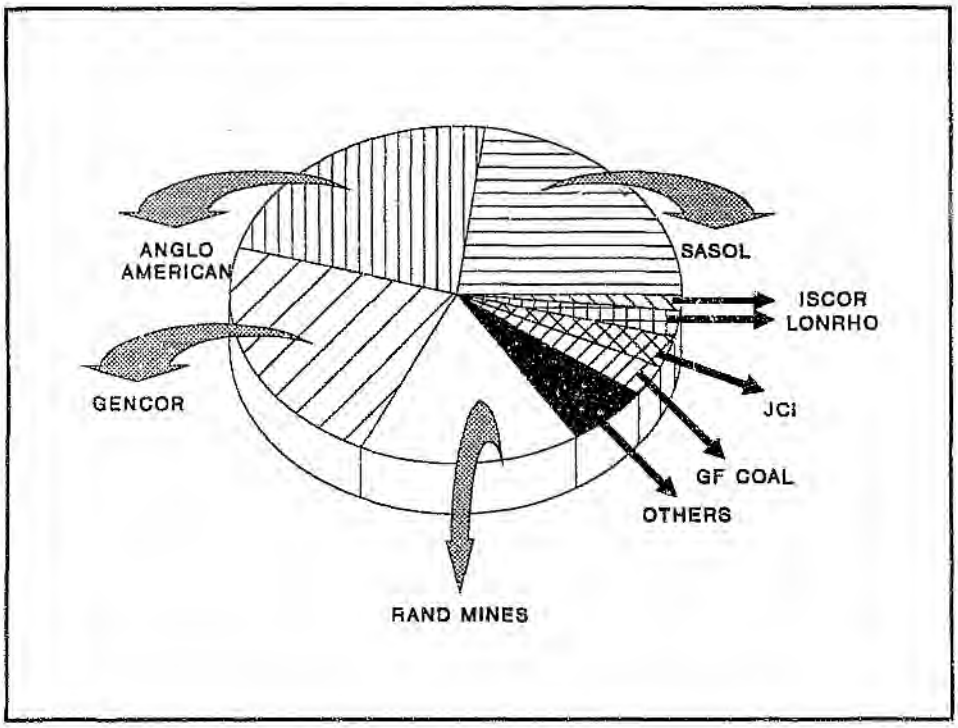


Figure B.3 Saleable coal production by controlling groups

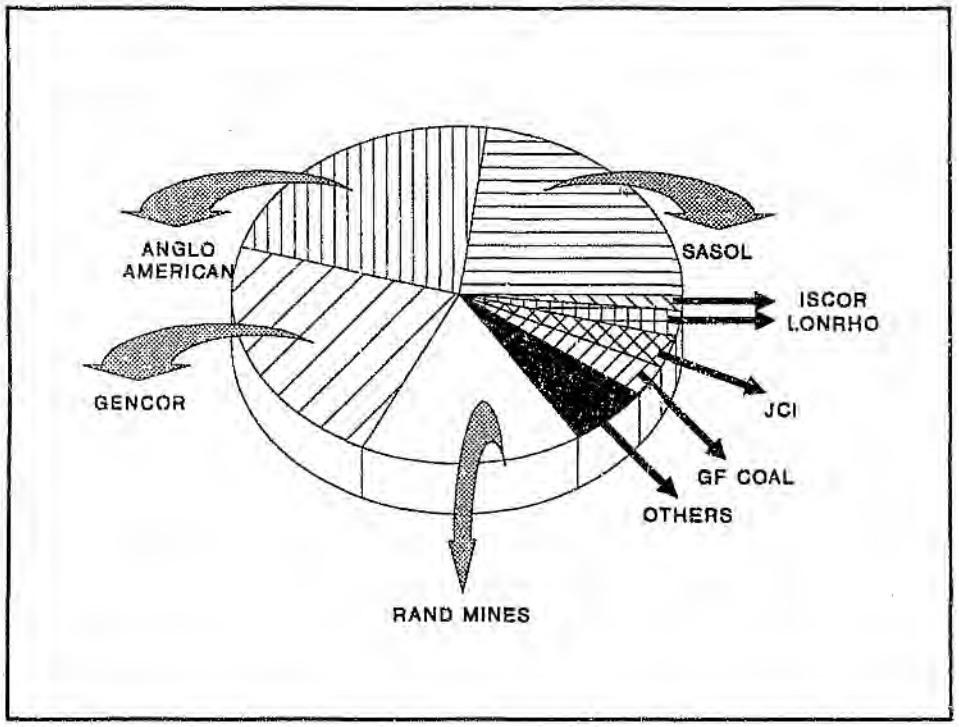


Figure B.3 Saleable coal production by controlling groups

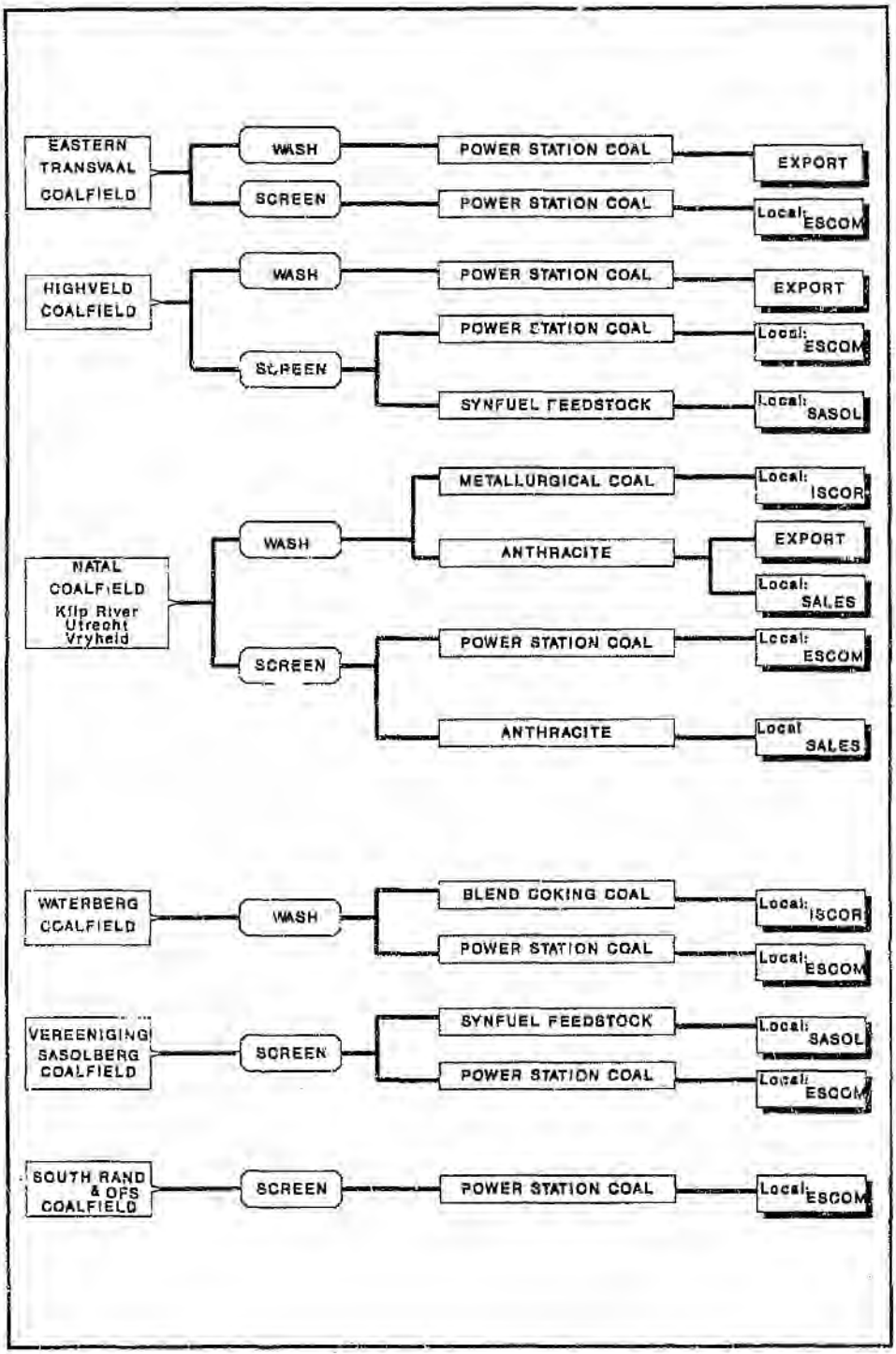
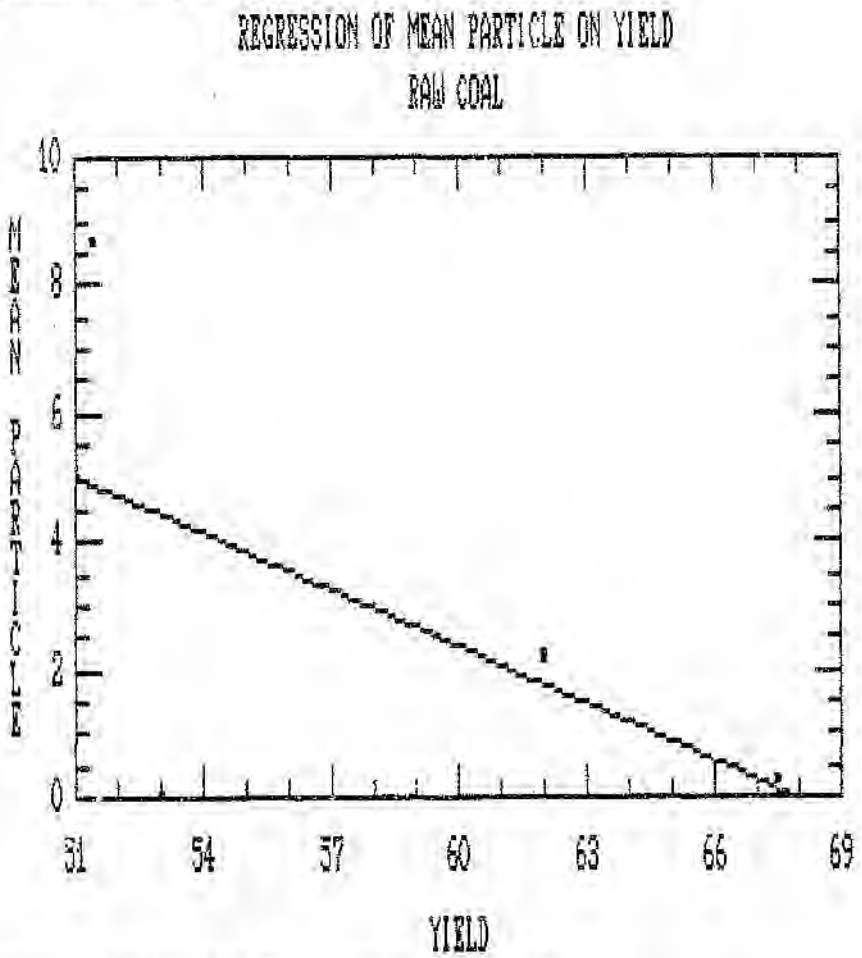


Figure B.4 Republic of South African Coal Industry (1990)

Appendix C Simple Regression Analysis



Regression Analysis - Linear model: Y = a+bX

Dependent variable: A:MEANRA.VAR0 Independent variable: A:MEANRA.VAR1

Parameter	Estimate	Standard Error	T Value	Prob. Level
Intercept	20.0831	18.3473	1.09461	0.387919
Slope	-0.295351	0.311787	-0.947284	0.443481

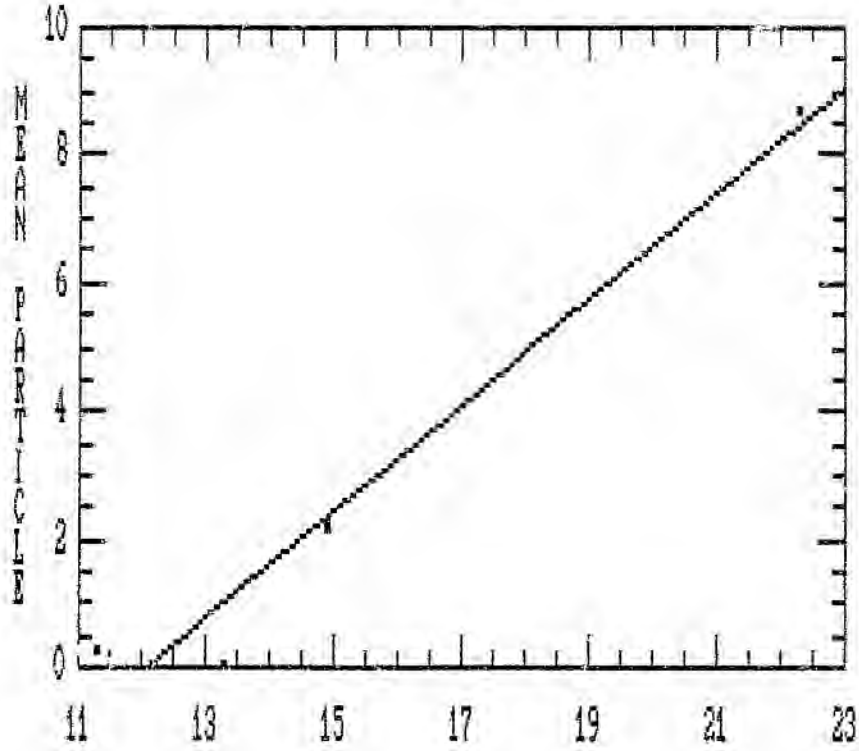
Analysis of Variance

Source	Sum of Squares	Df	Mean Square	F-Ratio	Prob. Level
Model	15.170334	1	15.170334	.897346	.44348
Error	33.811541	2	16.905771		
Total (Corr.)	48.981875	3			

Correlation Coefficient = -0.556519 R-squared = 30.97 percent
Std. Error of Est. = 4.11166

REGRESSION OF MEAN PARTICLE ON ASH

RAW COAL



Regression Analysis - Linear model: Y = a+bX

Dependent variable: A:MEANRA.VAR0 Independent variable: A:MEANRA.VAR2

Parameter	Estimate	Standard Error	T Value	Prob. Level
Intercept	-9.94488	1.87089	-5.31558	0.033617
Slope	0.82572	0.116938	7.0612	0.019472

Analysis of Variance

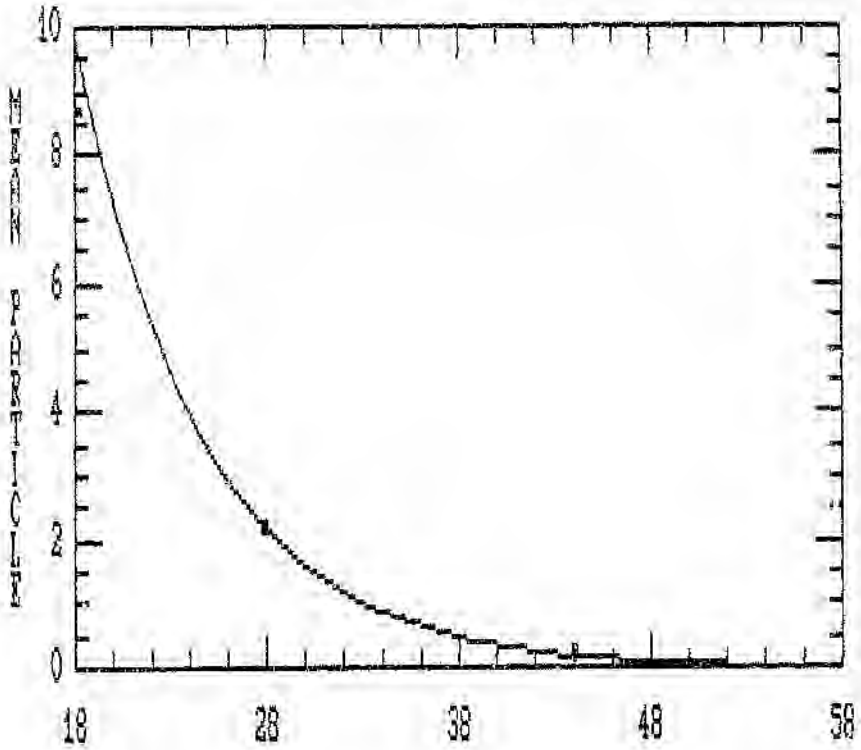
Source	Sum of Squares	Df	Mean Square	F-Ratio	Prob. Level
Model	47.092892	1	47.092892	49.8580	.01947
Error	1.8889829	2	.9444915		
Total (Corr.)	48.981875	3			

Correlation Coefficient = 0.980528 R-squared = 96.14 percent

Stnd. Error of Est. = 0.97185

REGRESSION OF MEAN PARTICLE ON YIELD

TESKA PRODUCT



Regression Analysis - Exponential model: Y = exp(a+bX)

Dependent variable: A:MEANPA.VAR0 Independent variable: A:MEANPA.VAR1

Parameter	Estimate	Standard Error	T Value	Prob. Level
Intercept	5.00494	0.591124	8.46681	0.0136643
Slope	-0.151156	0.01579	-9.57291	0.0107368

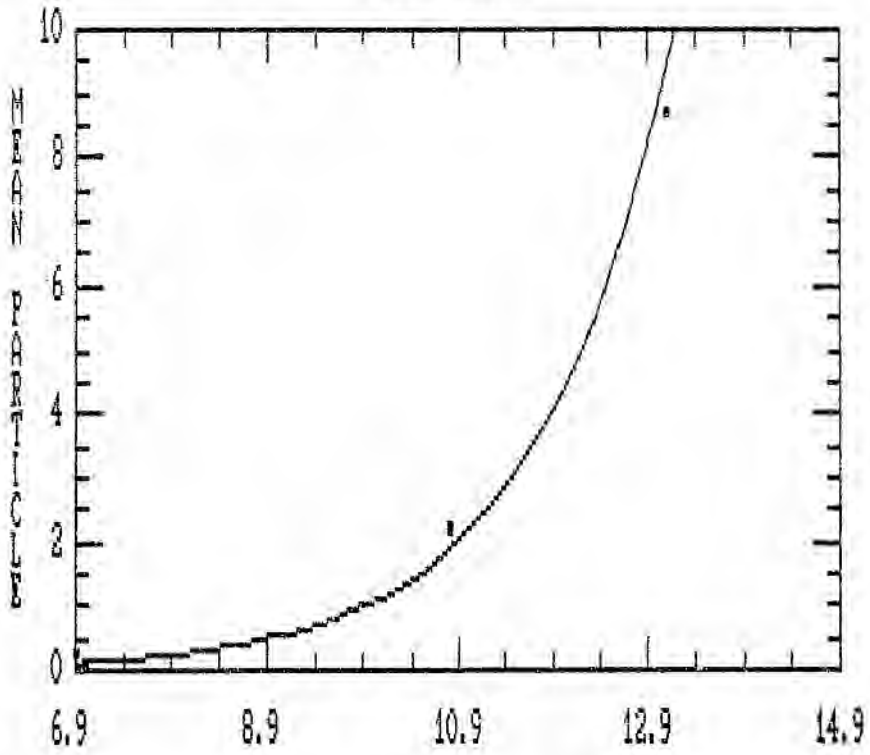
Analysis of Variance

Source	Sum of Squares	Df	Mean Square	F-Ratio	Prob. Level
Model	15.008709	1	15.008709	91.640681	.01074
Error	.3275556	2	.1637778		
Total (Corr.)	15.336265	3			

Correlation Coefficient = -0.989263 R-squared = 97.86 percent
Stnd. Error of Est. = 0.404695

REGRESSION OF MEAN PARTICLE ON ASH

TESKA PRODUCT



Regression Analysis - Exponential model: Y = exp(a+bX)

Dependent variable: A:MEANPA.VAR0 Independent variable: A:MEANPA.VAR2

Parameter	Estimate	Standard Error	T Value	Prob. Level
Intercept	-6.93218	1.75311	-3.95422	0.0584084
Slope	0.70(5	0.178726	3.91969	0.0593518

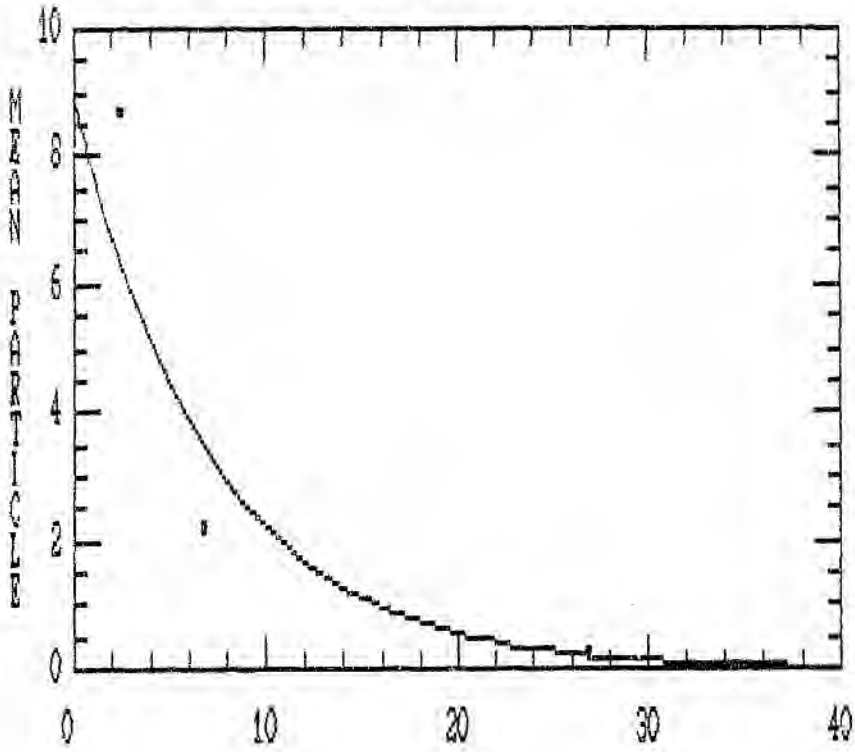
Analysis of Variance

Source	Sum of Squares	Df	Mean Square	F-Ratio	Prob. Level
Model	13.569818	1	13.569818	15.363970	.05935
Error	1.7664468	2	.8832234		
Total (Corr.)	15.336265	3			

Correlation Coefficient = 0.940648 R-squared = 88.48 percent
Stnd. Error of Est. = 0.9398

REGRESSION OF MEAN PARTICLE ON YIELD

MIDDLINGS



Regression Analysis - Exponential model: Y = exp(a+bX)

Dependent variable: A:MEANMA.VAR0 Independent variable: A:MEANMA.VAR1

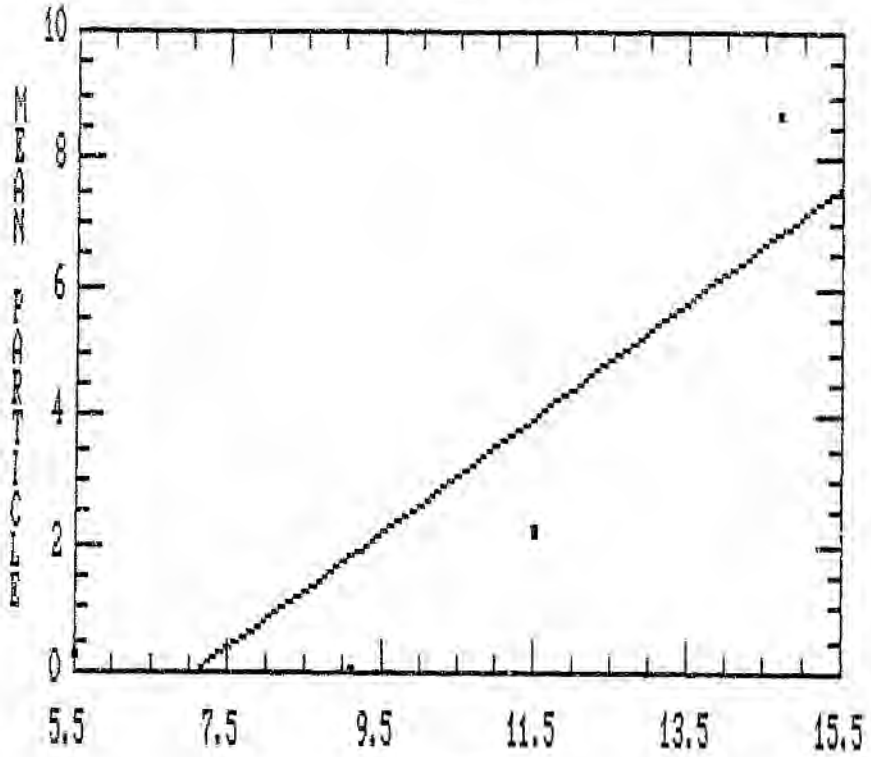
Parameter	Estimate	Standard Error	T Value	Prob. Level
Intercept	2.18285	0.381002	5.72923	0.0291403
Slope	-0.137646	0.016626	-8.27894	0.0142781

Analysis of Variance

Source	Sum of Squares	Df	Mean Square	F-Ratio	Prob. Level
Model	14.901445	1	14.901445	68.540782	.01428
Error	.4348198	2	.2174099		
Total (Corr.)	15.336265	3			

Correlation Coefficient = -0.985723 R-squared = 97.16 percent
Stnd. Error of Est. = 0.466272

REGRESSION OF MEAN PARTICLE ON ASH
MIDDLINGS



Regression Analysis - Linear model: Y = a+bX

Dependent variable: A:MEANMA.VAR0 Independent variable: A:MEANMA.VAR2

Parameter	Estimate	Standard Error	T Value	Prob. Level
Intercept	-6.32895	4.015	-1.57633	0.255655
Slope	0.89622	0.373832	2.39739	0.138692

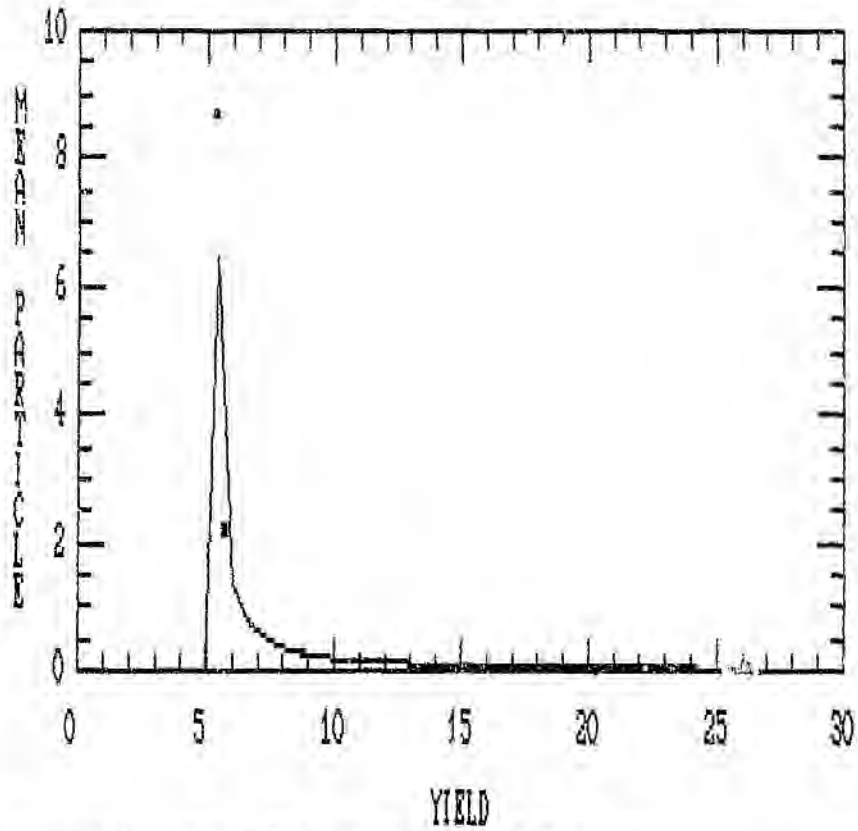
Analysis of Variance

Source	Sum of Squares	Df	Mean Square	F-Ratio	Prob. Level
Model	36.337246	1	36.337246	5.747460	.13869
Error	12.644629	2	6.322314		
Total (Corr.)	48.981875	3			

Correlation Coefficient = 0.861308 R-squared = 74.19 percent
Stnd. Error of Est. = 2.51442

REGRESSION OF MEAN PARTICLE ON YIELD

TESKA DISCARD



Regression Analysis - Reciprocal model: 1/Y = a+bX

Dependent variable: A:MEANDA.VAR0 Independent variable: A:MEANDA.VAR1

Parameter	Estimate	Standard Error	T Value	Prob. Level
Intercept	-5.01763	0.0582088	-86.2005	1.34553E-4
Slope	0.958025	4.06739E-3	235.538	1.80246E-5

Analysis of Variance

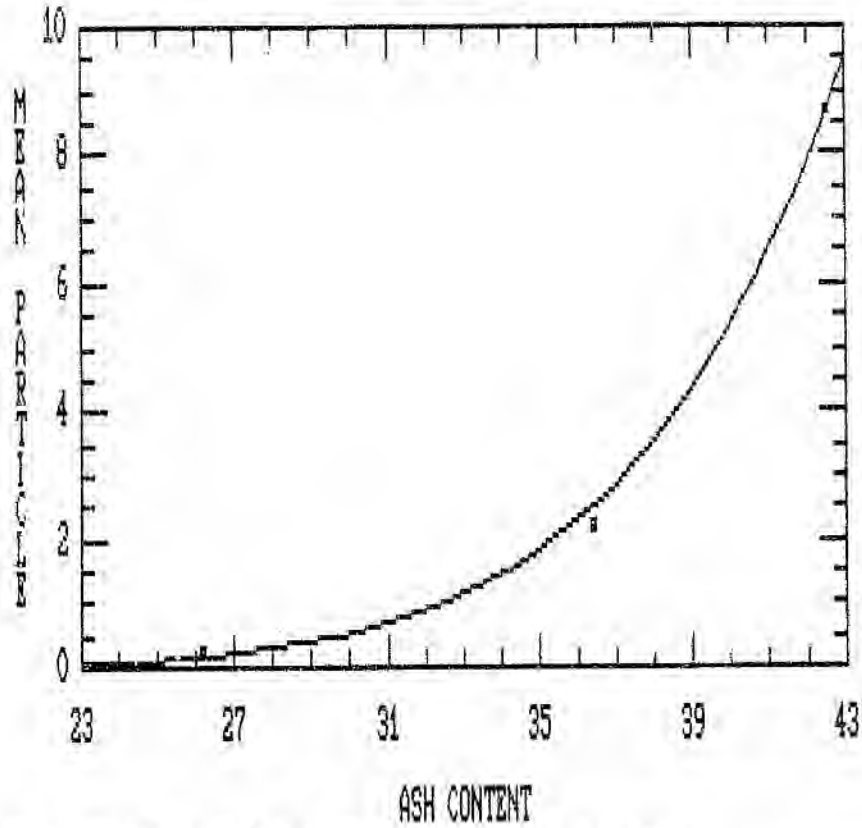
Source	Sum of Squares	Df	Mean Square	F-Ratio	Prob. Level
Model	268.485	1	268.485	55478.143	.00002
Error	.0096789	2	.0048395		
Total (Corr.)	268.49472	3			

Correlation Coefficient = 0.999982 R-squared = 100.00 percent

Stnd. Error of Est. = 0.0695663

REGRESSION OF MEAN PARTICLE ON ASH

TESKA DISCARD



Regression Analysis - Multiplicative model: Y = aX^b

Dependent variable: A:MEANDA.VAR0 Independent variable: A:MEANDA.VAR2

Parameter	Estimate	Standard Error	T Value	Prob. Level
Intercept*	-27.6047	3.0396	-9.08168	0.0119085
Slope	7.93739	0.881779	9.00157	0.0121175

* NOTE: The Intercept is equal to Log a.

Analysis of Variance

Source	Sum of Squares	Df	Mean Square	F-Ratio	Prob. Level
Model	14.966842	1	14.966842	81.028204	.01212
Error	.3694230	2	.1847115		
Total (Corr.)	15.336265	3			

Correlation Coefficient = 0.987882 R-squared = 97.59 percent

Stnd. Error of Est. = 0.429781

APPENDIX D The Results of Float and Sink Analysis on Raw Coal

Table D.1 Float and Sink Analysis on 15x5 mm.

Relative Density	Mass Distribution %	Ash (ad) %	Cum. Yield %	Cum. Ash (ad) %
F 1.30	1.1	2.5	1.1	2.5
1.30 – 1.40	18.9	11.7	20.0	11.2
1.40 – 1.50	17.5	22.0	37.5	16.2
1.50 – 1.60	8.8	31.9	46.3	19.2
1.60 – 1.70	5.1	40.5	51.4	22.3
1.70 – 1.80	4.0	50.3	55.4	24.3
1.80 – 1.90	4.5	59.5	59.9	26.9
1.90 – 2.00	3.9	67.9	63.8	29.5
2.00 – 2.20	7.9	79.3	71.7	34.6
S 2.20	28.4	83.6	100.0	48.7

Table D.2 Float and Sink Analysis on 5x1 mm.

Relative Density	Mass Distribution %	Ash (ad) %	Cum. Yield %	Cum. Ash (ad) %
F 1.30	7.3	1.8	7.3	1.8
1.30 – 1.40	29.9	7.9	37.2	6.7
1.40 – 1.50	13.6	21.2	50.8	10.6
1.50 – 1.60	7.0	31.2	57.8	13.1
1.60 – 1.70	4.2	40.7	62.0	14.9
1.70 – 1.80	3.2	48.3	65.2	16.6
1.80 – 1.90	3.1	55.8	68.3	18.4
1.90 – 2.00	3.0	64.7	71.3	20.3
2.00 – 2.20	6.8	75.0	78.1	25.1
S 2.20	22.0	83.9	100.0	38.0

Table D.3 Float and Sink Analysis on 1x0.1 mm.

Relative Density	Mass Distribution %	Ash (ad) %	Cum. Yield %	Cum. Ash (ad) %
F 1.30	7.9	2.1	7.9	2.1
1.30 – 1.40	40.3	6.1	48.2	5.4
1.40 – 1.50	10.3	19.3	58.5	7.9
1.50 – 1.60	5.3	29.5	63.8	9.7
1.60 – 1.70	3.7	38.5	67.5	11.3
1.70 – 1.80	2.6	46.9	70.1	12.6
1.80 –1.90	2.9	54.9	73.0	14.3
1.90 – 2.00	3.4	61.7	76.4	16.4
2.00 – 2.20	5.1	73.5	81.5	19.9
S 2.20	18.6	81.1	100.0	31.4

Table D.4 Float and Sink Analysis on -0.1 mm.

Relative Density	Cum. Yield %	Cum. Ash in floats %	Cum. Ash in sinks %
F 1.30	1.2	2.3	40.1
1.30 – 1.40	29.6	5.9	53.0
1.40 – 1.50	40.1	9.0	61.0
1.50 – 1.60	45.4	11.6	63.0
1.60 – 1.70	52.9	13.3	70.9
1.70 – 1.80	68.6	20.1	74.9
S 1.80	100.0	41.9	

Table D.5 Froth Flotation on −0.1 mm.

Product	Mass Distribution %	Ash (ad) %	Cum. Yield %	Cum. Ash (ad) %
1	22.0	11.2	22.0	11.2
2	23.3	28.3	45.3	20.0
3	1.3	29.1	46.6	20.3
4	8.1	32.6	54.7	22.1
5	5.1	34.5	59.8	23.2
6	9.2	46.8	69.0	26.3
7	8.1	64.4	77.1	30.3
8	22.9	74.3	100.0	40.4

Table D.6 Oil Agglomeration on −0.1 mm.

Oil I.D	Oil %	Time Minutes	Yield %	Ash in Agg. %	Ash in Tails %	Feed Ash %
	20	30'0	54.7			
S.SOL AB	15	30'0	51.9	9.1	70.9	41.2
	15	10'0	37.2	6.1	60.8	

APPENDIX E The Results of Float and Sink Analysis on Teska Product

Table E.1 Float and Sink Analysis on 15x5 mm.

Relative Density	Mass Distribution %	Ash (ad) %	Cum. Yield %	Cum. Ash (ad) %
F 1.30	-	-	-	-
1.30 – 1.40	18.2	13.1	18.2	13.1
1.40 – 1.50	30.0	23.0	48.2	19.3
1.50 – 1.60	19.8	32.8	68.0	23.2
1.60 – 1.70	14.3	41.9	82.3	26.5
1.70 –1.80	7.9	50.4	90.2	28.5
1.80 – 1.90	5.4	57.4	95.6	30.2
1.90 – 2.00	2.0	63.7	97.6	30.9
2.00 – 2.20	1.2	70.7	98.8	31.4
S 2.20	1.1	69.3	100.0	31.7

Table E.2 Float and Sink Analysis on 5x1 mm.

Relative Density	Mass Distribution %	Ash (ad) %	Cum. Yield %	Cum. Ash (ad) %
F 1.30	2.6	2.9	2.6	2.9
1.30 – 1.40 1.40	25.3	11.6	27.9	10.8
– 1.50	26.1	23.1	54.0	16.7
1.50 – 1.60	16.5	33.1	70.5	20.6
1.60 – 1.70	10.7	42.6	81.2	23.5
1.70 – 1.80	7.7	49.5	88.9	25.7
1.80 – 1.90	5.2	57.7	94.1	27.5
1.90 – 2.00	3.1	64.0	97.2	28.7
2.00 – 2.20	1.7	71.2	98.9	29.4
S 2.20	1.1	75.2	100.0	29.9

Table E.3 Float and Sink Analysis on 1x0.1 mm.

Relative Density	Mass Distribution %	Ash (ad) %	Cum. Yield %	Cum. Ash (ad) %
F 1.30	2.7	1.7	2.7	1.7
1.30 – 1.40	41.3	7.2	44.0	6.9
1.40 – 1.50	17.3	21.2	61.3	10.9
1.50 – 1.60	11.5	31.5	72.8	14.2
1.60 – 1.70	8.3	40.4	81.1	16.8
1.70 – 1.80	6.3	48.6	87.4	19.1
1.80 – 1.90	5.1	56.6	92.5	21.2
1.90 – 2.00	4.1	63.6	96.6	23.1
2.00 – 2.20	2.2	70.5	98.8	24.1
S 2.20	1.3	72.8	100.0	24.7

Table E.4 Float and Sink Analysis on -0.1 mm.

Relative Density	Cum. Yield %	Cum. Ash in floats %	Cum. Ash in sinks %
F 1.30	18.7	4.1	31.4
1.30 - 1.40	50.6	7.0	33.6
1.40 - 1.50	68.5	10.7	56.9
1.50 - 1.60	69.1	11.5	57.7
1.60 - 1.70	80.6	16.8	65.8
1.70 - 1.80	84.6	20.3	68.5
S 1.80	100.0	26.5	

Table E.5 Froth Flotation on −0.1 mm.

Product	Mass Distribution %	Ash (ad) %	Cum. Yield %	Cum. Ash (ad) %
1	70.9	13.5	70.9	13.5
2	4.7	32.7	75.7	14.6
3	1.9	37.5	77.6	15.2
4	4.3	38.5	81.9	16.5
5	5.2	45.2	87.1	18.1
6	2.4	56.3	89.5	19.2
7	3.9	58.7	93.3	19.4
8	6.7	68.8	100.0	22.7

Table E.6 Oil Agglomeration on -0.1 mm.

Oil I.D	Oil %	Time Minutes	Yield %	Ash in Agg. %	Ash in Tails %	Feed Ash %
	10	2'30	39.6	6.9	33.7	
	10	5'0	48.6	8.0	37.1	
S.SOL AB	10	8'30	46.7	8.8	30.0	
	20	8'0	77.7	12.7	61.7	
	20	8'0	76.9	12.5	63.7	
						25.7
	20	15'0	67.4	9.8	53.8	
	20	20'0	71.4	10.4	57.5	
S.SOL	20	8'0	52.6	7.2	43.4	
AB/K	10	8'30	31.6	5.9	32.8	
50/50	20	11'0	62.6	8.6	44.5	

APPENDIX F The Results of Float and Sink Analysis on Middlings

Table F.1 Float and Sink Analysis on 15x5 mm. (1)

Relative Density	Mass Distribution %	Ash (ad) %	Cum. Yield %	Cum. Ash (ad) %
F 1.30	-	-	-	-
1.30 - 1.40	2.4	14.7	2.4	14.7
1.40 - 1.50	36.8	23.1	39.2	22.1
1.50 - 1.60	29.2	31.9	68.4	26.3
1.60 - 1.70	17.2	40.6	85.6	29.2
1.70 - 1.80	5.9	46.4	91.5	30.3
1.80 - 1.90	2.4	56.2	93.9	30.9
1.90 - 2.00	1.2	65.5	95.1	31.4
2.00 - 2.20	2.1	76.9	97.2	32.4
S 2.20	2.8	81.4	100.00	33.7

Table F.2 Float and Sink Analysis on 5x1 mm. **(2)**

Relative Density	Mass Distribution %	Ash (ad) %	Cum. Yield %	Cum. Ash (ad) %
F 1.30	0.4	4.0	0.4	4.0
1.30 - 1.40	6.4	12.0	6.8	11.6
1.40 - 1.50	33.3	22.6	40.1	20.7
1.50 - 1.60	22.3	31.7	62.4	24.6
1.60 - 1.70	15.4	40.1	77.8	27.6
1.70 - 1.80	6.8	48.7	84.6	29.2
1.80 - 1.90	3.3	55.8	87.9	30.3
1.90 - 2.00	2.2	65.1	90.1	31.1
2.00 - 2.20	3.5	76.1	93.6	32.8
S 2.20	6.4	82.9	100.00	36.0

Table F.3 Float and Sink Analysis on 1x0.1 mm. (3)

Relative Density	Mass Distribution %	Ash (ad) %	Cum. Yield %	Cum. Ash (ad) %
F 1.30	4.3	2.0	4.3	2.0
1.30 - 1.40	22.6	6.2	26.9	5.5
1.40 - 1.50	11.1	20.7	38.0	10.0
1.50 - 1.60	10.1	30.6	48.1	14.3
1.60 - 1.70	7.4	39.0	55.5	17.6
1.70 - 1.80	5.1	47.2	60.6	20.1
1.80 - 1.90	4.5	54.5	65.1	22.5
1.90 - 2.00	4.1	62.3	69.2	24.8
2.00 - 2.20	6.3	71.8	75.5	28.7
S 2.20	24.6	81.9	100.00	41.8

Table F.4 Float and Sink Analysis on -0.1 mm.

Relative Density	Cum. Yield %	Cum. Ash in floats %	Cum. Ash in sinks %
F 1.30	10.7	7.5	40.7
1.30 - 1.40	36.4	9.1	54.9
1.40 - 1.50	49.3	11.8	61.3
1.50 - 1.60	58.3	14.9	68.4
1.60 - 1.70	62.3	16.9	72.4
1.70 - 1.80	67.9	20.0	74.3
S 1.80	100.00	40.1	

Table F.5 Float and Sink Analysis on -0.1 mm
Intermediate Size Crushed from 1x0.1 mm.**(3,1)**
(screened of 15x5 mm. Top Size)

Relative Density	Cum. Yield %	Cum. Ash in floats %	Cum. Ash in sinks %
F 1.30	0.5	1.6	43.5
1.30 - 1.40	19.0	4.9	51.3
1.40 - 1.50	23.6	7.9	53.1
1.50 - 1.60	40.8	13.1	62.8
1.60 - 1.70	49.8	15.9	68.4
1.70 - 1.80	57.2	18.1	74.6
S 1.80	100.0	42.9	

Table F.6 Float and Sink Analysis on 5x1 mm
Intermediate Size Crushed from 15x5 mm.**(1,1)**

Relative Density	Mass Distribution %	Ash (ad) %	Cum. Yield %	Cum. Ash (ad) %
F 1.30	0.3	3.1	0.3	3.1
1.30 - 1.40	7.8	11.7	8.1	11.4
1.40 - 1.50	23.8	22.4	31.9	19.6
1.50 - 1.60	22.3	31.4	54.2	24.5
1.60 - 1.70	16.1	38.7	70.3	27.7
1.70 - 1.80	10.5	45.3	80.9	30.0
1.80 - 1.90	6.9	52.1	87.7	31.8
1.90 - 2.00	3.3	57.7	90.9	32.7
2.00 - 2.20	3.9	65.3	94.9	34.0
S 2.20	5.1	81.7	100.0	36.5

Table F.7 Float and Sink Analysis on 1x0.1 mm
Intermediate Size Screened from -5 mm. **(1,2)**
(crushed of 15x5 mm.)

Relative Density	Mass Distribution %	Ash (ad) %	Cum. Yield %	Cum. Ash (ad) %
F 1.30	1.6	1.9	1.6	1.9
1.30 - 1.40	20.7	8.3	22.3	7.8
1.40 - 1.50	18.6	20.4	40.9	13.6
1.50 - 1.60	14.3	27.9	55.2	17.3
1.60 - 1.70	12.0	35.5	67.3	20.6
1.70 -1.80	7.9	42.3	75.2	22.9
1.80 - 1.90	6.7	49.1	81.9	25.0
1.90 - 2.00	8.9	50.4	90.8	27.5
2.00 - 2.20	4.0	55.9	94.8	28.7
S 2.20	5.2	80.9	100.0	31.4

Table F.8 Float and Sink Analysis on -0.1 mm
Intermediate Size Screened from -5 mm. **(1,3)**
(crushed of 15x5 mm.)

Relative Density	Cum. Yield %	Cum. Ash in floats %	Cum. Ash in sinks %
F 1.30	1.1	3.9	31.5
1.30 - 1.40	19.8	5.3	39.3
1.40 - 1.50	38.8	9.6	46.9
1.50 - 1.60	55.1	12.9	55.5
1.60 - 1.70	65.7	16.9	61.0
1.70 - 1.80	73.2	18.9	67.7
S 1.80	100.0	32.5	

Table F.9 Float and Sink Analysis on 1x0.1 mm.
Intermediate Size Crushed from 5x1 mm.**(1,12)**
(crushed of 15x5 mm.)

Relative Density	Mass Distribution %	Ash (ad) %	Cum. Yield %	Cum. Ash (ad) %
F 1.30	0.4	3.4	0.4	3.4
1.30 - 1.40	11.2	9.6	11.6	9.4
1.40 - 1.50	22.1	19.7	33.7	16.1
1.50 - 1.60	19.5	29.7	53.1	20.9
1.60 - 1.70	13.1	36.1	66.2	23.9
1.70 - 1.80	7.7	42.2	73.9	25.9
1.80 - 1.90	9.6	48.5	83.6	28.5
1.90 - 2.00	5.7	55.1	89.7	30.2
2.00 - 2.20	5.5	64.4	94.7	32.2
S 2.20	5.2	80.6	100.0	34.7

Table F.10 Float and Sink Analysis on -0.1 mm.
Intermediate Size from 1x0.1/15x5 mm. **(1,13)**

Relative Density	Cum. Yield %	Cum. Ash in floats %	Cum. Ash in sinks %
F 1.30	1.9	5.2	36.8
1.30 - 1.40	12.7	6.8	39.6
1.40 - 1.50	29.0	11.9	44.1
1.50 - 1.60	54.3	47.7	55.7
1.60 - 1.70	57.3	19.1	60.8
1.70 - 1.80	68.3	21.0	66.8
S 1.80	100.0	35.3	

Table F.11 Float and Sink Analysis on -0.1 mm.

Intermediate Size Crushed from 1x0.1 mm.**(1,121)** (screened of -1 mm.)

Relative Density	Cum. Yield %	Cum. Ash in floats %	Cum. Ash in sinks %
F 1.30	5.5	2.8	33.4
1.30 - 1.40	25.1	6.5	40.4
1.40 - 1.50	46.7	11.9	47.5
1.50 - 1.60	62.9	16.1	56.3
1.60 - 1.70	71.9	17.7	62.3
1.70 - 1.80	79.2	19.9	68.6
S 1.80	100.0	31.1	

Table F.12 Float and Sink Analysis on -0.1 mm.

Intermediate Size Crushed from -1 mm.**(1,21)** (screened of 5x1 mm.)

Relative Density	Cum. Yield %	Cum. Ash in floats %	Cum. Ash in sinks %
F 1.30	2.7	3.3	36.6
1.30 - 1.40	16.8	7.1	41.9
1.40 - 1.50	39.5	13.1	49.1
1.50 - 1.60	56.8	18.3	57.3
1.60 - 1.70	63.9	19.2	62.4
1.70 - 1.80	71.9	21.0	97.7
S 1.80	100.0	35.3	

Table F.13 Float and Sink Analysis on 1x0.1 mm
Intermediate Size Crushed from 5x1 mm. **(2,1)**
(screened of 15x5 mm. Top Size)

Relative Density	Mass Distribution %	Ash (ad) %	Cum. Yield %	Cum. Ash (ad) %
F 1.30	0.6	2.9	0.6	2.9
1.30 - 1.40	15.6	8.8	16.3	8.6
1.40 - 1.50	18.1	19.9	34.4	14.6
1.50 - 1.60	15.5	29.3	49.9	19.1
1.60 - 1.70	16.5	34.5	66.4	22.9
1.70 - 1.80	7.9	44.2	74.4	25.2
1.80 - 1.90	6.6	50.6	80.9	27.3
1.90 - 2.00	4.5	57.1	85.4	28.9
2.00 - 2.20	5.2	65.8	90.7	30.9
S 2.20	9.3	82.9	100.0	35.8

Table F.14 Float and Sink Analysis on -0.1 mm
Intermediate Size Screened from -1 mm. (2,2)
(crushed of 5x1 mm.)

Relative Density	Cum. Yield %	Cum. Ash in floats %	Cum. Ash in sinks %
F 1.30	1.3	0.3	37.3
1.30 - 1.40	11.3	5.1	41.7
1.40 - 1.50	27.9	9.9	47.6
1.50 - 1.60	49.8	14.7	58.6
1.60 - 1.70	58.5	17.3	65.9
1.70 - 1.80	65.9	19.9	71.5
S 1.80	100.0	37.2	

Table F.15 Float and Sink Analysis on -0.1 mm.
Intermediate Size Crushed from
1x0.1 mm. **(2,12)** (screened of -1 mm.)

Relative Density	Cum. Yield %	Cum. Ash in floats %	Cum. Ash in sinks %
F 1.30	2.5	2.5	36.6
1.30 - 1.40	15.6	6.3	42.6
1.40 - 1.50	35.3	12.7	48.9
1.50 - 1.60	56.2	18.1	59.6
1.60 - 1.70	63.1	19.3	65.3
1.70 - 1.80	71.0	21.2	70.6
S 1.80	100.0	36.2	

Table F.16 Froth flotation on -0.1 mm

Product	Mass Distribution %	Ash (ad) %	Cum. Yield %	Cum. Ash (ad) %
1	54.5	17.1	54.5	17.1
2	1.9	43.5	56.5	17.9
3	4.4	45.4	60.9	19.9
4	12.7	49.5	73.6	25.1
5	3.3	49.9	76.9	26.1
6	4.4	66.1	81.3	28.3
7	4.4	66.5	85.7	30.3
8	14.3	76.5	100.0	36.9

Table F.17 Oil Agglomeration on -0.1 mm.

Oil I.D	Oil %	Time Minutes	Yield %	Ash in Agg. %	Ash in Tails %	Feed Ash %
	20	8'0	42.1	9.0	56.8	
	20	20'0	57.3	12.0	66.5	
S.SOL AB	10	20'0	38.2	8.6	52.4	39.7
	10	8'0	36.0	8.1	51.6	
	15	20'0	55.3	11.8	65.8	

APPENDIX G The Results of Float and Sink Analysis on Teska Discard

Table G.1 Float and Sink Analysis on 15x5 mm.

Relative Density	Mass Distribution %	Ash (ad) %	Cum. Yield %	Cum. Ash (ad) %
F 1.30	–	–	–	–
1.30 – 1.40	–	–	–	–
1.40 – 1.50	0.5	22.1	0.5	22.1
1.50 – 1.60	0.7	33.1	1.2	28.5
1.60 – 1.70	1.4	42.0	2.6	35.7
1.70 – 1.80	2.7	49.0	5.3	42.5
1.80 – 1.90	3.7	55.9	9.0	48.0
1.90 – 2.00	3.0	63.2	12.0	51.8
2.00 – 2.20	9.0	73.2	21.0	60.9
S 2.20	78.9	86.7	100.0	81.2

Table G.2 Float and Sink Analysis on 5x1 mm.

Relative Density	Mass Distribution %	Ash (ad) %	Cum. Yield %	Cum. Ash (ad) %
F 1.30	–	–	–	–
1.30 – 1.40	–	–	–	–
1.40 – 1.50	1.3	15.1	1.3	15.1
1.50 – 1.60	0.9	31.1	2.2	21.6
1.60 – 1.70	1.4	41.0	3.6	29.2
1.70 – 1.80	2.1	48.8	5.7	36.4
1.80 – 1.90	3.1	56.0	8.8	43.3
1.90 – 2.00	3.3	61.4	12.1	48.2
2.00 – 2.20	8.8	71.9	20.9	58.2
S 2.20	79.0	86.7	100.0	80.7

Table G.3 Float and Sink Analysis on 1x0.1 mm.

Relative Density	Mass Distribution %	Ash (ad) %	Cum. Yield %	Cum. Ash (ad) %
F 1.30	0.3	3.3	0.3	3.3
1.30 – 1.40	2.2	8.1	2.5	7.5
1.40 – 1.50	1.7	17.8	4.2	11.7
1.50 – 1.60	1.7	26.9	5.9	18.7
1.60 – 1.70	1.2	36.2	7.1	21.7
1.70 – 1.80	1.7	45.2	8.8	26.2
1.80 – 1.90	2.1	53.4	10.9	31.5
1.90 – 2.00	2.7	60.8	13.6	37.3
2.00 – 2.20	7.3	70.2	20.9	48.8
S 2.20	79.2	84.6	100.0	77.2

Table G.4 Float and Sink Analysis on −0.1 mm

Relative Density	Cum. Yield %	Cum. Ash in floats %	Cum. Ash in sinks %
F 1.30	1.5	6.3	66.0
1.30 – 1.40	6.1	10.2	68.0
1.40 – 1.50	13.5	14.1	73.4
1.50 – 1.60	17.7	16.4	75.7
1.60 – 1.70	21.1	20.5	77.5
1.70 – 1.80	26.1	23.2	76.3
S 1.80	100.0	65.1	

Table G.5 Froth Flotation on -0.1 mm.

Product	Mass Distribution %	Ash (ad) %	Cum. Yield %	Cum. Ash (ad) %
1	26.9	31.5	26.9	31.5
2	0.9	65.9	27.8	32.5
3	23.6	67.8	51.4	48.7
4	2.2	69.1	53.6	49.6
5	2.4	69.6	56.0	50.4
6	11.2	77.9	67.2	55.0
7	3.1	78.7	70.3	56.0
8	29.8	84.3	100.0	64.4

Table G.6 Oil Agglomeration on −0.1 mm.

Oil I.D	Oil %	Time Minutes	Yield %	Ash in Agg. %	Ash in Tails %	Feed Ash %
	20	60'0	29.3	20.7	81.7	
S.SOL AB	20	30'0	26.3	19.9	80.2	64.6
	10	30'0	25.7	1[illegible].6	80.2	

Appendix H Output of Simulation Package

Table H.1 The simulation of washability data on 15 mm top size of middlings

d50	1.46
Epm*	0.01
Imp**	0.01
Near density of 0.10%	54.00
Near density of 0.05%	34.00
Product Yield%	26.00
Product Ash %	17.66
Discard Yield%	74.00
Discard Ash %	39.08
Theoretical Yield*%**	26.50
Organic Efficiency**%**	98.90
m.m***S in F %**	2.30
m.m F in S %	1.90
Σm.m %	4.20
RFA	33.40

* Epm = (d25 – d75)/2

** Imp = Epm/d50

*** Theoretical Yield = Maximum theoretical yield @ product ash

**** Organic Efficiency = 100x(product yield/max theoretical yield@product ash)

***** Misplaced material as % of feed

RFA Reconstituted Feed Ash

Table H.2 The simulation of washability data on 5 mm top size of middlings

d50	1.38
Epm	0.02
Imp	0.01
Near density of 0.10%	30.0
Near density of 0.05%	16.0
Product Yield%	10.2
Product Ash %	9.0
Discard Yield%	89.8
Discard Ash %	41.9
Theoretical Yield%	12.1
Organic Efficiency%	84.3
m.m S in F %	1.4
m.m F in S %	2.5
Σm.m %	3.9
RFA	38.6

Table H.3 The simulation of washability data on 1 mm top size of middlings

d50	1.40
Epm	0.04
Imp	0.03
Near density of 0.10%	31.0
Near density of 0.05%	20.0
Product Yield%	18.3
Product Ash %	10.0
Discard Yield%	81.7
Discard Ash %	41.3
Theoretical Yield%	23.9
Organic Efficiency%	76.4
m.m S in F %	4.1
m.m F in S %	4.6
Σm.m %	8.8
RFA	35.6

Table H.4 The simulation of washability data on 15 mm top size of discard

d50	2.32
Epm	0.06
Imp	0.03
Near Density Of 0.10%	29
Near Density of 0.05%	14
Product Yield%	40.4
Product Ash%	64.6
Discard Yield%	59.6
Discard Ash%	86.3
Theoretical Yield%	44.7
Organic Efficiency%	89.6
m.m S in F%	5.6
m.m F in S%	5.9
m.m%	11.6
RFA	77.2

Table H.5 The simulation of washability data on 5 mm top size of discard

d50	2.34
Epm	0.09
Imp	0.04
Near Density Of 0.10%	28
Near Density of 0.05%	14
Product Yield%	42
Product Ash%	64.6
Discard Yield%	57.9
Discard Ash%	88.5
Theoretical Yield%	47.9
Organic Efficiency%	87.7
m.m S in F%	7.1
m.m F in S%	7.0
m.m%	14.1
RFA	78.4

Table H.6 The simulation of washability data on 1 mm top size of discard

d50	2.52
Epm	0.05
Imp	0.02
Near Density Of 0.10%	28
Near Density of 0.05%	14
Product Yield%	65
Product Ash%	64.6
Discard Yield%	34.9
Discard Ash%	95.4
Theoretical Yield%	60.6
Organic Efficiency%	97.7
m.m S in F%	4.3
m.m F in S%	4.0
m.m%	8.3
RFA	75.3

Appendix I The example of calculation of capital and operating costs by either froth flotation or oil agglomeration on -0.1 mm size fraction.

Table I.1 The cost estimation for middlings (15 mm top size)

15 mm (by froth flotation)

CIRCUIT	T/H	EQUIPMENT	CAPITAL COST(Rm)
Classification	720	Cyclone	3.4
Separation	714.3	D.M Cyclone	38.5
Filtration	185.8	Bask. Centr.	0.2
Grinding	528.5	Rod Mill	29.56
Flotation	534.2	Froth Flotation	38.5
Filtration	291.2	Horz.Belt Filt.	33.8
Tails Thickener	243.0	40m	12.9
Tails Filter	243.0		28.2
Conveyors+handling			5%

TOTAL CAPITAL COST

185.06

<u>+ 9.25 (5%)</u>

R194.31m

CIRCUIT	T/H	OPERATING COST(Rm)
Classification	720	?
D.M Separation	714.3	6.1
Filtration	185.8	0.1
Grinding(-0.1)	528.5	2.6
Flotation	534.2	6.4
Filtration	291.2	0.6
Tails Thickener	243	?
Tails Filter	243	0.5
Conveyors+handling		5%

The operating cost for classification & tails thickener is assumed to be R3m.

Operating Cost= R20.3m

Maintenance Labour and Supplies = 0.15 x 194.31

= R29.1m

Operating Supplies = 0.15 x 29.1

= R4.4R

TOTAL OPERATING COST= R53.8m

PRODUCT 2.4 mt/annum @ 17% ash

INTEREST RATE 25%

PERIOD TO PAY BACK	PRICE OF COAL
Year	R/t
2	72
2.5	63
3	56
3.5	52
4	49

15 mm (by oil agglomeration)

CIRCUIT	T/H	EQUIPMENT	CAPITAL COST(Rm)
Classification	720	Cyclone	3.4
Separation	714.3	D.M Cyclone	38.5
Filtration	185.8	Bask. Centr.	0.2
Grinding	528.5	Rod Mili	29.56
Oil agglomeration	534.2		19.2
Filtration	204.5	Horz.Belt Filt.	23.7
Tails Thickener	329.8	40m	17.5
Tails Filtration	329.8	Horz.Belt Filt.	38.2
Conveyors+handling			5%

TOTAL CAPITAL COST

170.3

± 8.5 (5%)

178.8 mR

CIRCUIT	T/H	OPERATING COST(Rm)
Classification	720	?
D.M Separation	714.3	6.1
Filtration	185.8	0.1
Grinding(-0.1)	528.5	2.6
Oil Agglomeration	534.2	19.1
Filtration	204.5	0.4
Tails Thickener	329.8	?
Tails Filter	329.8	0.7
Conveyors+handling		5%

The operating cost for classification & tails thickener is assumed to be R3m.

Operating Cost= R33.6m

Maintenance Labour and Supplies = 0.15 x 178.8

= R26.8m

Operating Supplies = 0.15 x 26.8

= R4.0m

TOTAL OPERATING COST= R64.4m

PRODUCT 1.94 mt/annum @ 13% ash

INTEREST RATE 25%

PERIOD TO PAY BACK	PRICE OF COAL
Year	R/t
2	90
2.5	80
3	73
3.5	68
4	64

Table I.2 The cost estimation for discard(15 mm top size)

15 mm (by froth flotation)

CIRCUIT	T/H	EQUIPMENT	CAPITAL COST(Rm)
Crushing	1950		3.9
Classification	1950	Cyclone	9.2
D.M Separation	1918	Cyclone	103.5
Filtration	1139	Bask. Centr.	1.2
Grinding	780	Rod Mill	43.5
Flotation	811.2		58.4
Filtration(cons.)	226.2		26.2
Tails Thickener	585		31.1
Filtration(tails)	585		67.9
Con :yors+handling			5%

TOTAL CAPITAL COST

345

+ 17.2 (5%)

R362.25m

CIRCUIT	T/H	OPERATING COST(Rm)
Crushing	1950	2.4
Classification	1950	?
D.M Separation	1918	16.4
Filtration	1139	0.5
Grinding(-0.1)	780	3.9
Flotation	811.2	9.7
Filtration(cons)	226.2	0.5
Tails Thickener	585	?
Filtration(tails)	585	1.3
Conveyors+handling		5%

The operating cost for classification & tails thickener is assumed as R3m.

Operating Cost= R39.48m

Maintenance Labour and Supplies = 0.15 x 362.25

= R54.3m

Operating Supplies = 0.15 x 54.3

= R8.1R

TOTAL OPERATING COST = R101.88m

Product 1.1 mt/annum @ 32.5% ash

INTEREST RATE 25%

It should be noted that R5 per ton is already spent to remove the discard at the present preparation plant.

PERIOD TO PAY BACK	PRICE OF COAL
Year	R/t
2	329.5
2.5	303.9
3	270.7
3.5	261
4	255.5
4.5	252.7
5	251.9

15 mm (by oil agglomeration)

CIRCUIT	T/H	EQUIPMENT	CAPITAL COST(Rm)
Crushing	1950		3.9
Classification	1950	Cyclone	9.2
D.M Separation	1918	Cyclone	103.5
Filtration	1139	Bask. Centr.	1.2
Grinding	780	Rod Mill	43.5
Oil agglomeration	811.2		29.2
Filtration(cons.)	234		27.1
Tails Thickener	577.2		30.7
Filtration(tails)	577.2		66.9
Conveyors+handling			5%

TOTAL CAPITAL COST

315.3

+ 15.8 (5%)

R331.1m

CIRCUIT	T/H	OPERATING COST(Rm)
Crushing	1950	2.4
Classification	1950	?
D.M Separation	1918	16.4
Filtration	1139	0.5
Grinding(-0.1)	780	3.9
Oil agglomeration	711.2	29
Filtration(cons)	234	0.8
Tails Thickener	577.2	?
Filtration(tails)	577.2	1.3
Conveyors+handling		5%

The operating cost for classification & tails thickener is assumed as R3m.

Operating Cost= R60.06m

Maintenance Labour and Supplies = 0.15 x 331.1

= R49.7m

Operating Supplies = 0.15 x 49.7

= R7.4m

TOTAL OPERATING COST = R117.16m

Product 1.2 mt/annum @ 20.7% ash

PERIOD TO PAY BACK	PRICE OF COAL
Year	R/t
2	287.7
2.5	266
3	253.7
3.5	246.8
4	243.3
4.5	242.3
5	242.9

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