AN INVESTIGATION INTO THE FROTH FLOTATION OF FOUR SOUTH AFRICAN COALS

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SYNOPSIS

Froth flotation is used extensively throughout the world to beneficiate fine (-0,5 mm) coal. However when the same technology is applied to the majority of South African coals it has proved relatively unsuccessful. As the beginning of an ongoing research program, this thesis examines the effects of basic flotation parameters on the beneficiation by flotation, of four South African coals, to identify research areas where a more in-depth investigation is needed. An important conclusion is that South African coals are floatable, and flotation selectivity can be obtained, provided process parameters are controlled within a narrow range. A more in-depth study of the liberation characteristics of the coal and gangue and of the individual macerals is needed because only this will limit the final result. Finally, more research is needed to identify chemicals which exhibit a high degree of specific adsorption on the individual coal components, so that ultimately maceral selectivity can be achieved.

SAMEVATTING

Skuim flottasie word dwarsdeur die wêreld tot 'n groot mate gebruik om fyn (-0,5 mm) steenkool te benefisieer. Wanneer hierdie tegnologie egter gebruik word vir die grootste gedeelte van Suid-Afrikaanse steenkool, is die resultate vergelykend onsuksesvol. Hierdie tesis ondersoek die effekte van basiese flottasie parameters op opgradering deur flottasie van vier verskillende Suid-Afrikaanse steenkole, synde die begin van 'n uitgebreide navorsingsprogram, om die navorsingsgebiede waar 'n dieperliggende ondersoek nodig is te identifiseer. 'n Belangrike gevolgtrekking is dat Suid-Afrikaanse steenkole wel flotteerbaar is, en die selektiwiteit benodig kan verkry word, solank sekere proses parameters in 'n nou band gekontrolleer word. 'n Dieperliggende studie word egter benodig op die vrystellings karakteristiek van die steenkool en uitskot, want slegs hierdie sal die uiteindelike resultaat aan bande lê. Ten slotte is verdere navorsing nodig om die chemikaliëe te identifiseer wat 'n hoë graad van spesifieke adsorpsie op individuele steenkool komponente sal openbaar, sodat selektiwiteit ten opsigte van maseraal uiteindelik kan bereik word.

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CHAPTER 1

INTRODUCTION

1.1 BACKGROUND

In the South African coal mining industry the minus 0,5 mm coal is considered to be fines. However with low screening and cyclone efficiencies the actual top size in the fines fraction may be 1 mm.

Because of the size distribution, the fines fraction should contain a large proportion of liberated material with the potential to produce a very low ash product. However the majority of South African coal fines are not beneficiated, but are either discarded or added to middlings material for use as thermal coal. Almost a decade ago, Horsfall (1976) pointed out that this latter alternative may be false economy for the unbeneficiated fines generally contain high ash material and a high water content resulting in uneconomic plant operation and lower efficiencies.

Changing circumstances altered the situation. The percentage of run-of-mine coal fines produced in this country has escalated markedly primarily due to the increasing use of mechanical mining methods. The percentage of run-of-mine coal reporting in the fines fraction may be as high as 16 per cent (Anon, 1985). This increased loss of coal, together with the reduced supply of naturally clean coals, the need to supply low ash coal for gasification, liquefaction and metallurgical purposes, as well as the price differential between low ash and thermal coal all provide an incentive for developing a viable method of fines beneficiation for South African coals.

Some of the methods of fines beneficiation may be listed as follows:

froth flotation,
oil agglomeration,
selective flocculation,
water only cyclones,
dense medium cyclones,
jigs,
shaking tables
and spirals.

The first three processes utilise differences in surface chemical characteristics between the valuable component and the gangue to effect their separation; the others rely on differences in specific gravity (i.e. bulk properties). Generally beneficiation processes that use differences in bulk properties are more efficient and cost effective.

Below a particle size of 0,1 mm the efficiency of the density separation processes drops off rapidly for coal particles (Horsfall, 1976) because of viscous and drag forces. Froth flotation, oil agglomeration and selective flocculation could be used to upgrade the minus 0,1 mm particles. However, the latter two processes have not been used in commercial applications on South African coals, nor has any testwork been published.

In view of the proven nature of froth flotation as a technically satisfactory process - millions of tons of fine material are recovered every year by this method, throughout the world - it may be argued that this is one of the most viable alternatives that ought to be considered.

Indeed froth flotation is practised extensively overseas. Especially in Europe and North America, it is an easy process, which uses the natural hydrophobicity

of the coal in conjunction with a frother and occasionally a non-polar collector such as paraffin or kerosene. This has resulted in efficient recovery and optimal production of low ash coals, so in-depth investigations of the mechanisms involved have not been necessary.

Unfortunately, except for the Natal coking coals, the same technology applied to South African coals has proved relatively unsuccessful. This has resulted in South African coal fines generally not being cleaned, although it can be shown that the composition of the majority of South African coals is quite different from that of overseas coals. They have different relative proportions of petrographic constituents and a much higher percent of middlings. It is not surprising therefore that the identical technology does not produce the desired results.

A more detailed study of the froth flotation of South African coals (which does take due account of the different nature of local coals) should be undertaken to establish whether the process may be a viable means of fine coal beneficiation.

1.2 AIM

The aims of this project were to provide a basis for a fundamental investigation into the floatability of South African coals and to establish the optimum process parameters for obtaining the best grades and recoveries. In particular, high vitrinite content coals exhibiting a range of ranks were examined. Vitrinite is the most important constituent of coking coals for which froth flotation is known to be a viable process. Vitrinite is also the most chemically reactive component of coal.

Because the chief objective of this thesis was to

provide the basis for a more fundamental study of coal flotation, the effects of primary process variables were investigated. This included different collector and frother dosages, pulp densities, aeration rates, impeller speeds and particle size distributions.

To extend the validity of this study, four different coals were chosen for investigation, on the basis of their rank and maceral compositions. These samples were obtained from Grootegeluk, Greenside, Nonsana and Longridge Collieries. To obtain the most information from the results, a factorial design method was used to plan the experiments and analyse the results.

In the following chapters, this thesis outlines the findings of a literature survey, undertaken to identify the most important factors that have been found to affect coal flotation; describes the choice and preparation of samples and equipment, for the experimental work, and the establishment of analytical techniques and experimental methods; and details the results obtained from the test-work. These results are discussed and recommendations for further tests are proposed.

CHAPTER 2

LITERATURE SURVEY

Before any beneficiation technique can be evaluated it is necessary to investigate the nature of the feed material and details of the process. In this chapter the general nature of coal is discussed and attention is focussed on the properties of South African coals compared with European and North American coals. Finally the basic principles of froth flotation are discussed before the effects of process variables are investigated.

2.1 NATURE OF COAL

Coal is a heterogeneous combustible material consisting of a variety of carbonaceous and inorganic mineral substances. A specific coal may not be characterised by the conventional proximate and ultimate analyses only.

Most of the chemical, physical and technological properties of a coal will be determined by its grade, type and rank. Grade, type and rank are the result of the deposition, biochemical degradation and metamorphosis of the original organic constituents. Determination of these properties involves the microscopic study, identification and classification of various carbon remnants and is known as coal petrology.

2.1.1 Coal Type or Petrography

The coal type is determined by the nature of the dominant, original plant material that was deposited in the swamps prior to peatification. Coal type is also influenced by the mode and extent of the subsequent

biochemical degradation of the plant material.

The end products of the conversion consist of a number of distinct organic materials, known as macerals. Some of the macerals are genetically related but others are independent entities.

There are three groups of macerals which represent broad groups of carbonaceous material. Each group exhibits similar chemical, physical and technological properties. The three groups are

(a) vitrinite

(b) exinite

(c) inertinite.

Table 2.1 shows some of the relative properties of the various macerals.

The relative proportions and the association of the macerals (called microlithotypes) as well as the degree of oxidation and metamorphosis (or rank) determine the technological and beneficiation properties of a coal (Falcon, 1977; Falcon and Falcon, 1982). Figure 2.1 shows the most common microlithotypes and their component macerals.

2.1.2 Coal Rank or Degree of Coalification

The rank of a coal is not a directly measurable quantity but is determined by the degree or extent of metamorphosis of the coal. Maturation of the original organic material will depend on:

(a) The burial depth of the coal (pressure)

(b) The geothermal gradient (temperature)

(c) The duration of heating and pressure (time)

(d) The occurrence of igneous instrusions (temperature).

TABLE 2.1

Physical and Chemical Properties
of the Different Macerals (Falcon (a), (b), 1978)

	VITRINITE	EXINITE	INERTINITE
Origin	Woody plants : Anaerobic Decomposition	Chemically Resistant Parts of a Plant *e g* spores, algae	Woody plants : Aerobic Decomposition
Reactivity	Very reactive	Reactive	Inert except for reactive semi-fusinite
Volatile Composition	High : Important component in coke manufacture	Very high : Yields oil/gas by-products when heated	Inert in the range of coking coals
Ease of Oxidation	Easily oxidised : Combusts spontaneously	Less easily oxidised	Very difficult to oxidise
Breakability	Brittle : Breaks or fractures relatively easily	Stronger : Impart strength to the microlithotypes in which they occur	
Chemical Composition	Oxygen Rich	Hydrogen Rich	Carbon Rich
Specific Gravity (Low Rank)	1,2 - 1,3	1,1	1,4 - 1,6

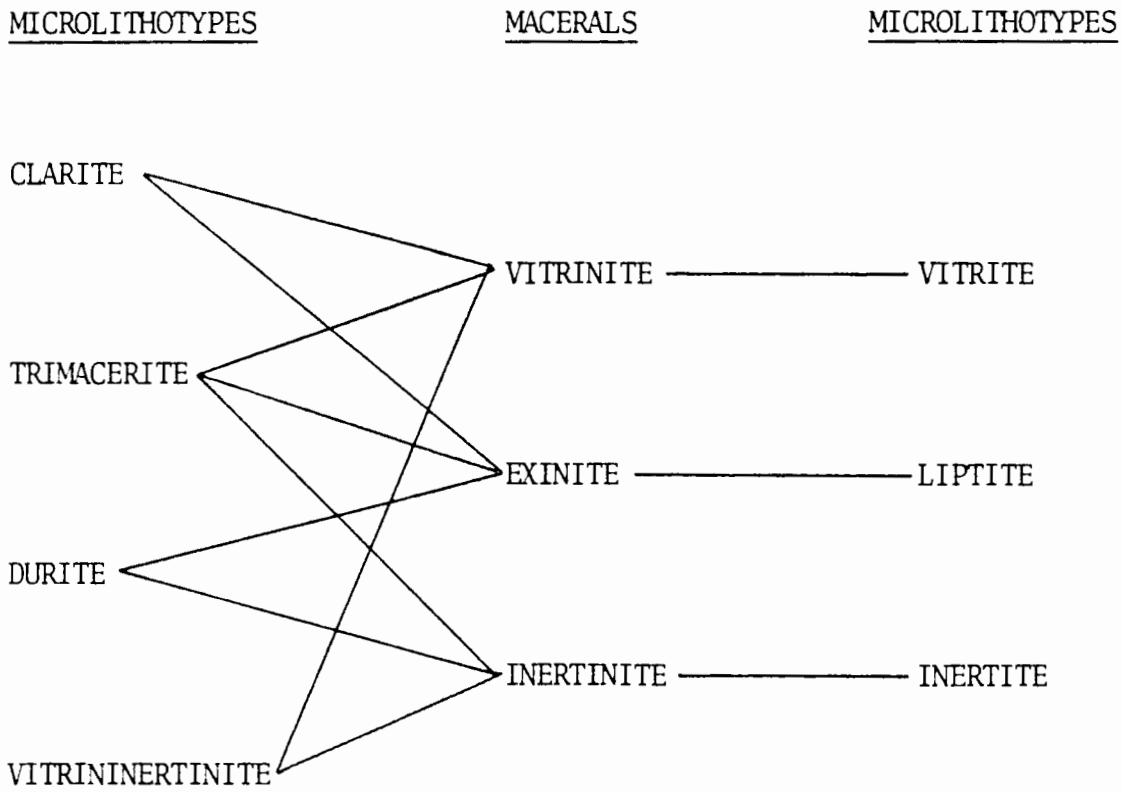

Figure 2.1 The Characteristic Associations of the Macerals to form Microlithotypes.

The effect of pressure and temperature will be to change the physical and chemical properties respectively, of the different macerals. The magnitude of these effects will vary depending on the individual macerals.

With increasing rank the different macerals are affected as follows (Falcon, 1978 (a), (b)):

(a) Vitrinite - increases in carbon but decreases in volatile matter composition

(b) Exinite - is affected in four stages:
  (i) increases in volatile matter particularly hydrogen;
  (ii) increases in oil products;
  (iii) increases in gaseous products;
  (iv) inert

(c) Inertinite - changes very little except for reactive semi-fusinite which behaves like vitrinite.

With Northern Hemisphere coals it has been possible to empirically correlate rank with a specific chemical or physical property such as volatile matter or carbon content. The diversity of South African coals does not yield a sufficiently good correlation. It is possible, however, to use vitrinite as a reference material because it is the most homogeneous petrographic component. Its characteristics change continuously and linearly with rank; it has the lowest ash content; and it is easily identified by microscopic examination. Rank determinations are thus carried out by reflectance measurements according to the specifications proposed by the International Committee for Coal Petrology (1963, 1971).

2.1.3 Coal Grade or Gangue Mineral Association

The minerals occurring in coal are widely variable in chemical and physical properties. However it is

possible to subdivide the minerals on the basis of their origin.

(a) Syngenetic or Inherent Minerals - these are accumulated in coal swamps up to the time of peatification. They are finely disseminated and intimately intergrown with the coal and are difficult to remove in beneficiation operations. Coal containing minerals in this form is known as middlings material.

(b) Epigenetic or Extraneous Minerals - these are deposited in cleats and fissures in an existing coal seam and are therefore more easily removed in beneficiation operations.

The relative proportions of minerals occurring in either of these two forms will determine the liberation potential of a clean coal or its ultimate ash content.

In South African coals, clays constitute 60 to 70 per cent of the mineral intergrowth, with the major components being kaolinite, illite and chorite; the relative proportions of each depends on the rank of the coal in which they occur. They are associated with all the maceral groups and are therefore the most difficult to liberate.

Different macerals are associated with different types of clay by virtue of the prevailing environmental conditions at the time of deposition. Vitrinite contains colloidal fine grained clays while inertinite contains irregularly shaped grains as a result of wind or water borne detritus. Other minerals associated with coal are carbonates, sulphides, quartz and glauconites.

2.1.4 Coal Characterisation

The conventional use of proximate and ultimate analyses gives no indication of the technological and

beneficiation properties of a coal. As has been shown, the coal petrography must be determined in order to infer these properties.

Falcon (1977, 1978 (a), (b)) proposed that in order to fully characterise a coal the following should be known

(a) Ratio of reactive to inert maceral groups

(b) Identification of maceral groups

(c) Proportion and type of microlithotypes

(d) Type, form and proportion of mineral matter

(e) Rank measurement by the reflectance of vitrinite

(f) Degree of oxidation.

2.1.5 Some Differences between South African and European Coals

The major differences between South African and European coals lie in the relative proportions of the different macerals and the amounts of middlings coal.

As can be seen from Table 2.2, South African coals tend to be inertinite enriched, which makes them generally less reactive than the equivalent European coals (except for Natal coals). They also contain a larger proportion of syngenetic minerals, which means that they contain a smaller amount of coal which may be easily beneficiated.

South African coals tend to be chemically rather than physically changed because not only are pressure effects smaller (shallower burial depth) but temperature effects associated with widespread igneous intrusions are more significant.

South African coals were decomposed in a positive redox environment. They therefore contain a high

TABLE 2.2

The Average Relative Proportions of Coal Constituents in South African and European Coals (Falcon, 1978 (a))

	Average Relative Proportions (%)	
	South Africa	Europe
Vitrinite	40	70
Exinite	0 - 5	15
Inertinite	60 (± 20)	15
Syngenetic Mineral	14	3

proportion of silica-alumina enriched clay minerals and relatively small amounts of syngenetic carbonates and iron sulphides. This applies to the Free State and Transvaal coals but not the Natal coals which exhibit similar characteristics to the European coals.

Finally, unlike European coals in which the proportion of reactive semi-fusinite (RSF) to other macerals is constant, South African coals contain variable proportions of RSF. This is the transition material between reactive vitrinite and inert semi-fusinite.

2.2 PRINCIPLES OF FROTH FLOTATION

Beneficiation utilises differences in properties between the valuable component and the gangue to effect their separation.

Froth flotation is a physico-chemical method of beneficiating a pulp of finely ground ore which involves imparting a hydrophobic character to the desired mineral surface by chemicals called collectors. Under favourable conditions, the chemical-coated particles become attached to air that is bubbled through the pulp. With the addition of a frother, a stabilised mineral-laden froth forms at the surface which may be skimmed off to yield a concentrate.

This separation of the valuable mineral from the gangue depends on differences in their surface properties. Flotation involves the development of specific characteristics in one or more of the mineral species such that there is an increase in the force of adhesion of the desired particles to the air bubble.

2.2.1 Contact Angle

A method of quantifying the degree of adherence of a mineral to an air bubble is by measuring the contact angle Θ between the air bubble and the mineral surface. (Figure 2.2)

The contact angle increases as the hydrophobicity of the mineral increases. Horsley and Smith (1952) used contact angle measurements to study the influence of different collectors on the efficiency of flotation.

Rogers (1976) showed that any factor which influences the contact angle will also affect the flotation of a mineral, although the converse is not true. The measurement of contact angles might give an indication of the expected success of a flotation process.

2.2.2 Flotation as an Equilibrium/Rate Controlled Process

Traditionally the success of a flotation process in the laboratory or on the industrial scale has been measured largely on the basis of a rougher recovery at a sufficiently long time to yield an equilibrium condition (Dowling *et al* 1984). A tacit assumption has been that the time-recovery profile is not affected by changes in the flotation parameters.

In fact the recovery-time and recovery-grade profiles have also been used to determine the relative efficiency of flotation under different sets of conditions. The aim of flotation is to increase both the rate of flotation and the equilibrium recovery of the valuable material relative to those of the gangue. However there is usually a trade-off between recovery and rate which is largely determined by the average residence time of the pulp in the industrial flotation cell.

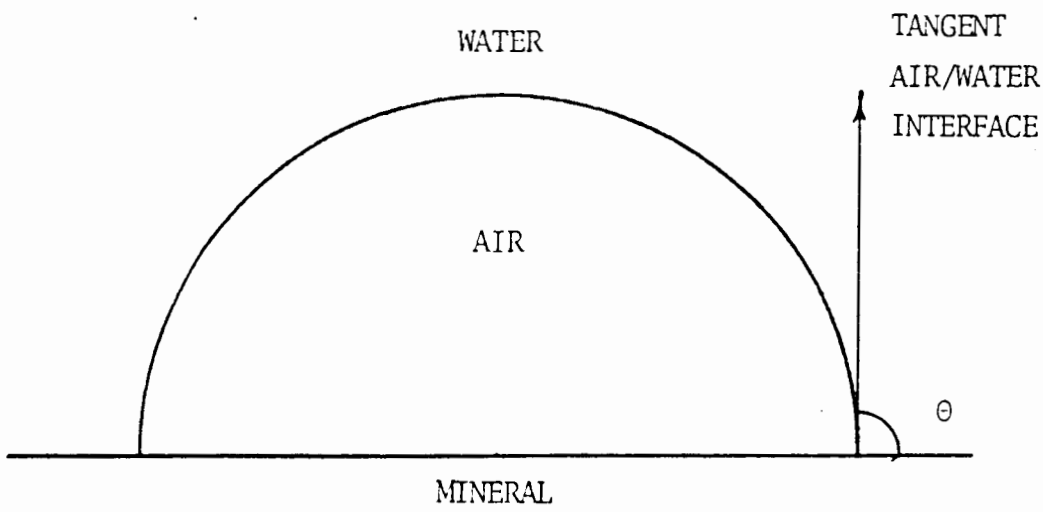

Figure 2.2 Three phase line of contact between air-water-mineral, showing the contact angle Θ.

2.2.3 Mathematical Models of Flotation

From the above discussion, it is obvious that mathematical flotation models which incorporate both a recovery and rate function can provide a much better tool for evaluating flotation results than models based on recovery alone.

Levenspiel (1972) points out that the "requirement for a good engineering model is that it be the closest representation of reality which can be treated without too many mathematical complexities."

Lynch *et al* (1981) have defined three categories of flotation models:

(a) Empirical models: these define a relationship between the measured responses e.g. grade and recovery, and the flotation variables e.g. reagent addition rates, mechanical factors.

(b) Kinetic models: these define the mass balance of elements in the flotation system as a function of time i.e. they contain rate and equilibrium recovery functions. Practically this is done by following the time-based recovery of metallurgical components.

(c) Probability models: these define successful flotation as the product of the probabilities of particle-air bubble collision and adhesion. They also contain a froth stability factor.

Dowling *et al* (1984) show that they reduce to kinetic models in the limiting case.

Dowling *et al* (1984) also identified and tested 13 kinetic flotation models for applicability to time-recovery profiles. They concluded that a first order flotation model with rectangular distribution of floatabilities exhibited the best fit of all the models

tested, and the confidence limits for the rate and equilibrium parameters were small. The mathematical form of the equation is given by

$$R = \phi \{1 - \frac{1}{kt} (1 - \exp(-kt))\} \quad (1)$$

where : R = recovery at time t
ϕ = "ultimate" or equilibrium recovery
k = rectangularly distributed rate constant.

This model was derived independently by Klimpel (1980) and Huber-Panu *et al* (1976) who considered the first order mass balance of a single flotation component from a monodisperse feed having a rectangular distribution of floatabilities. A rectangular distribution of values means that there is no central tendency and all values are equally likely. Meyer and Klimpel (1984) arrived at the same mathematical form of the equation by considering the mechanical rate of mass removal from the froth.

Figure 2.3 shows classical recovery-time profiles which are completely described by the kinetic model parameters ϕ and k for the different systems.

For similar concentrate grades, test 2 (Figure 2.3 (a)) is consistently superior to test 1 because for any residence time test 2 yields higher recoveries. For the same reasons test B is better than test A only when the residence time is less than T_{RE}. If the residence time is less than T_{RE} the system is "rate controlled" otherwise "equilibrium controlled."

The interpretation of the effects of process changes on the constants ϕ and k is relatively easy but in some cases it may be misleading because the value of k is related to the value of ϕ. This k value measures the rate with which the process approaches the equilibrium recovery value and is not directly associated with the initial slopes of recovery-time curves (Dunne, 1984).

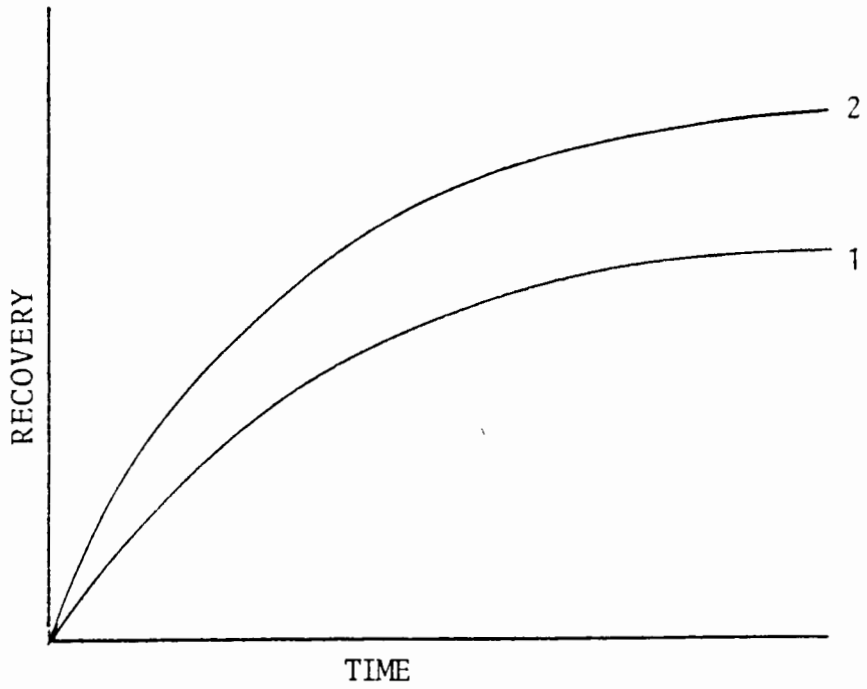

Figure 2.3 (a) : The effect of different kinetic model constants on the recovery-time profile

$\phi_1 < \phi_2$: $k_1 < k_2$

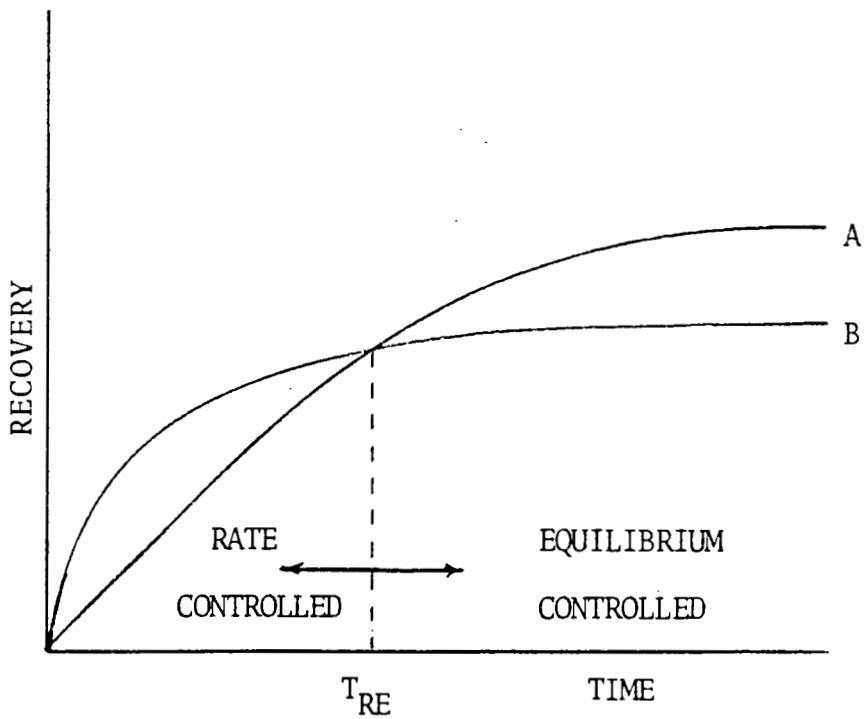

Figure 2.3 (b) : The effect of different kinetic model constants on the recovery-time profile

$\phi_A > \phi_B$: $k_A < k_B$

This is obvious since the derivative with respect to time of equation (1) is a function of both ϕ and k.

The model described by equation (1) can be fitted to recovery-time data for the different mineralogical species found in any flotation concentrate. Specifically in coal flotation it is possible to fully describe the recovery-time data of the dry, ash free coal and gangue, from which the recovery-ash profile can be calculated as follows:

$$Ac = \frac{Rg \quad x \quad 100}{Rg + Rc \ (100/f - 1)}$$

where Ac = cumulative ash in concentrate (%) at time t
Rg = recovery of gangue in concentrate at time t (%)
Rc = recovery of coal in concentrate at time t (%)
f = ash in the feed (%)

Rg and Rc may be calculated from the kinetic model once the parameters ϕ and k are known.

2.3 PARAMETERS AFFECTING COAL FLOTATION

Wark and Sutherland (1955) identified some 32 variables affecting flotation, of which 22 are under the control of the metallurgist. Some of these parameters are listed in Table 2.3.

A consequence of the complexity of flotation is that most of the research has been conducted empirically or by investigating the effects of a single parameter. However the interactions between the variables may be more important than the variables themselves.

Klimpel (1984) states that flotation is an interactive engineering system involving many complex chemical and physical interactions. The flotation response is related to chemical, equipment and operational factors (Figure 2.4).

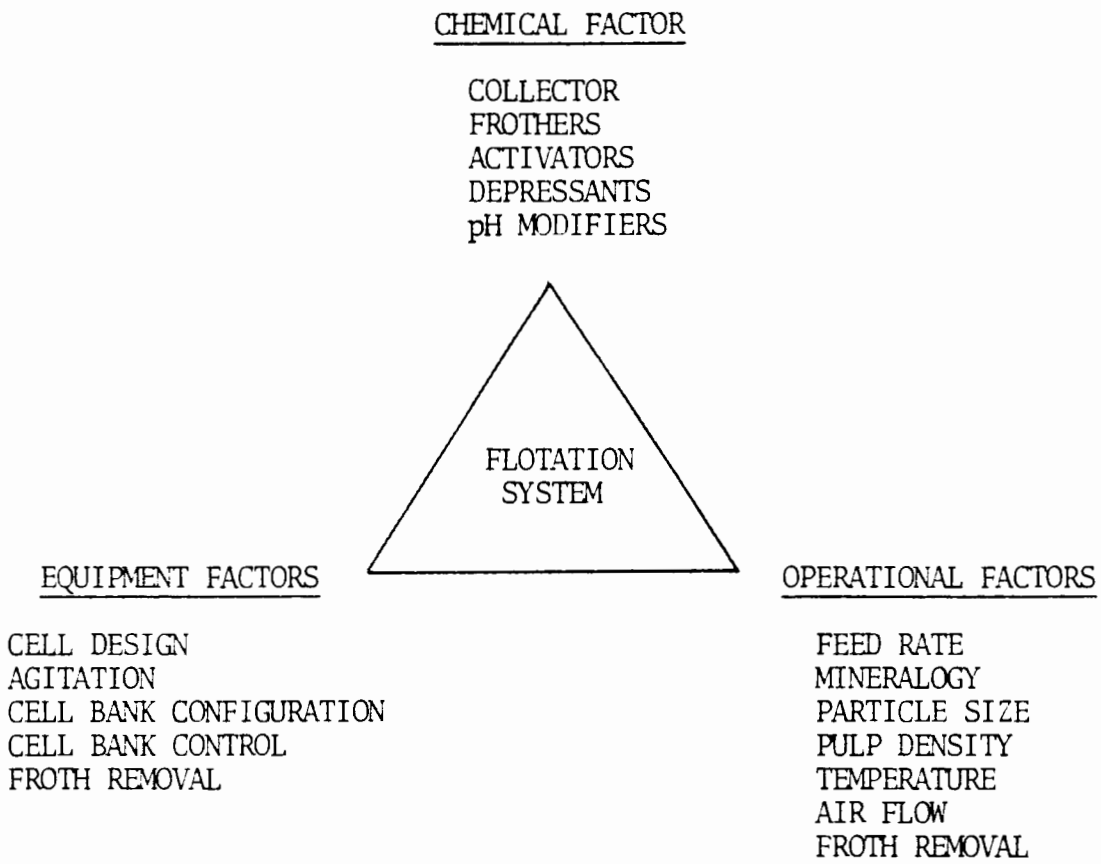

Figure 2.4 Flotation as an Interactive Engineering System (Klimpel, 1984)

TABLE 2.3

Some of the Parameters Controlling the Froth Flotation Process (Wark and Sutherland, 1955). Those below the line may be controlled by the Metallurgist.

RAW MATERIAL	GRINDING AND CLASSIFICATION	CONDITIONING	FLOTATION
1. Nature of Mineral 2. Associated Gangue 3. Soluble Constituents of the Ore 4. Degree of Oxidation 5. Mineral Oxidation 6. Composition of Water	1. Relative Hardness of Mineral to Gangue 2. Milling Time 3. Mineral Association		
1. Soluble Impurities in the Water 2. pH of the Water	1. Type of Machine 2. Oxidation during Milling 3. Chemical Feed during Milling 4. Pulp Density 5. Mill Type 6. Recirculation Rate	1. Chemicals added - Type, Quantity, Order and Form 2. Temperature 3. Mean Residence Time	1. Chemicals added - Type, Quantity, Order and Form 2. Pulp Density 3. pH value 4. Type of Machine 5. Residence Time 6. Aeration Rate 7. Impeller Speed 8. Froth Height

The highly complex heterogeneous nature of coal and its tendency to undergo substantial changes on exposure to various environments further complicates the flotation process.

The more important variables controlling flotation in general and coal flotation in particular are discussed below. Only the effects of individual, primary variables are considered at this stage, allowing for the fact that various degrees of interaction exist between these parameters.

2.3.1 Effect of Petrographic Composition and Rank

As was shown earlier (Section 2.1), it is possible to characterise a coal by determining its petrographic composition and rank. It was also stated that the chemical reactivity of a coal is determined by its rank and maceral composition. Knowing that floatability is dependent on the surface characteristics, it is reasonable to expect that coal petrography and rank will determine the floatability.

Researchers have established that ease of floatability increases with

(a) increasing rank, reaching a maximum for coals exhibiting medium to high rank; and

(b) coals containing a high proportion of vitrinite and exinite.

Brown and Smith (1962) in reviewing coal flotation noted that in changing from lignite to bituminous coals the carbon content increases and the inherent moisture content decreases while hydrophobicity increases. Rosenbaum *et al* (1982) postulated that this effect may be indirect and the same factors which influence inherent moisture are also responsible for the floatability. The point of maximum hydrophobicity (carbon content of 81 per cent) is approximately equivalent to the point of minimum

porosity, moisture and hardness (Falcon, 1978 (b)). Beyond this point, as rank increases (corresponding to anthracite coals) hydrophobicity decreases as the atomic structure becomes denser and more aromatic in character.

It is possible to explain the difference in the floatabilities of the different macerals similarly. Brown and Smith (1962) links this with the fraction of total carbon which increases in the order exinite, vitrinite, inertinite. He proposed that the ordered graphite structure of inertinite negates its natural hydrophobicity. In practice it has been found that the order of increasing floatability is inertinite, exinite, vitrinite (Falcon, 1978 (a)).

Taggart *et al* (1939) implied that the difference in floatability between anthracite and bituminous coals was caused by a variation of the carbon hydrogen ratio; but this does not apply to other coals.

Sun (1954 (a)) proposed a 'surface components hypothesis' to explain the different floatabilities of coal, carbon and hydrocarbon materials. The gist of the hypothesis is that coals consist of floatable and non-floatable components. The floatability is governed by the proportions of these two groups of material. He based an empirical relationship (the floatability index) on ultimate and proximate analysis. Aplan (1976) pointed out two defects of Sun's work:

(i) Since the analysis is based on bulk sample properties, surface oxidation might completely destroy the estimate of floatability.

(ii) Complete chemical analyses are rarely available.

The floatability index would probably not apply to South African coals because of their very different and diverse characteristics.

In later work, Campbell and Sun (1970 (a), (b)) showed that the hydrophobicities of the different lithotypes vary and are dependent on pH. They suggested that the lithotypes were therefore potentially separable by froth flotation.

In experiments with British coals, Burdon (1962) showed that the maceral content of flotation concentrates changed with time because the rates of flotation of the different macerals were different. He showed that vitrinite floated faster than semi-fusinite, although at finer particle sizes the flotation rates were nearly equal.

2.3.2 Effect of Oxidation

A number of investigators (Firth and Nicol 1981, Gray *et al* 1976, Nimerick and Scott 1980, Sun 1954 (b), Wen and Sun 1981) have shown that oxidised coals are less amenable to flotation than fresh coal. Wen and Sun (1981) discovered that when a few percent of oxidised coal is added to unoxidised coal, the overall flotation efficiency is drastically reduced.

According to Sun (1954 (b)) bituminous and lower rank coals undergo atmospheric oxidation during mining and storage, the degree of oxidation increasing with temperature and time. Grey *et al* (1976) found that the ease of oxidation decreases with increase in rank, and (because of the different reactivities of the macerals) that the ease of oxidation also decreases in the order vitrinite, exinite, inertinite.

Sun (1954 (b)) proposed that the oxidation of coal occurs in three stages

(i) Superficial oxidation characterised by the formation of coal-oxygen complexes which exhibit acidic properties.

(ii) Transformation of organic components into hydrocarboxylic acids, generally termed humic acids. (This might account for oxidised coals being alkali soluble).

(iii) Degradation of humic acids to water soluble acids.

Sun deduced that the water soluble humic acids are chiefly to blame for the low floatability of oxidised coal because the hydroxyl and carboxyl groups are hydrophilic. These water soluble products are also mixtures of weak collectors and coal depressants. Firth and Nicol (1981) have recently supported the proposal that humic acids might be responsible for the poor floatability of oxidised coals.

Nimerick and Scott (1981) showed that the poor flotation efficiency of oxidised coals can be overcome by using cationic compounds (especially amines) as collectors or as activators prior to flotation with the usual oily collector. Gray *et al* (1976) found that the floatability of oxidised coals improved when using pine oil as opposed to an aliphatic alcohol as a frother.

2.3.3 Effect of Particle Size Distribution

As noted above, the minerals associated with South African coals are finely disseminated and hence very fine crushing is needed to provide adequate liberation. In addition, because of the high proportions of clay in South African coals (60 to 70 per cent), the presence of slimes (non-coal ultrafines) might largely determine the efficiency of flotation.

Firth *et al* (1978) stated that while difference in rank, petrographic composition, degree of oxidation and the presence of slime coatings influence coal flotation, the size distribution of the feed may override these

factors. They also showed that the recovery of coal was relatively insensitive to the mean particle size but was markedly dependent on the degree of polydispersity of the feed.

A number of investigators have shown that in polydisperse pulps, highest flotation efficiencies are achieved by particles in the intermediate size range. (Blagov *et al* 1982, Burdon 1962, Eveson *et al* 1957, Fuerstenau 1980, Miller *et al* 1967, and Trahar 1981). Bennett, Chapman and Dell (1962) have proposed that each size fraction appears to have its own individual flotation rate whether floated as an individual size fraction or in a polydisperse feed.

Trahar (1981) found that the flotation rate of ultrafines is low relative to that of other sizes, primarily as a result of the decreased probability of bubble-particle collision as particle size is reduced. He suggested that this might be caused by hydrodynamic forces controlling the approach of small particles to bubbles as opposed to the inertial forces associated with the larger particle-bubble collisions.

The flotation recovery of ultrafine particles might be determined by entrainment, which would set a lower limit to the size at which conventional flotation is efficient. Trahar (1981) also found that recovery by entrainment is linked to the recovery of water in the concentrate. Lynch *et al* (1981) proposed that the presence of ultrafines increases froth stability by retarding film drainage, resulting in higher recoveries but lower grades being attained.

Other problems caused by ultrafines and slimes are excessive consumption of reagents, overfrothing, concentrate contamination and a reduction in the recovery of coal. Jowett *et al* (1956) and Mishra (1978) proposed that these slimes are produced by the degradation and

disintegration of particles of clay and shale in the feed. The slime particles are colloidal in size and character, and are able to coat coal particles thus rendering the coal surfaces hydrophilic.

Dunne (1982) reviewed the poor recovery of coarse particles in flotation, and suggested that:

(a) The collector coverage of the particle surface might be insufficient;

(b) The coarse particles might not be kept in suspension because of too low an impeller speed (on the other hand, higher impeller speeds might cause excessive turbulence resulting in increased entrainment and pulp carryover into the concentrate); or

(c) The air bubbles might not be large enough to provide sufficient buoyancy to lift the particles out of the pulp phase.

In summary, it has been shown that the degree of polydispersity of the feed plays an important part in determining the flotation efficiency. In conclusion, it may be noted that a number of ways of overcoming this problem have been suggested:

(a) separate conditioning of individual particle size classes

(b) stagewise addition of collector

(c) use of a selective, chemisorbing collector.

2.3.4 Effect of Reagents

Historically, because of low coal prices and the relatively high costs of flotation compared to alternative beneficiation processes, reagent costs have had to be minimised. Also, because of the easily floatable nature of overseas coals, flotation has been a relatively

efficient process. This has resulted in limited research being carried out in this area. Unlike mineral flotation, there has been no economic incentive to develop selective collectors, depressants, activators or pH modifiers. Generally, very similar reagent systems have been used without any regard for the different nature of the coals being treated.

Horsfall (1976) highlighted the need for developing selective collectors for South African coals. Collectors are the single most important parameter controlling the effectiveness of flotation, although, according to Wark and Sutherland (1955), the frother type and dosage are probably more important.

2.3.4.1 Collectors. Ideally collectors adsorb preferentially onto the valuable mineral surface imparting or increasing the natural hydrophobicity of the surface thereby improving the probability of successful bubble/particle collisions. Collectors are the single most important factors determining the efficiency and selectivity of froth flotation.

Collectors may be subdivided on the basis of their mode of absorption onto the mineral surface i.e. physisorption or chemisorption. Chemisorbing collectors are more selective and are characterised by a higher absorption energy than physisorbing collectors. This is shown in sulphide mineral flotation in which specific sulphides can be selectively and sequentially floated.

To date the collectors used in coal flotation have been the oily by-products of the chemical and especially the petrochemical industries such as the bottoms products from alcohol distillations, as well as paraffin, kerosene and diesel oil. These oily collectors are believed to be physisorbing. Lynch *et al* (1981) and Nimerick and Scott

(1980) have proposed that oily collectors do not induce hydrophobicity of the coal surface but merely extend this property.

It is believed that oily collectors aid flotation by increasing the particle/bubble collision efficiency and/or the rupture distance of the liquid bridge by imparting a greater elasticity to it.

In aqueous media oily collectors are characterised by a high degree of dispersion. The reagents form aggregations of various sizes from large micelles and emulsion drops down to molecular clusters (Naumov, 1978). Klassen *et al* (1958) postulated an optimum dispersion of the reagents in droplets of 4 to 6 microns. They arrived at this by considering an average pore size of 4 to 6 microns, thus limiting absorption of the collector in the pores.

The degree of dispersion of the collector will affect concentrate recovery, because the greater the dispersion the greater is the probability of collector droplet-particle collision. The dispersion of the reagents will depend on the intensity of agitation and the conditioning time as well as the presence of emulsifying agents.

Naumov (1978) proposed that the use of chemical additives is the most efficient method of dispersing the collector. He concluded that the effect of surfactants is not related to their collecting or foaming properties but to the change in the degree of dispersion of the collector. For this reason the frother is usually added to the oily collector prior to conditioning the pulp.

Other mechanical methods of increasing the dispersion of the apolar collectors exist, utilising water jets, aerosol sprays or venturi mixers.

Sun (1954 (b)) found that increased addition of oily collectors resulted in the depression of oxidised coals.

Nimerick and Scott (1980) showed that oxidised coals can be floated by using cationic compounds (especially amines) either as collectors or activators. If a cationic compound is used as an activator, the coal can be effectively floated using an oily collector.

Previous workers have tested a variety of different chemical homologues as coal collectors. The following trends have been noted:

(a) Carboxylic acids have a low selectivity, apparently because of their ability of reacting chemically with some gangue minerals. (Sorokin *et al* 1979). Although C_6 to C_{10} fatty acids have been shown to be efficient coal collectors (Klassen *et al* 1958), they are less efficient than their corresponding alcohols.

(b) Esters are better collectors than the alcohol and acids from which they are produced (Plaksin and Klassen, 1962).

(c) Aromatic compounds are not as effective collectors as the corresponding aliphatic compounds (Klassen *et al* 1958).

(d) Unsaturated organics irrespective of their structure and composition have poorer flotation properties than the corresponding saturated compounds of the same class (Plaksin and Klassen, 1962).

(e) Branched hydrocarbons have a higher flotation activity than the corresponding straight chain compounds.

The optimum collector addition rates have been found to depend on pulp density, particle size distribution, presence of slimes, impeller speed and degree of dispersion, but most especially on the characteristics of the coal to be floated.

2.3.4.2 Frothers. The function of the frother is to create a stable froth that is capable of carrying the hydrophobic particles. The frother absorbs at the air-water interface thereby lowering the surface tension. The secondary effects of reducing surface tension are to increase the froth elasticity and hence froth stability as well as to provide a uniform and reduced bubble size. The smaller bubbles mean more efficient distribution and utilisation of air as well as a faster rate of flotation. This higher flotation rate is usually accompanied by lower grades because of increased entrainment. The smaller bubble size probably also results in more entrapment.

Bennett, Chapman and Dell (1958) showed that at constant conditions a low frother addition results in a wide bubble size distribution. Higher frother concentrations reduce the bubble size and the distribution of sizes. Burdon (1962) proposed an optimum bubble size of 1 to 2 mm. Smaller bubbles remain in suspension, while larger bubbles have a limited dispersion and greater rising velocities. The latter results in a decreased efficiency of particle collection.

Frothers are generally heteropolar compounds which when mixed with the oily collector prior to conditioning, have the secondary function of aiding the collector dispersion. They may be classified into three groups on the basis of their chemical structure.

(a) Alcohol frothers - these are sub-divided into 3 groups, i.e. aliphatic, cyclic and aromatic alcohols. Examples of these are methyl isobutyl carbinol (MIBC), pine oil and cresylic acid respectively.

The froths created by the aliphatics are larger in bubble size and less compact than those produced by pine oil and cresylic acid. The disadvantage of the

latter two frothers is that at excessive dosages they tend to flatten the froth and cause effervescence.

In many cases the froth produced by alcohols is not as long lasting as those created by other frothers and so they require staged additions to maintain an effective concentration (Booth *et al* 1962). This is an advantage in differential flotation circuits.

(b) Alkoxy type frothers - these are used extensively in sulphide flotation producing froths that resemble those of pine oil, except that excess addition has little tendency to flatten the froth, e.g. 1, 1, 2 Triethoxybutane (TEB)

(c) Polyglycol type frothers - these are non-ionic surface active agents that produce compact, long lasting froths. They do not flatten and kill froths when added in large amounts. They have the added advantage in that they are generally water soluble and hence easily dispersed.

In general it has been found that alcohol frothers are the most efficient for coal flotation. MIBC and cresylic acid are the most widely used. The loose froth structure created by these frothers might be conducive to gangue elimination (Griffiths, 1976).

Eveson *et al* (1957) have shown an optimal MIBC concentration to be in the range of 15 to 20 mg/ℓ. They use an efficiency index which is a function of perfect separation (determined by float/sinks analysis). Higher concentrations result in a decreased efficiency.

Dunne (1984) proposed that the froth characteristics

determine the efficiency of flotation. This will be mainly controlled by the frother type and dosage, although Booth *et al* (1962) showed that the froth characteristics also depend on

(a) the type of collector

(b) the type and degree of polydispersivity of the mineral

(c) the mechanical factors such as aeration rate and impeller speed

(d) the presence of other organic and inorganic species in the pulp

(e) the pH value of the pulp.

This supports the findings of other investigators (Burdon 1962, Eveson *et al* 1957) who report that frother type and concentration interact with other flotation variables to determine bubble size.

2.3.5 Effect of Pulp Density

Eveson *et al* (1957) proposed that mineral flotation in general requires a pulp density such that each particle may move more freely with respect to the others. Otherwise undesirable material might be recovered as a result of mechanical entrapment and entrainment in the froth and associated liquid. This is especially important when considering the nature of coal flotation where the bulk of the feed constitutes the valuable material.

Dell (1968) found that lower pulp densities resulted in lower recoveries and ash contents, which increased when the pulp density was increased. Burdon (1962) stated that in the presence of slimes, concentrate grades decreased as pulp density increased. However when the feed was deslimed concentrate grades were maintained at the higher pulp densities.

Investigators report optimum pulp densities in the range of 80 to 150 g/ℓ (8 to 15 per cent solids).

2.3.6. Effect of Aeration Rate

According to Dunne (1982) the effect of increasing the aeration rate is twofold

(i) Increases the number of bubbles, resulting in higher flotation rates

(ii) Increases the size of the bubbles, resulting in higher buoyancies and therefore larger particles being floated.

The disadvantage of a higher aeration rate is that it decreases the agitation efficiency because an envelope of fine bubbles surround the impeller. At sufficiently high aeration rates the impeller is no longer able to shear the air into bubbles and 'boiling' occurs. This can be overcome if the impeller speed is increased, but again it will result in decreased concentrate grades.

Aplan (1976) proposed that the optimum air flow can be obtained from the following correlation:

$$NQ = \frac{Q}{ND^3}$$ where Q = air flow (cfm)
N = impeller speed (rpm)
D = impeller diameter (feet)
with the air flow number $N_Q = 3 \times 10^{-2}$

This corresponds to the optimum value quoted in Aplan (1976) for mineral ores.

2.3.7 Effect of Impeller Speed

A number of investigators (Aplan 1976, Burdon 1962, Dunne 1982, and Sun and Zimmerman 1950) have stated that impeller speed is important in determining particle suspension, collector and air dispersion, and the probability of bubble/particle contact.

Sun and Zimmerman (1950) showed that the minimum agitation for successful flotation increases with increase in particle size and mineral density in that it controls the particle suspension.

2.3.8 Effect of Temperature

Aplan (1976) has stated that over the average ambient temperature range of 3 to 50°C there is very little difference in coal flotation. In contrast, O'Connor *et al* (1983) found that temperature affects the rate of pyrite flotation but does not influence the equilibrium recovery. They proposed that this might be caused by temperature effects on pH, pulp viscosity, bubble formation and froth stability.

CHAPTER 3

EXPERIMENTAL EQUIPMENT, METHOD AND PROCEDURES

3.1 INTRODUCTION

The experimental work involved designing and carrying out a series of batch flotation tests to investigate the effects of various flotation parameters on four different coal samples.

This chapter describes the petrographic composition of the washed and raw coal samples, the experimental equipment and procedures used in the investigation as well as the analytical and computational methods employed in analysing the data.

3.2 CHOICE OF COAL SAMPLES

The four coals chosen for this investigation were from Grootegeluk, Greenside (No 2 Seam), Nonsana and Longridge Collieries. Samples of run-of-mine coal and washed product were obtained.

As was shown above (Section 2.1) a coal is characterised not only by the proximate analysis but also by its petrography, i.e. organic (maceral) and inorganic (mineral) composition, as well as its rank (degree of metamorphosis). The coals selected for this investigation were chosen on the basis of a petrographic analysis performed by R Falcon Research Laboratory (Pty) Limited. The following sections are a summary of the petrographic survey compiled by Dr R Falcon (1983). The analytical procedures and terminology applied to the description of these samples are those proposed by the International Committee for Coal Petrology (1963 and 1971). Additional

terminology concerning the semi-reactive macerals is that proposed by ISCOR (1982). Deviations in the quantitative analyses were maintained within 3 per cent limits.

3.2.1 Washed Coals

The maceral and mineral compositions of the washed coals are presented in Tables 3.1 and 3.2 respectively. Figure 3.1 shows the rank of each coal sample relative to its total reactive macerals and vitrinite composition. Table 3.3 shows the proximate analysis of the four washed coal samples. This is followed by a summary of the microscopic description of the coals.

(a) Grootegeluk

This coal sample was a High Volatile Bituminous B rank coking coal (Point 1 in Figure 3.1). It was relatively clean, with 42 per cent clean coal particles, the remaining minerals being in the form of very small, finely dispersed clays.

It had an extremely high vitrinite (86,3 per cent) and low exinite (2,5 per cent) and inertinite (11,2 per cent) content, the vitrinite existing in thick homogeneous bands. The vitrinite contained a high proportion of pseudovitrinite which may affect the properties generally associated with vitrinite.

The coal showed some degree of weathering and oxidation.

(b) Greenside

This coal sample was a High Volatile Bituminous B rank blend coking coal (Point 2 in Figure 3.1). It had a lower clean coal content (26 per cent) than the Grootegeluk coal, with a higher clay mineral content (66,5 per cent) distributed in the form of very small finely dispersed clay aggregates. It also contained syngenetic intergrowths

TABLE 3.1

Maceral Analysis of the Washed Coal Samples

Note: The total inertinite fractions includes the Reactive Semi Fusinite Fraction

Coal	Vitrinite	Exinite	Reactive Semi Fusinite	Total Inertinite	Total Reactive
Grootegeluk	86,3	2,5	1,2	11,2	90,0
Greenside	47,2	7,1	11,0	45,7	65,2
Nonsana	43,1	0,0	15,8	56,9	58,9
Longridge	63,3	0,0	12,6	36,7	75,9

TABLE 3.2

Mineral Analysis of the Washed Coal Samples

Coal	Quartz	Clay	Pyrite	Carbonates	Clean Coal
Grootegeluk	0,0	56,0	4,0	0,0	41,0
Greenside	0,0	66,5	4,5	3,0	26,0
Nonsana	0,0	69,0	26,0	0,0	5,0
Longridge	0,0	59,0	31,0	6,0	4,0

TABLE 3.3

Proximate Analysis of the Four Washed Coal Samples

Coal	Moisture (%)	Ash (%)	Volatiles (%)	Fixed Carbon (%)
Grootegeluk	3,1	12,6	34,1	49,9
Greenside	2,2	7,9	33,4	56,5
Nonsana	2,1	15,2	9,6	73,1
Longridge	2,0	10,7	9,4	77,9

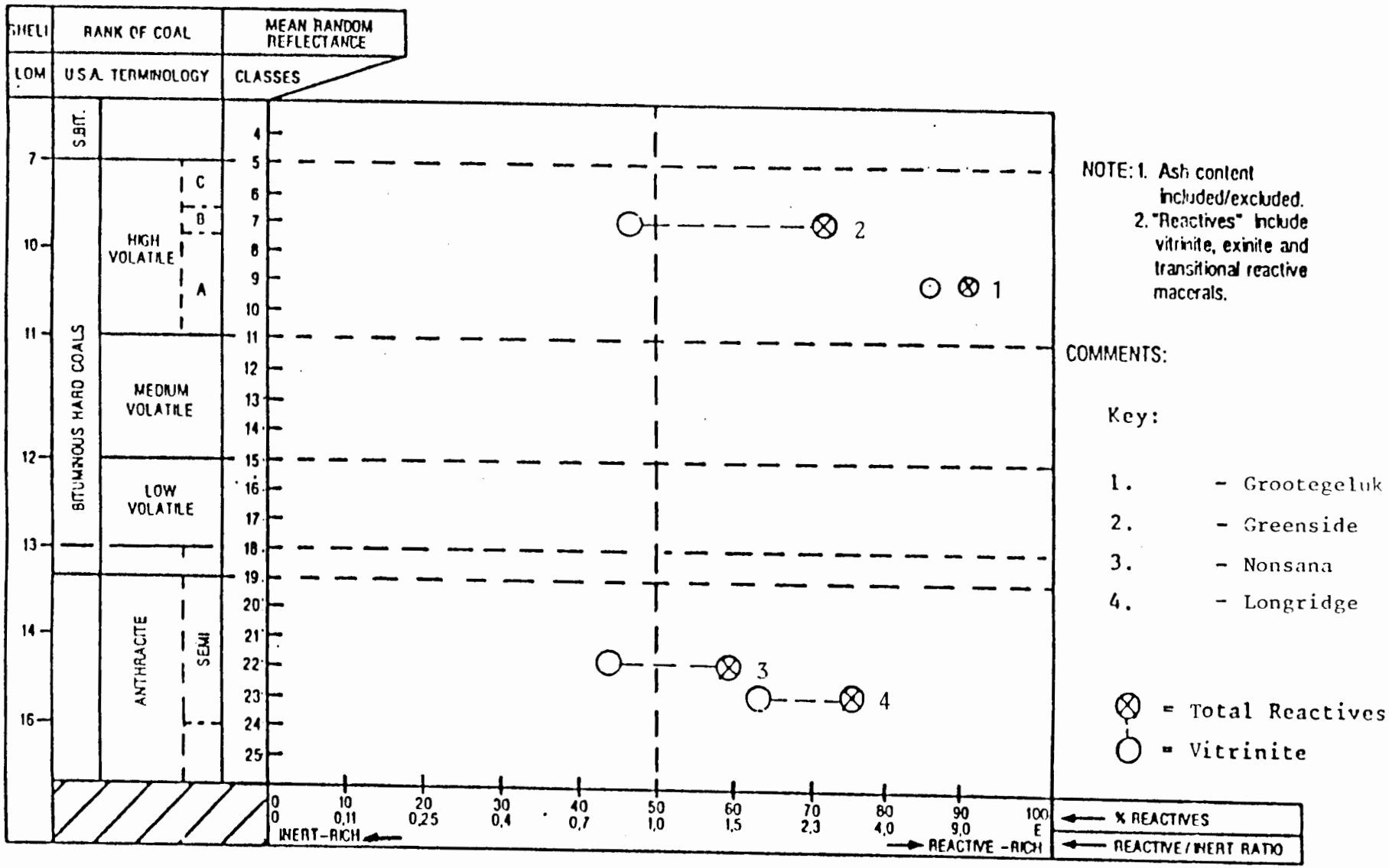

Figure 3.1 Rank analysis in relation to reactive/inert organic composition.

of carbonates.

It also had a much lower vitrinite content (47,2 per cent) than the Grootegeluk coal sample but a higher exinite (7,1 per cent), inertinite (45,7 per cent) and reactive semi-fusinite (11,0 per cent) content.

(c) Nonsana

This was a semi-anthracite coal (Point 3 in Figure 3.1) characterised by a very low clean coal (5 per cent) content. The majority of the minerals consisted of intimately dispersed clay aggregates (69 per cent) and small, finely distributed syngenetic pyrite spherules. It contained roughly equal amounts of reactive vitrinite and inert inertinite, with a high proportion of reactive semi-fusinite (15,8 per cent) and no exinite.

The coal had probably been subjected to heat effects which might have affected the preparation characteristics.

(d) Longridge

This coal sample was a semi-anthracite coal exhibiting a higher rank than the Nonsana coal sample (Point 4 in Figure 3.1) and with a very low clean coal (4 per cent) content. It was similar to the Nonsana coal in that it contained a high clay (59,0 per cent) and pyrite content (31 per cent). These existed as finely distributed clays and syngenetic pyrite spherules.

It had also been subjected to heat effects which may have influenced the preparation characteristics.

3.2.2 Raw Coals

The maceral and mineral compositions of the raw coals are presented in Tables 3.4 and 3.5 respectively. Figure 3.1 shows the rank of each coal sample relative to its total reactive macerals and vitrinite compositions.

TABLE 3.4

Maceral Analysis of the Raw Coal Samples

Note: The total inertinite fractions includes the Reactive Semi Fusinite Fraction

Coal	Vitrinite	Exinite	Reactive Semi Fusinite	Total Inertinite	Total Reactive
Grootegeluk	65,3	8,2	4,0	26,5	77,5
Greenside	24,5	2,0	15,0	73,5	57,7
Nonsana	48,0	0,0	5,5	52,0	54,3
Longridge	49,5	0,0	12,5	50,5	62,7

TABLE 3.5

Mineral Analysis of the Raw Coal Samples

Coal	Quartz	Clay	Pyrite	Carbonates	Clean Coal
Grootegeluk	0,0	87,0	9,0	3,0	1,0
Greenside	0,0	84,0	3,0	11,0	2,0
Nonsana	0,0	80,0	19,0	1,0	0,0
Longridge	0,0	55,0	35,0	10,0	0,0

TABLE 3.6

Proximate Analysis of the Four Raw Coal Samples

Coal	Moisture (%)	Ash (%)	Volatiles (%)	Fixed Carbon (%)
Grootegeluk	1,6	56,4	20,1	21,9
Greenside	2,3	20,1	14,1	53,5
Nonsana	1,9	29,2	7,6	61,3
Longridge	1,7	18,8	11,9	67,6

Table 3.6 shows the proximate analyses of the four raw coal samples.

The microscopic components of the coal may be described as follows:

(a) Grootegeluk

This coal sample was a High Volatile, Bituminous B rank coking coal (Figure 3.1), with a low clean coal content (1,0 per cent). Clays were the most abundant mineral (87 per cent) existing in both coarse and finely disseminated forms. It contained a high proportion of reactive macerals (77,5 per cent) of which the majority was vitrinite (65,3 per cent).

(b) Greenside

This was a High Volatile Bituminous B rank blend coking coal with a low clean coal content. Again clays accounted for 84 per cent of the mineral content with the majority existing as very small, finely distributed particles. It had a much lower vitrinite content than the Grootegeluk coal sample but a higher inertinite (73,5 per cent) and RSF (15 per cent) content.

(c) Nonsana

This was a semi-anthracite coal (Figure 3.1) containing no pure coal particles. The most abundant minerals were clays (80,0 per cent) which with pyrite (19 per cent) existed as very small finely disseminated particles. It contained more vitrinite (48 per cent) than the Greenside coal sample but less RSF (5,5 per cent).

(d) Longridge

This was semi-anthracite coal (Figure 3.1) but with a higher rank than the Nonsana coal sample, and no clean coal particles. The clay constituted 55 per cent of the total

minerals, which was lower than the other samples, but it contained a high proportion of finely disseminated pyrite (35 per cent). Its maceral composition was similar to the Nonsana sample although there was a higher proportion of RSF which resulted in a higher total reactive content.

3.3 EQUIPMENT

The washed and raw coal samples were obtained from the collieries and prepared by MINTEK, and divided into 1 kg representative samples in the particle size range of -10 + 6 mm. They were stored in a nitrogen atmosphere in an apparatus designed for this purpose and milled immediately prior to flotation to the required particle size. (This limited and controlled the effects of surface oxidation on the flotation efficiency). Conventional batch flotation tests were carried out using a modified Leeds cell. The concentrates and tailings from the batch flotation tests were dried and ashed.

3.3.1 Rod Mill

The mill used was a rod mill made of stainless steel to avoid the introduction of abraded iron and rust into the sample, which might affect the flotation of coal.

The critical speed of the mill was calculated from the Chemical Engineers Handbook (Perry and Chilton, 1973)

$$N_c = \frac{76,6}{D^{\frac{1}{2}}}$$

where N_c = critical speed (rpm)

D = diameter (feet)

For the mill diameter of 31,6 cm the formula gives a critical speed of 75,2 rpm. The mill was operated at 0,9 N_c (67 rpm) which is in the region of optimum efficiency.

3.3.2 Flotation Cell

The flotation cell used in the batch testwork was a modified version of the "Leeds" cell described and developed by Dell and Bunyard (1972). It was manufactured by Chas Cook and Sons, Birmingham. It has been shown (Dell and Bunyard, 1972; Liddel and Dunne, 1983) that highly reproducible results may be obtained using this cell. In its original form it features control of impeller speed, aeration rate, pulp level and froth removal.

The modifications made were as follows:

(a) The original cell utilised a stationary froth deflection plate, made of porous ceramic. Water was forced through this plate thus preventing the froth from sticking to it. However preliminary work showed that with the coal samples a high water flow rate was needed to prevent the froth sticking to the plate, which resulted in the cell overflowing.

The plate was therefore removed, and not used in any of the experiments.

Froth removal was accomplished by manual scraping using a scraper the width of the cell. A scraping technique was developed for removing the concentrate at set intervals.

(b) The diaphragm type level controller supplied with the cell was found to operate erratically, producing widely variable results. It was replaced by a constant head type of level controller, shown in Figure 3.2. The pulp level was controlled by adding dilution water to the cell.

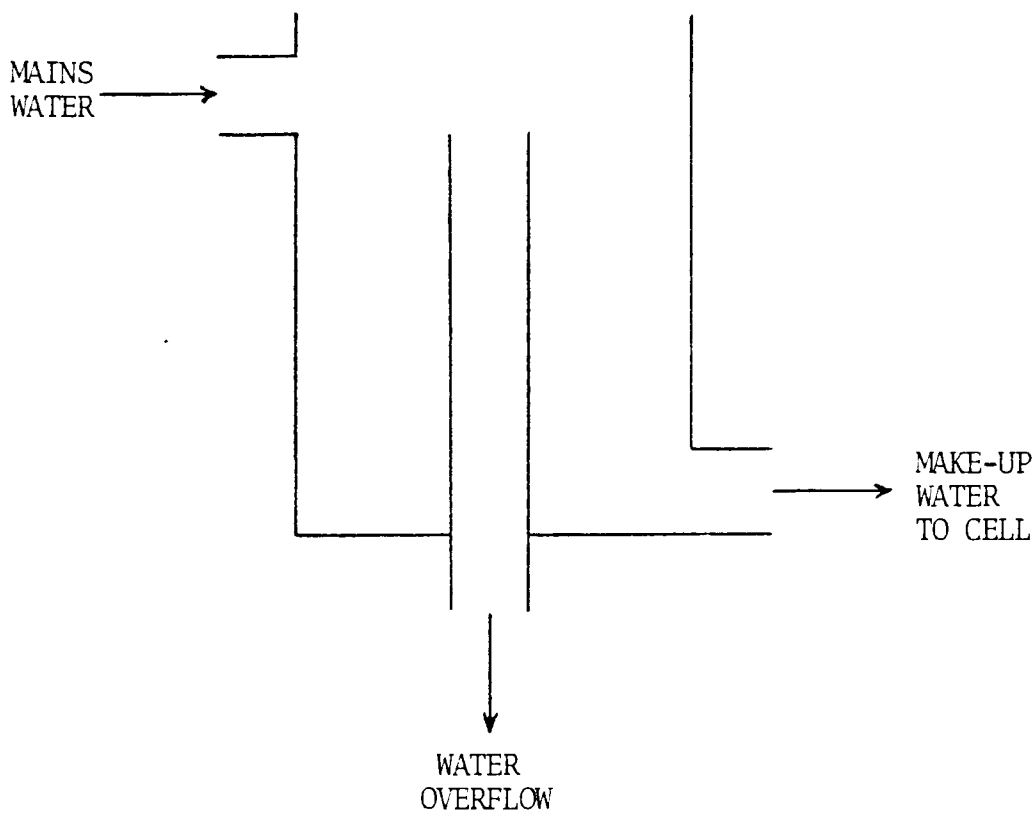

Figure 3.2 Constant head to supply make-up water to the batch flotation cell.

The final cell configuration is shown in Figure 3.3.

The air flow rate was controlled by a Flostat controller and a rotameter.

3.3.3 Anti-Oxidation Apparatus

The coarse coal samples (-10 + 6 mm) received from MINTEK were stored under a nitrogen atmosphere to limit and control the degree of surface oxidation.

The apparatus used for this purpose is shown in Figure 3.4. The vacuum pump was used to remove the air from the drum before nitrogen was introduced. This cycle was repeated five times thereby greatly lowering the oxygen content of the gases in the drum. Finally a positive gauge pressure of nitrogen was maintained to prevent air diffusing back into the drum.

3.4 EXPERIMENTAL PROCEDURE

3.4.1 Sample Preparation

The coal samples used were prepared by MINTEK and supplied in 1 kg bags in the particle size range of -10 + 6 mm. They were stored in a drum under a nitrogen atmosphere to limit surface oxidation.

Immediately prior to flotation a 1,0 kg subsample was removed from the drum and milled in a stainless steel mill to the required particle size distribution. The necessary milling time was determined from a milling curve of time versus percent passing the required sieve size.

The sample was homogenised on a sheet of paper before being divided on a rotary splitter. The individual samples were then weighed and/or recombined to provide

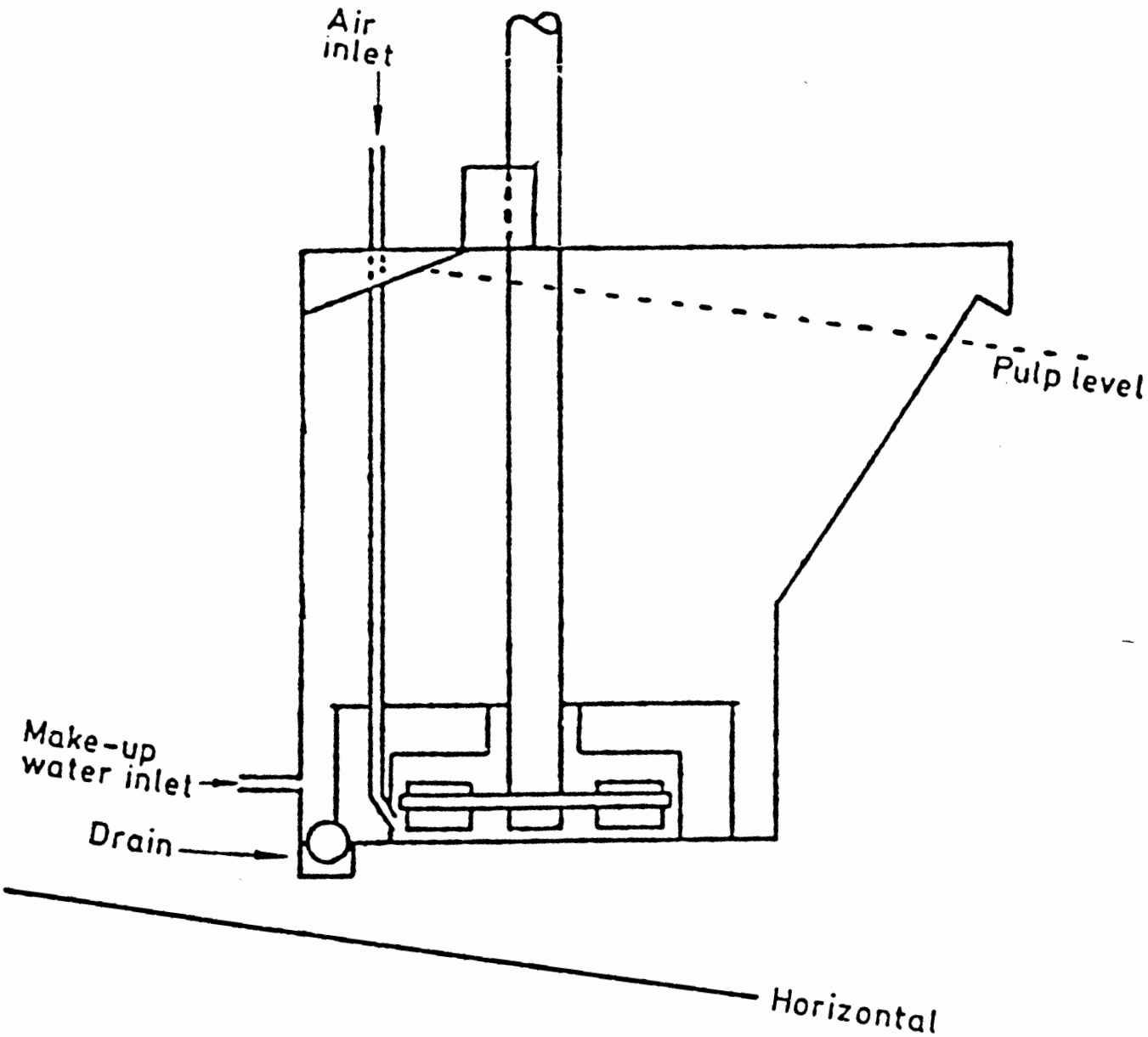

Figure 3.3 Schematic diagram of the Leed's flotation cell (After Dell and Bunyard 1972).

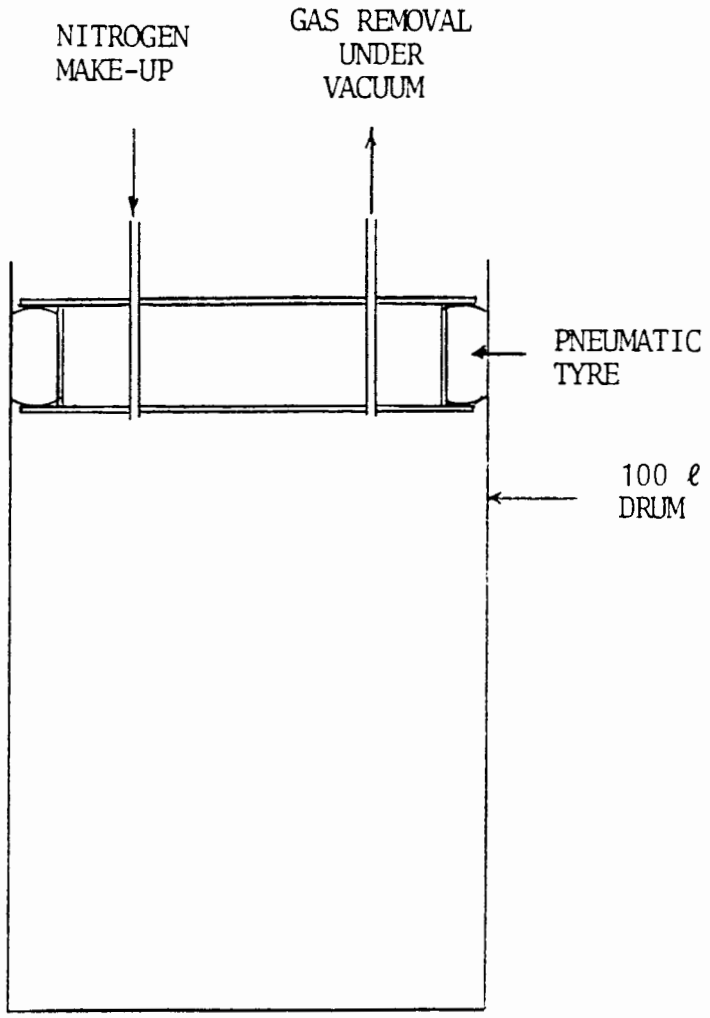

Figure 3.4 Apparatus used to store coal samples in a nitrogen atmosphere to prevent surface oxidation.

the required mass of sample. For a solids concentration of 5 per cent, 150 g of sample was used; for 8 per cent, 240 g; and for 10 per cent, 300 g.

These masses were not exactly equivalent to the relative solids concentrations, but the remaining coal, from a 1 kg sample, was sufficient for feed particle size and ash analyses.

This remaining sample was completely wetted and then filtered, thereby ensuring that it was subjected to a similar treatment as the coal used in the flotation tests.

3.4.2 Batch Flotation Tests

The sequence and timing of all the operations was rigidly adhered to so that all tests were carried out under conditions that were as similar as possible.

Approximately 1ℓ of water was introduced into the cell before the coal sample was added. This was then agitated at 1500 rpm for two minutes to ensure that the (hydrophobic) coal was completely wetted. Coal particles adhering to the walls of the cell and impeller were washed into the pulp.

After two minutes had elapsed, make-up water was introduced into the cell at a fixed flow rate. The level controller was set to give a pulp level of 1,5 cm below the lip of the cell. If the froth depth was increased a stagnant froth formed.

Ten minutes after the coal was placed in the cell, the collector (viz paraffin) was added using an automatic pippette. A conditioning time of four minutes was allowed before the frother was added, using a microsyringe. After a further minute of conditioning, aeration and concentrate collection commenced.

As is shown later (Section 4.2) separate addition of reagents resulted in better reproducibility and increased flotation efficiency. In industrial operations the frother and collector are usually combined because the frother is said to aid dispersion of the oily collector.

3.4.3 Computational Procedure

The cumulative recovery of dry ash free (d.a.f) coal and gangue in each concentrate as well as the cumulative ash content were calculated. Appendix A shows a sample calculation. The recovery-time data were fitted to the kinetic flotation model (Section 2.2.3) using a Nelder and Mead (1964) optimisation routine and the kinetic model constants k and ϕ were obtained.

After the start of the aeration a constant five second lag time was used to allow the froth to build up to the level of the overflow lip in the cell. Concentrates were then collected at fixed, discrete time intervals. If the start of aeration is detailed t = -5 sec then the concentrates were collected at

t = 0 to 15 s
t = 15 to 30 s
t = 30 to 60 s
t = 60 to 90 s
t = 90 to 150 s

This allowed batch test comparison by rate and equilibrium recovery data for the coal and gangue. It was found that the yield of coal after 2,5 minutes was negligible.

3.4.4 Analytical Procedure

The ash content of the flotation concentrates relative to the feed and tailing was used as the measure

of flotation performance. Samples were dried at 110° C for a minimum of fifteen hours to remove the inherent moisture from the coal. The feed and concentrates were then weighed and ashed using SABS Standard Method 926.

Initial analyses showed that the overall mass and ash balances agreed within 5 per cent limits. Thus it was decided that it was not necessary to analyse the tailings for ash content every time, because the calculated ash value agreed very closely to the analysed value. Periodic checks were performed to ensure that this accuracy was maintained.

CHAPTER 4

PRELIMINARY WORK

Before the factorial design experiments on the samples of the four washed and raw coals could be carried out, it was necessary to undertake a number of preliminary investigations. These were aimed at determining the effect of surface oxidation on freshly milled samples, and on setting the mode of reagent addition and the levels of the factors to be used in the detailed experiments.

This chapter outlines the results of this work.

4.1 EFFECT OF OXIDATION

The effect of surface oxidation on the freshly milled samples was investigated. This entailed carrying out four identical batch floats on each of the coal samples at set intervals over a four hour time period. This corresponded to the time normally required to carry out six consecutive tests. The flotation conditions used are detailed in Table 4.1.

The results obtained are summarised in Tables 4.2 (a) and (b) and shown in Figures 4.1 and 4.2 Obviously the effect of surface oxidation on the floatability of the coals was confounded with the experimental error. The results show that the cumulative effect of surface oxidation over a four hour period and experimental error on the equilibrium recovery and flotation rate constant is less than 10 per cent. The largest variations corresponded to the worst fit of the data to the kinetic model.

Another test of reproducibility is to check the mass

TABLE 4.1

Levels of the Flotation Variables used during the Reproducibility and Oxidation Tests

Aeration Rate	3 ℓ/min
Impeller Speed	1200 rpm
Paraffin Addition	1200 g/t
MIBC Addition	15 mg/ℓ
Particle Size	95 per cent minus 300 μm
Per cent Solids	5 per cent

TABLE 4.2 (a)

Kinetic Model Constants for Reproducibility Tests
Coal (d.a.f.)

	Equilibrium Recovery (%)	Flotation Rate (s^{-1}) Constant	(Σ Error^2) Max
Grootegeluk	46,8 $\pm$ 2,5	0,1191 $\pm$ 0,0063	3,5
Greenside	72,0 $\pm$ 1,8	0,1719 $\pm$ 0,0091	8,3
Nonsana	47,4 $\pm$ 2,2	0,0869 $\pm$ 0,0048	3,3
Longridge	83,3 $\pm$ 2,1	0,0805 $\pm$ 0,0034	4,9

TABLE 4.2 (b)

Kinetic Model Constants for Reproducibility Test
Gangue

	Equilibrium Recovery (%)	Flotation Rate (s^{-1}) Constant	(Σ Error^2) Max
Grootegeluk	29,2 $\pm$ 2,5	0,1162 $\pm$ 0,0072	7,8
Greenside	55,2 $\pm$ 2,1	0,1873 $\pm$ 0,0064	6,3
Nonsana	41,0 $\pm$ 2,4	0,0866 $\pm$ 0,0051	4,7
Longridge	48,1 $\pm$ 2,1	0,0804 $\pm$ 0,0078	6,2

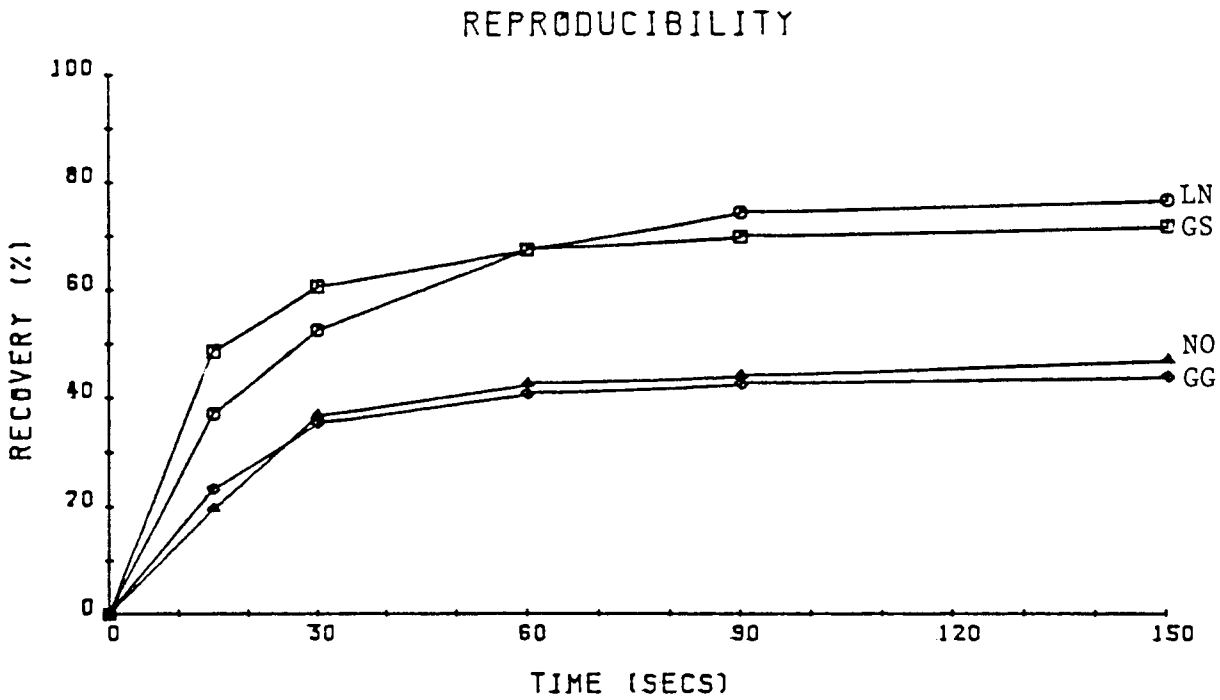

Figure 4.1 Recovery-Time profile showing the combined effects of experimental error and surface oxidation on the repeatability of the batch tests. (GG - Grootegeluk, GS - Greenside, NO - Nonsana, LN - Longridge).

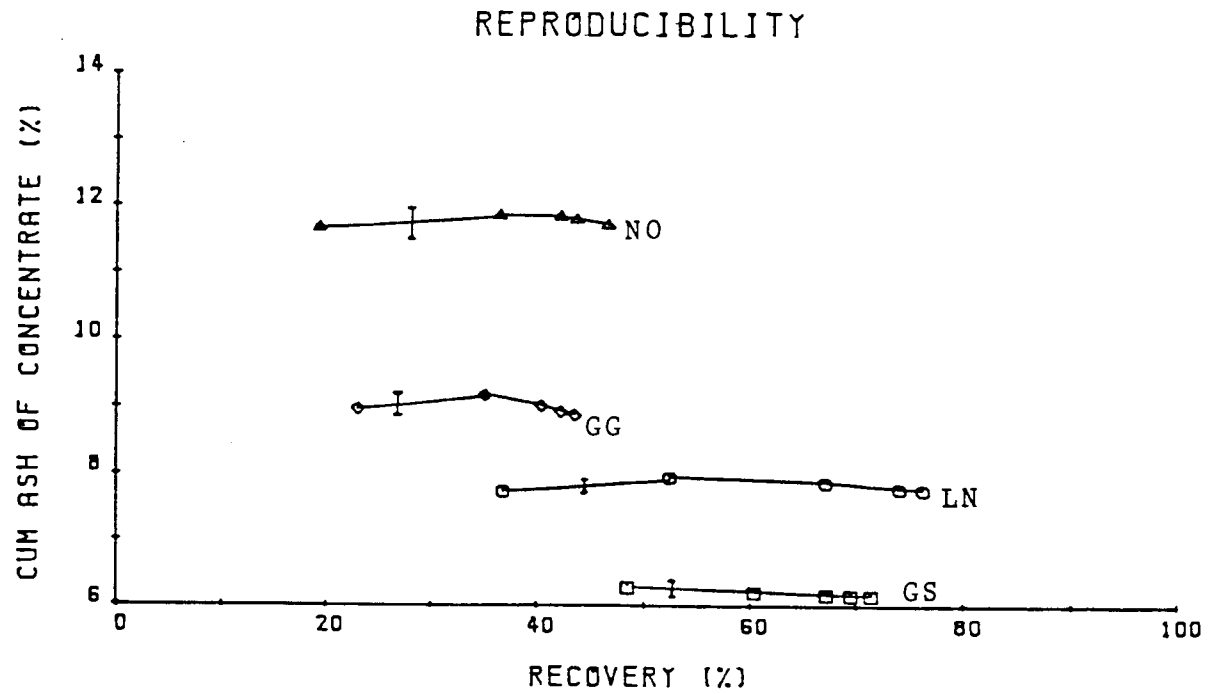

Figure 4.2 Recovery-Ash profile showing the combined effects of experimental error and surface oxidation on the repeatability of the batch tests. (GG - Grootegeluk, GS - Greenside, NO - Nonsana, LN - Longridge).

balance over the flotation cell. A variation in the mass balance of greater than 5 per cent is deemed unacceptable. In the tests (Tables 4.2 (a) and (b)) the calculated and experimental ash values of the concentrate and tailings agreed within this limit, when the moisture content of the feed was taken into account.

It was therefore assumed in the subsequent testwork that if the floats were carried out within four hours of milling, the effect of surface oxidation was negligible.

4.2 MODE OF PARAFFIN/MIBC ADDITION

It is general practice in coal flotation plants to combine the MIBC and Paraffin reagents prior to addition to the flotation cell, the reason being that MIBC aids the dispersion of the insoluble paraffin by lowering its surface tension. It was thought prudent before commencing with the factorial design experiments to investigate this mode of reagent addition.

Batch tests were performed on the Greenside No 2 seam coal sample under the following conditions:

Agitation Rate	1200 rpm
MIBC Addition	10 mg/ℓ
Paraffin Addition	500 g/t
Particle Size	95 per cent minus 500 microns

The MIBC and paraffin were mixed and placed in an ultrasonic bath for 30 seconds to ensure maximum dispersion of the paraffin. This was then added to pulps of 5 per cent and 15 per cent solids concentration and conditioned for five minutes prior to flotation.

The results of the tests are shown in Table 4.3. It is apparent that conditioning at a higher pulp density increases both the flotation rate and equilibrium recovery

TABLE 4.3

Showing the effects on the Klimpel Model Constants of varying modes of reagent addition and conditioning

	MIXED REAGENTS		SEPARATE ADDITION OF REAGENTS 5% SOLIDS
	CONDITIONING AT 5% SOLIDS	CONDITIONING AT 15% SOLIDS	
Equilibrium Recovery (Coal) (%)	73,2 ± 3,2	85,3 ± 4,3	83,2 ± 2,6
Equilibrium Recovery (Ash) (%)	56,4 ± 2,8	68,3 ± 3,7	67,1 ± 2,4
Rate Constant (Coal) (s^{-1})	0,155 ± 0,015	0,169 ± 0,013	0,158 ± 0,007
Rate Constant (Ash) (s^{-1})	0,184 ± 0,016	0,177 ± 0,015	0,172 ± 0,007

for the coal and ash. This is thought to be due to the increased probability of particle-reagent contact at the higher pulp density.

In a further test the MIBC and paraffin were added separately to a 5 per cent solids pulp. The paraffin was added first and conditioned for a further minute prior to aeration. The results also shown in Table 4.3 indicate a much better performance in terms of recovery and flotation rate than the addition of mixed reagents to a pulp containing 5 per cent solids. Separate reagent addition to a 5 per cent solids pulp resulted in slightly lower recoveries and flotation rate constant of coal and ash than the addition of mixed reagents for conditioning at 15 per cent solids pulp. However it is doubtful whether this difference is statistically significant.

The improved dispersion obtained when adding the reagents separately might be the result of the mode of addition of the reagents. The reagents were added using a microsyringe and were injected below the surface of the pulp because it was noticed that otherwise they formed an oily film on the surface.

As these results were obtained from only six duplicated batch tests, it is unwise to draw too many conclusions. The method of conditioning is in itself a complicated subject.

However, on the basis of these results it was decided to continue adding the reagents separately at 5 per cent solids because this gave:

(a) better reproducibility than when adding the mixed reagent;

(b) results that were as good as conditioning at 15 per cent solids when adding the combined reagents.

4.3 PRELIMINARY FACTORIAL DESIGN ON GROOTEGELUK WASHED COAL

A complete replicated factorial design was carried out on the sample of Grootegeluk washed coal. Four factors viz. paraffin addition, MIBC addition, aeration rate and impeller speed, were examined. The objectives of this initial series of experiments were to gain an insight into the significant main effects and interactions and also to obtain an independent statistical estimate of the experimental error to be used in further work. This would allow confidence limits to be determined for the main effects and interactions.

From average values obtained from the literature survey, two levels of each of four factors were chosen for these preliminary tests. The combinations of parameter values that were investigated in each run are shown in Table 4.4. All the experiments were repeated three times to evaluate any error inherent in the experimental procedure.

The results obtained are listed in Appendix B, and in Table 4.5. The tables in Appendix B show the average kinetic model constants ϕ and k calculated from this data. Included in these results are the effect of surface oxidation on flotation; however, it was shown above (Section 4.1) that the combined effect of oxidation and experimental error was small. It was decided to use a 5 per cent level of significance (95 per cent confidence) as an indication of the factor or interaction whose effect was greater than the experimental or random error.

As may be seen from Table 4.5, which is calculated from the data in Appendix B, the most significant factor is the MIBC addition rate; this agrees with the findings of Eveson *et al* (1957). Furthermore, all interactions involving frother addition affect either the rate of

TABLE 4.4

Showing all possible combinations of the selected flotation conditions

			Impeller Speed (I)			
			1200 rpm		1500 rpm	
			Paraffin Addition (P) (g/t)			
			500	1000	500	1000
Aeration Rate (AE) (ℓ/min)	4	MIBC Addition (M) (mg/ℓ) 15				
		30				
	6	15				
		30				

TABLE 4.5

Summary of Results obtained for Factorial Designed Experiments on Grootegeluk Coal Sample

Treatment Combination	Effect on	
	Equilibrium Recovery	Flotation Rate
I	6,0	-0,008
P	0,0	0,0
AE	-4,4	0,0
M	21,8	0,0492
I x P	-10,4	-0,0305
I x AE	0,0	0,0
I x M	-18,3	0,0
P x AE	-8,4	0,0
P x M	0,0	-0,0248
AE x M	15,5	0,0531

flotation or equilibrium recovery of coal, or both. These interactions have been shown by Bennet *et al* (1958) and Burdon (1962) to play a role in determining the number and size distribution of the air bubbles.

It is interesting to note that apart from the interaction between frother addition and aeration rate (AE x M) all the other interactions have a detrimental effect on the recovery and flotation rate constant of coal.

These results may be taken at face value; on the other hand it is possible that the absence of highly significant main effects other than MIBC addition may be due to too limited a range of levels having been chosen for the other factors.

At this stage it became necessary to replace the impeller and air diffuser in the laboratory flotation cell because of excessive wear; a new error estimate therefore had to be obtained. It was also thought advisable to check the levels of the factors in case some of them needed to be altered in order to obtain as much meaningful information as possible from the work. The primary effects of changing individual factors while keeping all others constant was investigated. The factors studied were MIBC and paraffin dosage.

4.4 DETERMINATION OF REAGENT ADDITION RATES FOR USE IN THE FACTORIAL DESIGNED EXPERIMENTS

Initially these tests were carried out only on the Greenside washed coal sample but were later extended to include the other washed coals (Section 5.1.7).

4.4.1 MIBC Addition

In a series of batch flotation tests the MIBC concentration was varied between 5 and 25 mg/ℓ while all

other process variables were kept constant at the following levels:

Particle Size	95 per cent minus 300 μm
Solids Concentration	5 per cent
Paraffin Addition	1200 g/t
Aeration Rate	4 ℓ/min
Impeller Speed	1200 rpm

The equilibrium recovery and flotation rate data for coal and gangue are plotted in Figure 4.3 (a) and (b).

From Figure 4.3 (a) it can be seen that the recovery of coal and ash reached a maximum at an addition of 15 mg MIBC/ℓ pulp, with larger additions not causing any significant variation. This also corresponded to a maximum coal and gangue flotation rate constant (Figure 4.3 (b)).

From these results it would appear that the MIBC levels chosen in the initial factorial design were too high (i.e. 15 and 30 mg /ℓ) and thus were probably the cause of the highly significant effects. On the basis of the above results three MIBC addition rates were selected for the factorial design experiments with the four coals. These were 5, 10 and 15 mg /ℓ, which lie on the approximately linear portion of the curve.

The existence of a maximum equilibrium recovery might be caused by a pseudo-equilibrium at lower MIBC concentrations, as a result of starvation quantities of frother producing a non-stable, short-life froth.

The flotation rate constant increased with higher MIBC concentrations, reaching a maximum, as expected because as the surface tension is lowered, smaller and hence more bubbles are produced. Above a certain addition rate the presence of excess frother might not only form smaller bubbles but produce a froth

of more consistent character, i.e. the froth stability does not change as the float progresses.

4.4.2 Paraffin Addition

In another series of tests on the Greenside coal the paraffin addition was varied between 400 and 1400 g/t while all other factors were kept constant at the following levels:

Particle Size	95 per cent minus 300 μm
Impeller Speed	1200 rpm
Solids Concentration	5 per cent
Aeration Rate	3 ℓ/min
MIBC Addition	15 mg/ℓ

The equilibrium recovery and rate of flotation of coal and gangue are plotted in Figures 4.4 (a) and (b).

At higher paraffin concentrations the equilibrium recovery of coal and gangue increased reaching a maximum at a dosage 1,2 to 1,6 kg/t. Slightly higher dosages did not significantly increase the coal or gangue recoveries.

These same concentrations corresponded to a maximum flotation rate; higher and lesser dosages resulted in a lower flotation rate.

The effect of the paraffin is twofold (i) it increases the proportion of particles with sufficient hydrophobicity (surface coverage) so that the probability of fruitful bubble/particle adhesion is increased; and (ii) it acts as a frother, lowering the surface tension of the pulp.

The first effect must increase the recovery of particles from the pulp, while the second increases the rate of flotation for the reasons shown earlier. At some point, collector addition is no longer the limiting factor

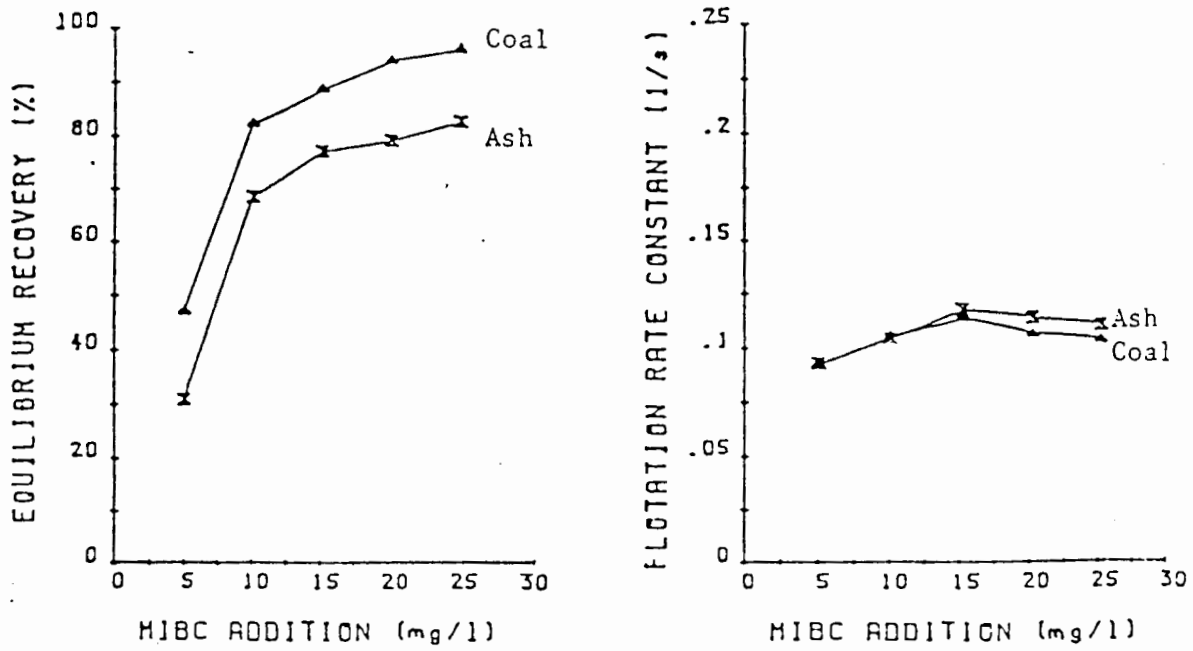

Figure 4.3 The effects of different MIBC concentrations on the kinetic model constants for the Greenside washed coal sample.

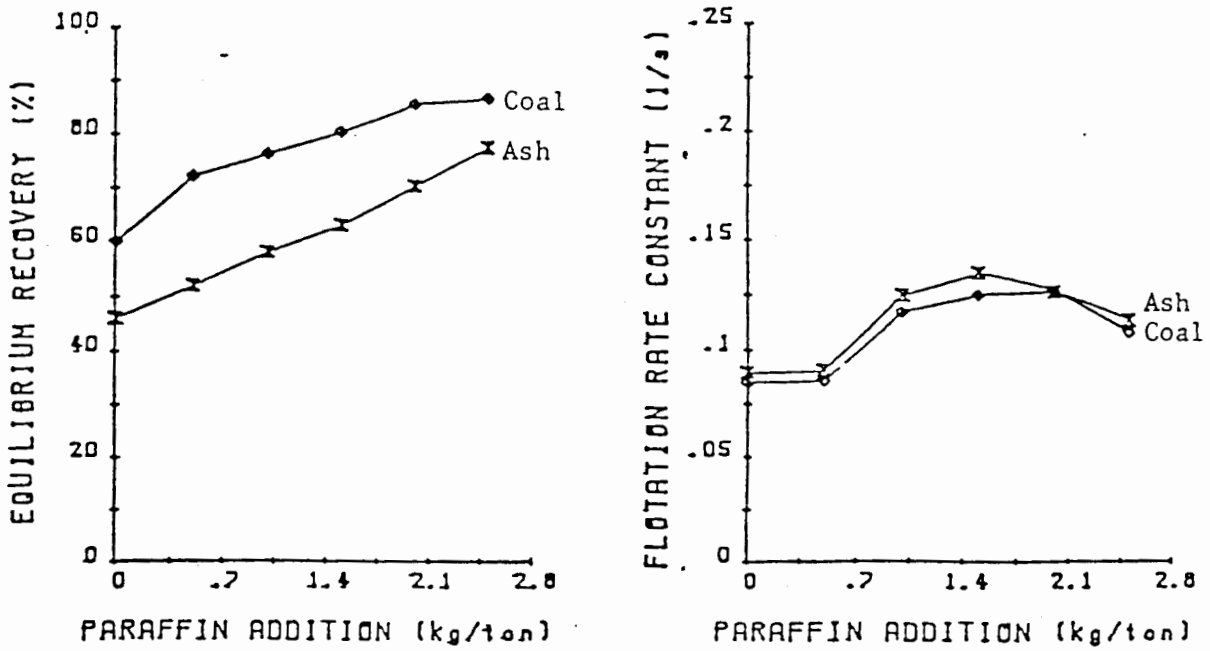

Figure 4.4 The effects of different paraffin addition rates on the kinetic model constants for the Greenside washed coal.

and therefore equilibrium is attained.

For these reasons the levels of paraffin dosage chosen for subsequent testwork were 800 g/t and 1200 g/t.

CHAPTER 5

RESULTS AND DISCUSSION

The investigation was divided into two stages. Firstly, a complete set of factorial design experiments was carried out on each of the four washed coals, to investigate the effects and interactions of the following parameters

(a) Paraffin addition/Impeller speed

(b) MIBC addition

(c) Aeration rate

(d) Particle size of feed

(e) Pulp density.

The levels of the factors were chosen on the basis of the preliminary factorial design and the one-at-a-time experiments carried out on the Greenside coal sample.

The objectives were to identify and quantify the important effects and interactions of the factors with the aim of determining the mechanisms involved in the froth flotation of South African coals and thus hopefully optimising the flotation process.

In the second stage of the investigation, a series of batch flotation tests was carried out on each of the washed and raw coal samples, at conditions obtained from the results of the first stage. The concentrates and tailings from these tests were analysed for maceral composition as well as ash. The aim was to obtain data on the relative flotation characteristics of the petrographic components of coal as well as the effect of varying rank on their floatability.

5.1 FIRST STAGE OF THE INVESTIGATION

5.1.1 Experimental Design

From the results of the batch tests on the Greenside coal it was concluded that the MIBC levels chosen in the initial factorial design experiments (f.d.e) on the Grootegeluk coal sample were too high and were probably the cause of the highly significant effect. Consequently, three MIBC addition rates were selected for the f.d.e. on the four coals. These were 5, 10 and 15 mg/ℓ which lie on the approximately linear portion of the curve.

Furthermore, since in the earlier f.d.e. neither paraffin addition nor impeller speed appeared to be highly significant, but their interaction was (c f Table 4.5), they were combined into a single factor. The levels selected for this factor (PA) were 800 g/t at 900 rpm and 1200 g/t at 1200 rpm.

It was also necessary to increase the range of aeration rates from 4 and 6 ℓ/min to 3 and 6 ℓ/min.

The preliminary factorial design was extended to include the effects of particle size distribution and pulp density. To decrease surface oxidation on the freshly milled coal surface, a decision was made to investigate two particle size ranges of feed, where at the lower level 95 per cent of the feed was to pass 500 μm and at the higher level 95 per cent was to pass 300 μm.

On the basis of the literature survey and two final year undergraduate projects (Jack (1983), Harris (1983)) two levels of pulp density were chosen at 5 and 10 per cent solids.

The combination of flotation parameters that were investigated during each run are most easily shown by means of Table 5.1.

TABLE 5.1

Combination of the Flotation Parameters Investigated

Particle Size (95% - x μm) (PS)	Aeration Rate (ℓ/min) (AE)	Percent Solids (PD)	MIBC Addition (mg/ℓ) (M)					
			5		10		15	
			Paraffin/Agitation (g/t : rpm) (PA)					
			800: 900	1200: 1200	800: 900	1200: 1200	800: 900	1200: 1200
500	3	5	1	2	3	4	5	6
		10	7	8	9	10	11	12
	6	5	13	14	15	16	17	18
		10	19	20	21	22	23	24
300	3	5	25	26	27	28	29	30
		10	31	32	33	34	35	36
	6	5	37	38	39	40	41	42
		10	43	44	45	46	47	48

5.1.2 Factorial Analysis

The factors investigated were aeration rate (AE), particle size (PS), pulp density (PD), MIBC addition (M) and the combined paraffin/agitation rate (PA) at the levels described above.

Because the impeller in the Leeds cell was replaced after the initial factorial design on the Grootegeluk coal was completed, it was necessary to repeat the reproducibility tests as it was felt that the error term would be different. The impeller was replaced because the rotor vanes were worn as a consequence of earlier work.

A half replicated factorial design was carried out on the four coals which in this case would yield an error sum of squares with 24 degrees of freedom. This is sufficient in terms of an F-test significance calculation. Furthermore, it was decided to retain the 5 per cent significance level when testing the null hypothesis.

The results obtained during the batch tests are summarised in Appendix C. Also tabulated are the equilibrium recovery and the first order rate constant for both the coal and associated gangue. These values are the Klimpel model constants obtained using the computerised curve fitting procedure (Section 3.4.4).

The f.d.e. were analysed using a standard, well documented statistical procedure which is described in Box, Hunter and Hunter (1978) and Davis (1980). The analysis of variance tables can be found in Appendix D. Third and higher order interactions have not been considered, and are assumed to be the result of highly significant lower order interactions. This is supported by Griffiths (1962) who states that 'three factor and higher order interactions can seldom be established at any reasonable confidence level.' Furthermore any attempt to

explain the higher order interactions with reference to first principles could be grossly misleading and controversial.

5.1.3 Results of the Factorial Designed Experiments

The results of the factorial design experiments are presented in Tables 5.2 to 5.5 using the notation shown in Appendix E. These tables show the magnitude with which the measured response changes when one of the parameters is changed. An analysis of these results is presented below.

The individual parameters (main effects) had the same effect on the measured responses for all the four coals.

The combined factor paraffin addition and impeller speed had the single most important effect on increasing the recovery of coal and gangue. The higher levels of pulp density and MIBC also increased the recovery. Increasing the aeration rate did not significantly affect the recovery. Although the finer particle size affected the recovery, this depended on the percentage of -38 μm material and the ash distribution in the feed.

Higher levels of aeration, paraffin/agitation rate and MIBC addition increased the flotation rate of coal and gangue, although the relative importance of the effects varied for the different coals.

Increasing the pulp density resulted in a lower rate of flotation.

It was noted above (Section 3.1) that the samples from Greenside and Grootegeluk were of similar rank as were the Nonsana and Longridge samples. However their maceral compositions were very different and as this is known to affect the floatability of coal, the results of

the interactions will be presented separately.

(a) Greenside Coal Sample

Generally the interactions resulted in a lower recovery and a higher rate of flotation. The most important interactions were those involving the combined factor paraffin addition/ impeller speed (PA) and MIBC concentration (Table 5.2).

(b) Grootegeluk Coal Sample

Most of the interactions resulted in an increase in the recovery and flotation rate of coal and gangue. All the interactions involving the combined factor PA affected these responses (Table 5.3).

(c) Nonsana Coal Sample

Although very few of the interactions affected the recovery, they were responsible for changing the rate of flotation of coal and gangue. Again the most significant factor involved in the interactions was the combined parameter of paraffin addition and impeller speed (PA) (Table 5.4).

(d) Longridge Coal Sample

The primary effects of the interactions were to change the flotation rate of coal and gangue, decrease the recovery of coal and increase the recovery of gangue. The most significant was the frother dosage (M) and the combined factor (PA) (Table 5.5).

TABLE 5.2

Summary of the Factors and their Interactions which show an effect on the 5% significance level - Greenside No 2 Seam Coal Sample

Contrast	Effect			
	Equilibrium Recovery (%)		Flotation Rate (x 10^3 s^{-1})	
	Coal	Gangue	Coal	Gangue
M12	16,7	19,5	43,1	41,7
M23	8,4	7,9	0,0	0,0
PA	17,5	17,1	44,8	40,4
PS	0,0	0,0	26,6	23,3
AE	0,0	6,2	66,1	68,1
PD	13,7	16,4	-22,1	-22,2
PA x M12	0,0	0,0	42,9	44,2
PA x M23	-13,9	-12,1	6,4	0,0
PS x M12	4,9	11,0	0,0	0,0
PS x M23	-8,8	-13,1	0,0	27,9
AE x M12	6,3	6,7	47,9	39,9
AE x M23	0,0	6,7	38,3	20,3
PD x M12	-7,6	6,3	-11,0	0,0
PD x M23	0,0	-5,8	21,6	0,0
PA x PS	-12,6	-14,9	0,0	12,1
PA x AE	-7,8	0,0	49,3	32,4
PA x PD	-8,7	-11,1	34,5	-48,1
PS x AE	0,0	0,0	-15,5	0,0
PS x PD	0,0	0,0	-21,3	0,0
AE x PD	-5,5	-9,8	35,1	12,1
MEAN	69,0	55,4	133,6	140,6

TABLE 5.3

Summary of the Factors and Interactions which show an effect on the 5% significance level - Grootegeluk Coal Sample

Contrast	Effect			
	Equilibrium Recovery (%)		Flotation Rate (x 10^3 s^{-1})	
	Coal	Gangue	Coal	Gangue
M12	9,9	6,4	39,3	41,9
M23	14,9	12,6	25,1	26,3
PA	10,7	7,0	52,7	54,0
PS	3,0	0,0	0,0	0,0
AE	0,0	0,0	39,8	37,1
PD	7,1	3,7	0,0	0,0
PA x M12	3,0	3,9	45,3	45,8
PA x M23	6,3	3,5	10,8	9,0
PS x M12	0,0	0,0	0,0	0,0
PS x M23	0,0	0,0	0,0	0,0
AE x M12	0,0	0,0	25,3	32,0
AE x M23	0,0	0,0	0,0	0,0
PD x M12	0,0	2,9	12,1	11,8
PD x M23	0,0	-2,9	48,3	-52,5
PA x PS	-4,1	-2,7	17,8	14,9
PA x AE	0,0	3,4	54,1	53,8
PA x PD	6,7	4,2	0,0	0,0
PS x AE	0,0	0,0	17,1	19,3
PS x PD	2,5	6,8	0,0	0,0
AE x PD	0,0	0,0	30,4	33,9
MEAN	24,7	15,1	75,8	76,0

TABLE 5.4

Summary of the Factors and Interactions which show an effect on the 5% significance level - Nonsana Coal Sample

Contrast	Effect			
	Equilibrium Recovery (%)		Flotation Rate (x 10^3 s^{-1})	
	Coal	Gangue	Coal	Gangue
M12	5,8	5,8	10,6	13,1
M23	7,0	7,6	15,0	8,5
PA	16,9	16,6	0,0	0,0
PS	14,6	6,3	-62,4	-60,1
AE	0,0	0,0	25,5	17,8
PD	6,5	5,3	-47,2	-46,9
PA x M12	0,0	0,0	-21,7	-27,0
PA x M23	0,0	0,0	0,0	0,0
PS x M12	0,0	0,0	0,0	0,0
PS x M23	0,0	0,0	-14,4	0,0
AE x M12	0,0	0,0	0,0	0,0
AE x M23	0,0	0,0	-22,3	14,1
PD x M12	0,0	-5,2	-18,1	-15,2
PD x M23	0,0	0,0	-28,6	-11,1
PA x PS	0,0	0,0	22,2	26,9
PA x AE	0,0	0,0	73,4	78,1
PA x PD	4,6	5,3	-30,6	-27,0
PS x AE	6,8	3,1	23,0	36,2
PS x PD	0,0	12,9	25,9	21,5
AE x PD	-11,2	8,0	-15,0	-26,9
MEAN	49,9	36,5	106,5	108,4

TABLE 5.5

Summary of the Factors and Interactions which show an effect on the 5% significance level - Longridge Coal Sample

Contrast	Effect			
	Equilibrium Recovery (%)		Flotation Rate (x 10^3 s^{-1})	
	Coal	Gangue	Coal	Gangue
M12	8,1	7,0	0,0	0,0
M23	5,7	4,3	24,1	62,5
PA	23,3	19,5	-29,6	-22,7
PS	-7,4	-8,9	0,0	0,0
AE	0,0	0,0	83,8	69,0
PD	11,1	5,5	-65,7	42,7
PA x M12	0,0	7,3	-30,5	0,0
PA x M23	6,4	0,0	-46,8	-60,0
PS x M12	-4,5	0,0	-22,0	-41,8
PS x M23	0,0	0,0	76,5	68,7
AE x M12	0,0	7,1	31,3	0,0
AE x M23	-6,5	4,8	23,6	18,9
PD x M12	0,0	0,0	56,8	54,9
PD x M23	-14,7	-14,4	58,5	-26,7
PA x PS	0,0	0,0	16,6	0,0
PA x AE	-9,0	-4,5	0,0	-28,8
PA x PD	12,2	9,5	-20,3	0,0
PS x AE	0,0	0,0	55,5	78,5
PS x PD	0,0	0,0	-21,0	-57,6
AE x PD	0,0	6,9	-31,1	-48,4
MEAN	65,6	41,3	123,8	129,7

Probably the most important point arising from the results is that although changing the flotation parameters affected the recovery and flotation rate of coal, there was a corresponding equivalent change in these responses for the ash. This might have been the result either of insufficient liberation of the finely dispersed gangue or alternatively unselective process conditions. Taking into account the results of the float/sink experiments (Section 5.1.4.4) the former is the more probable, but not the only reason for this phenomenon.

5.1.4 Interpretation of the Results

A summary of the results obtained from the factorial design experiments is given in Table 5.6. The results are different for each of the coal samples studied; the effect of changing the parameters varied depending on the individual coal. This is especially apparent when the interactions between the factors are considered.

5.1.4.1 Main Effects:

(a) MIBC Addition (M) The overall effect of increasing the MIBC dosage was to increase the recovery and flotation rate of coal and gangue (Table 5.6).

The increase in recovery was probably a pseudo effect and not a result of increasing the surface hydrophobicity of the particles. This might be the result of two mechanisms

(i) Frother addition is the limiting factor so a stable mineral laden froth is not formed

(ii) At the higher flotation rates, the rate of mass pull increases and the selectivity drops as entrainment and entrapment increase.

The increase in flotation rate was the result of lowered surface tension which produced a larger number of smaller bubbles at constant aeration.

TABLE 5.6

The Relative Effects of the Factors and their Interactions for the different Coals

Note: (i) GS refers to Greenside
GG refers to Grootegeluk
NO refers to Nonsana
LN refers to Longridge

(ii) + positive significant effect
- negative significant effect
0 no significant effect

Contrast	Effect															
	Equilibrium Recovery								Flotation Rate							
	Coal				Gangue				Coal				Gangue			
	GS	GG	NO	LN	GS	GG	NO	LN	GS	GG	NO	LN	GS	GG	NO	LN
M12	+	+	+	+	+	+	+	+	+	+	+	0	+	+	+	0
M23	+	+	+	+	+	+	+	+	0	+	+	+	0	+	+	+
PA	+	+	+	+	+	+	+	+	+	+	0	-	+	+	0	-
PS	0	-	+	-	0	0	+	-	+	0	-	0	+	0	-	0
AE	0	0	0	0	+	0	0	0	+	+	+	+	+	+	+	+
PD	+	+	+	+	+	+	+	+	-	0	-	-	-	0	-	+
PA x M12	0	+	0	0	0	+	0	+	+	+	-	-	+	+	-	0
PA x M23	-	+	0	+	-	+	0	0	+	+	0	-	0	+	0	-
PS x M12	+	0	0	-	+	0	0	0	0	0	0	-	0	0	0	-
PS x M23	-	0	0	0	-	0	0	0	0	0	-	+	+	0	0	+
AE x M12	+	0	0	0	+	0	0	+	+	+	0	+	+	+	0	0
AE x M23	0	0	0	-	+	0	0	+	+	0	-	+	+	0	+	+
PD x M12	-	0	0	0	+	+	-	0	-	+	-	+	0	+	-	+
PD x M23	0	0	0	-	-	-	0	-	+	+	-	+	0	-	-	-
PA x PS	-	-	0	0	-	-	0	0	0	+	+	+	+	+	+	0
PA x AE	-	0	0	-	0	+	0	-	+	+	+	0	+	+	+	-
PA x PD	-	+	+	+	-	+	+	+	+	0	-	-	-	0	-	0
PS x AE	0	0	+	0	0	0	+	0	-	+	+	+	0	+	+	+
PS x PD	0	+	0	0	0	+	+	0	-	0	+	-	0	0	+	-
AE x PD	-	0	-	0	-	0	+	+	+	+	-	-	+	+	-	-

For the anthracite coals the incremental change in flotation rate was greater at the higher MIBC concentration (M23 > M12). The reverse was true for the lower rank bituminous coals i.e. Greenside and Grootegeluk.

(b) Paraffin Addition/Impeller Speed (PA) The overall effect of the combined parameter was to increase the equilibrium recovery of coal and gangue.

The recovery increase was probably caused by the higher collector concentration, indicating that at the lower level (800 g/t) the paraffin dosage was limiting the recovery of coal and gangue. Also the lower impeller speed might not have been keeping the coarser particles in suspension (Dunne, 1982).

The higher impeller speed (1200 rpm) would form smaller and more numerous air bubbles which would increase the flotation rate as was seen for the bituminous coals. This factor did not significantly change the flotation rate of the Nonsana coal but decreased the flotation rate constant for the Longridge coal sample.

(c) Aeration Rate (AE) As expected, increasing the aeration rate resulted in much higher flotation rates but did not affect the recovery of coal and gangue. The higher aeration rate supplies more air bubbles to the system which increases the probability of bubble/particle contact and therefore the flotation rate.

Any associated increase in recovery would probably have been caused by greater entrainment.

(d) Pulp Density (PD) The higher solids concentration generally resulted in a higher recovery and lower flotation rate constant (recovery of the gangue increased for the Longridge coal sample).

The lower recovery at the lower pulp density agrees with the findings of Dell (1968). This might be caused

by insufficient particles in the froth which results in decreased froth stability so that the hydrophobic particles are not being removed from the cell.

At the higher pulp density there is a greater probability of particle/bubble contact so the flotation rate would be expected to increase. The higher concentration must increase particle competition for the same number of bubbles, restricting the relative movement of particles and lowering the flotation rate.

5.1.4.2 Interactions: For the reasons discussed earlier only two factor interactions have been considered. The results of the factorial design experiments showed that the magnitude and effects of the interactions on the measured responses were not consistent for all the coals over the same range of levels of the factors. This might be explained by the fact that even though the Greenside and Grootegeluk coal samples were of similar rank, as were those from Nonsana and Longridge, their maceral compositions were all very different. Thus the individual coals would be expected to behave differently.

The interactions would be negative, positive or zero depending on where the operating point lay on the response surface. Two or more plausible and competing mechanisms can be proposed to explain the variations in each of the interactions.

(a) Particle Size x MIBC Concentration (PS x M): Although neither of these factors influences the surface hydrophobicity and should not influence the recovery, the MIBC concentration may be a limiting factor, so that at the lower levels an unstable froth is formed; at the higher dosages the smaller bubbles and fine particles lower the particle/bubble collection and collision rate. There is

probably an optimum bubble/particle size relationship which affects the flotation rate constant.

(b) Pulp Density x MIBC Concentration (PD x M): The higher concentration of particles and the presence of smaller more numerous bubbles would increase the competition between hydrophobic particles for the air bubbles, lowering the recovery. Furthermore the smaller bubbles would decrease the efficiency of particle/bubble collision. There would be more entrainment as the mass transfer from the pulp to the froth phase increased. The higher concentration of particles and bubbles would increase the flotation rate constant.

Because the MIBC concentration was determined in mg/ℓ pulp and not mg /kg solids this might be a limiting factor; the equilibrium recovery would be approached sooner and because of the rate/recovery relationship the flotation rate constant would be higher.

(c) Aeration Rate x MIBC Concentration (AE x M): These two parameters control the size and number of bubbles in the flotation system. Generally associated with a higher flotation rate is increased entrainment resulting in higher gangue recoveries as selectivity decreases.

(d) Particle Size x MIBC Concentration (PD x M): At the higher levels of these two factors the concentrations of particles and bubbles increased; the result is a greater probability of fruitful particles/bubbles collisions and the flotation rate constant increases. However at some point the mass

transfer from the pulp to the froth is restricted and this interaction causes a reduction in the flotation rate constant

(e) Particle Size x Aeration Rate (PS x AE): The overall effect is to increase the flotation rate constant because for finer particles and a larger number of bubbles the probability of particle/bubble collisions increases. The total surface area of the finer particles has increased and therefore a greater bubble surface area is needed for the same flotation rate constant.

(f) Particle Size x Pulp Density (PS x PD): The concentration of particles in the cell increases with fineness of grind and pulp density, resulting in a higher probability of particle/bubble contact. However there exists a critical proportion of hydrophobic particles to bubbles, where the surface area provided by the bubbles is the limiting factor; the time taken for the system to reach equilibrium must increase and the flotation rate constant decreases.

(g) Aeration Rate x Pulp Density (AE x PD): Again the flotation rate constant increases with the number of bubbles and particle concentration until the mass transfer through the pulp/froth interface is the limiting factor.

(h) Interactions involving Paraffin Addition and Impeller Speed (PA): These are confounded three factor interactions where

the individual two factor interactions cannot be independantly assessed.

To aid the interpretation of the results it is useful to simplify the events involved in the flotation process. A five step model is proposed, as shown in Figure 5.1. The steps are:

(i) Collector adsorption - for soluble collectors this corresponds to the rate and extent of adsorption. However with oily collectors the degree of collector dispersion and probability of collector droplet and particle collision will determine the recovery and rate of flotation respectively.

(ii) Bubble/particle collision - the concentration and size of the bubbles and particles will affect the rate of flotation by determining the probability of collision. The probability of successful bubble and particle adhesion will affect the recovery.

(iii) Mass transfer of bubble and particle into froth phase - this will be affected by the ascent velocity, bubble loading and relative bubble crowding.

(iv) Elutriation - this will be affected by the froth stability, and will in turn affect the recovery, grade and flotation rate.

(v) Froth removal - the froth stability will affect both the recovery and flotation rate The flotation rate and the recovery will be affected by the froth velocity. High froth removal rates will decrease elutriation and hence increase the entrainment as well as increasing the mass transfer of particles

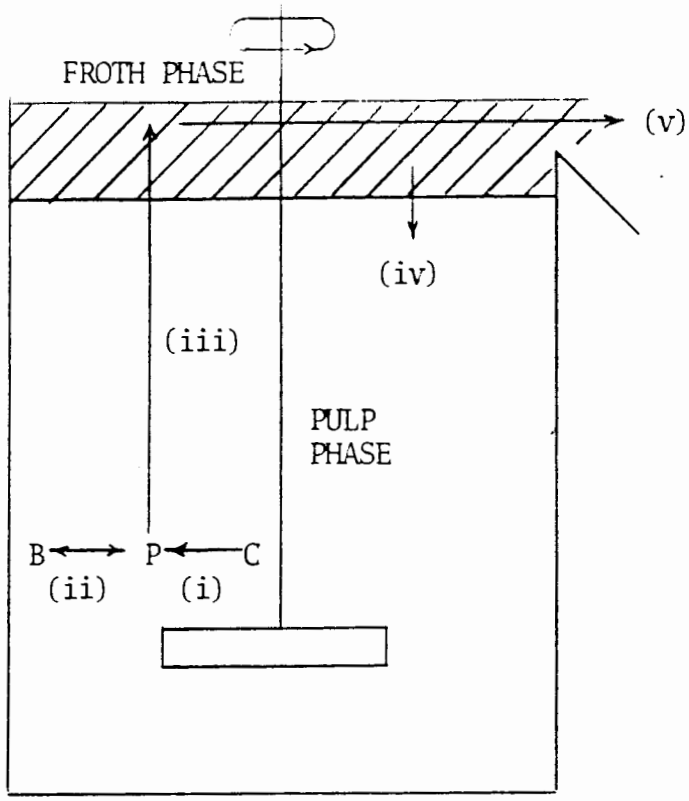

Figure 5.1 Simplified five stage model of flotation (B - bubble, C - collector, P - mineral particle)

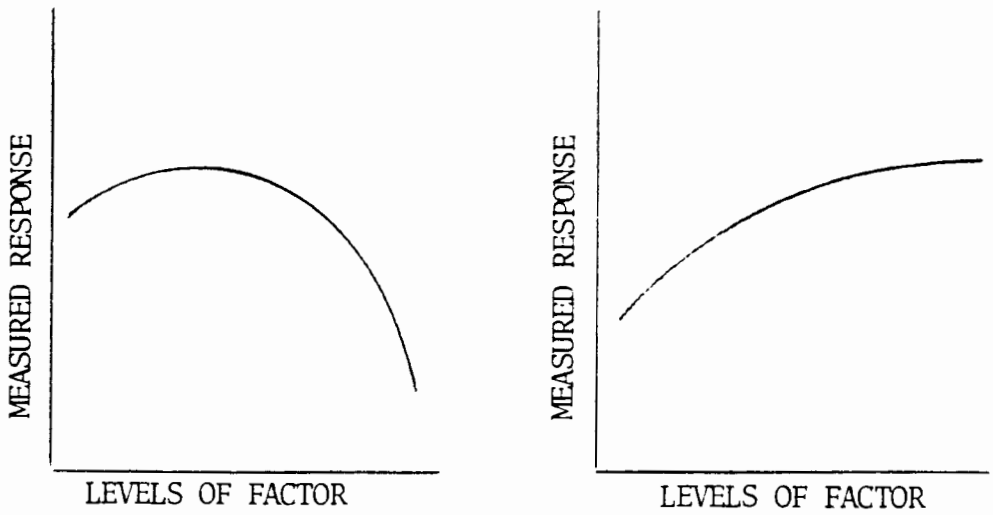

Figure 5.2 Simplified effect of two or more opposite and competing mechanisms.

into the froth phase. A highly stable slow moving froth will decrease the rate of flotation and possibly prevent the transfer of particles into the froth phase. It might also increase entrainment again by preventing elutriation. Entrapment would also increase.

Although flotation can be considered in these terms as a simple five step process it is obvious that the steps are interrelated. The degree of interrelation will vary to determine the limiting step and therefore the process efficiency. The extent of interrelation will determine the magnitude of the interactions. The levels of the factors will affect each of these steps to varying degrees.

It would be reasonable to expect that there are two or more opposite and competing mechanisms which result in the measured response either passing through a maximum, minimum or approaching an asymptote (Figure 5.2).

With different coals the optimum corresponds to different levels of the flotation factors and this will account for the measured response change increasing, decreasing or showing no effect.

The rate of flotation is analogous to a chemical rate of reaction so this would be expected to go through a maximum, decreasing on either side of the optimum. This would correspond to the rate of the slowest step in a series of rate controlling mechanisms.

The interpretation of the main effects and interactions support the simplified five step model of flotation because it is feasible to propose controlling or limiting mechanisms by considering the effects of these factors on each of the steps. However flotation research is not sufficiently advanced to quantify accurately, or to separate, the individual stages in the flotation process. This has been attempted by Moys (1978, 1984) who tried to

model the flotation froth phase from residence time distribution data. Most importantly this five-step mechanism provides a valuable insight into the sub-processes involved in flotation.

Summarising these results it is apparent that although the recovery of coal is governed by the surface hydrophobicity, other factors, either directly or indirectly, affect the viability of flotation.

The major effect of the parameters, at the levels at which they were studied, was to change the probability of successful particle/bubble attachment. But in the presence of other processes, their combined effects could be more important if they were the limiting factors. That is, the sum of the interactions can be more important than the main effects.

Analagous to the classical recovery/grade trade-off is the rate/recovery compromise that has been shown by Klimpel (1980), where changing an operating variable can result in an increased flotation rate constant and a decrease in recovery, or vice versa. Obviously increasing both is desirable. Generally, as shown above, as the rate of flotation is increased the recovery of gangue through entrainment rises and the grade decreases.

5.1.5 Effect of Rank and Maceral Composition

The rank of the coal samples investigated increased in the order Grootegeluk, Greenside, Nonsana, Longridge and the reactive maceral composition increased in the order Nonsana, Greenside, Longridge and Grootegeluk (Figure 3.1).

Table 5.7 shows that it is not possible from the results of the experimental work undertaken to order the floatability of different coals on the basis of rank or maceral composition. It would appear that floatability is a function of both characteristics, so that the optimum flotation conditions for the different coals are very

TABLE 5.7

Average Equilibrium Recoveries and Rates of Flotation of Coal and Gangue for the four Coal Samples.

	Content of Reactive Macerals	Increasing Rank	Equilibrium Recovery		Flotation Rate	
			Coal	Gangue	Coal	Gangue
Grootegeluk	90,0	1	24,7	15,1	0,0758	0,0760
Greenside	65,2	1	69,0	55,4	0,1336	0,1406
Nonsana	58,9	2	49,9	36,5	0,1065	0,1084
Longridge	75,9	2	65,6	41,3	0,1238	0,1297

different. This can be seen in Table 5.6 and Figures 5.3 to 5.6.

The interactions for the Greenside and Longridge coal samples show approximately the same number of increasing and decreasing effects (Table 5.6). This shows that the levels of the factors chosen cover the optimum range of flotation conditions. The interactions for the Nonsana coal sample show that there are slightly more increasing effects than decreasing effects. This indicates that the range of levels chosen for the f.d.e. approaches optimum conditions. The majority of the interactions for the Grootegeluk coal sample however, increase the recovery and flotation rate. This shows that the levels of the parameter chosen for the f.d.e. were too low. This is supported by the results of the one-at-a-time tests done on the other coals (Section 5.1.7).

5.1.6 Rate and Ash Content versus Recovery

These compromises were briefly discussed in Section 5.1.4.2. This section covers the qualitative relationship between the flotation recovery and rate constant and the ash content. The effectiveness of flotation was compared with an ideal relative density separation.

Table 5.8 shows the flotation conditions which generally gave the largest and smallest flotation rate constants for each of the four coals. This shows that the two factors that have the most influence on the flotation rate constant are the frother concentration and aeration rate.

Figures 5.3 to 5.6 show cumulative recovery of coal in the concentrate against ash. The numbered points correspond to the batch test numbers given in Table 5.1.

Although the testwork was carried out on washed coal samples it should still be possible to compare the batch runs on the basis of concentrate ash. This might not be accurate but a qualitative relationship between concentrate ash and flotation rate constant can be seen if figures 5.1 to 5.4 are compared with Table 5.8.

TABLE 5.8

The Levels of Factors, which gave the Highest and Lowest Flotation Rate Constants.

	MIBC (mg/ℓ)	Paraffin/ Agitation (g/t : rpm	Aeration (ℓ/min)	Pulp Density	Particle Size
Highest k	10 or 15	1200 : 1200	6	Either	Either
Lowest k	5	Either	3	Either	Either

They show that the high flotation rate constant corresponded to high concentrate ashes and similarly low rates corresponded to the low ashes. The corollary is not true.

The high ashes may correspond to the portion on the response surface after the optimum, where, although the flotation rate constant is decreasing, a different mechanism is controlling the rate; but the initial sub-process responsible for the high ashes still exists, i.e. entrainment is still occurring but the time to reach equilibrium recovery has increased.

Most importantly, these figures show that for efficient operation the various parameters involved in the flotation process have to be controlled within a narrow range of values. For each coal these values may be identified and were in fact used in Section 5.2 on the raw coals.

Figures 5.3 to 5.6 also show the results of float and sink tests done on deslimed (+ 38 μm) flotation feed samples using the method of Franzidis and Harris (1985). Specially designed and modified centrifuge tubes were used to carry out the density separations, with zinc chloride solutions as the dense medium.

A comparison of the results of the float and sink analyses with those of the flotation tests shows that, except for the Grootegeluk coal sample, froth flotation is nearly as efficient as an ideal relative density separation provided the flotation parameters are controlled within narrow limits.

These results support the conclusions of Section 5.1.3 which showed that although the low process selectivity may be caused by two different factors, viz. the collector or limited liberation, the latter is probably the most likely. Therefore finer grinding is necessary to provide complete liberation of coal and

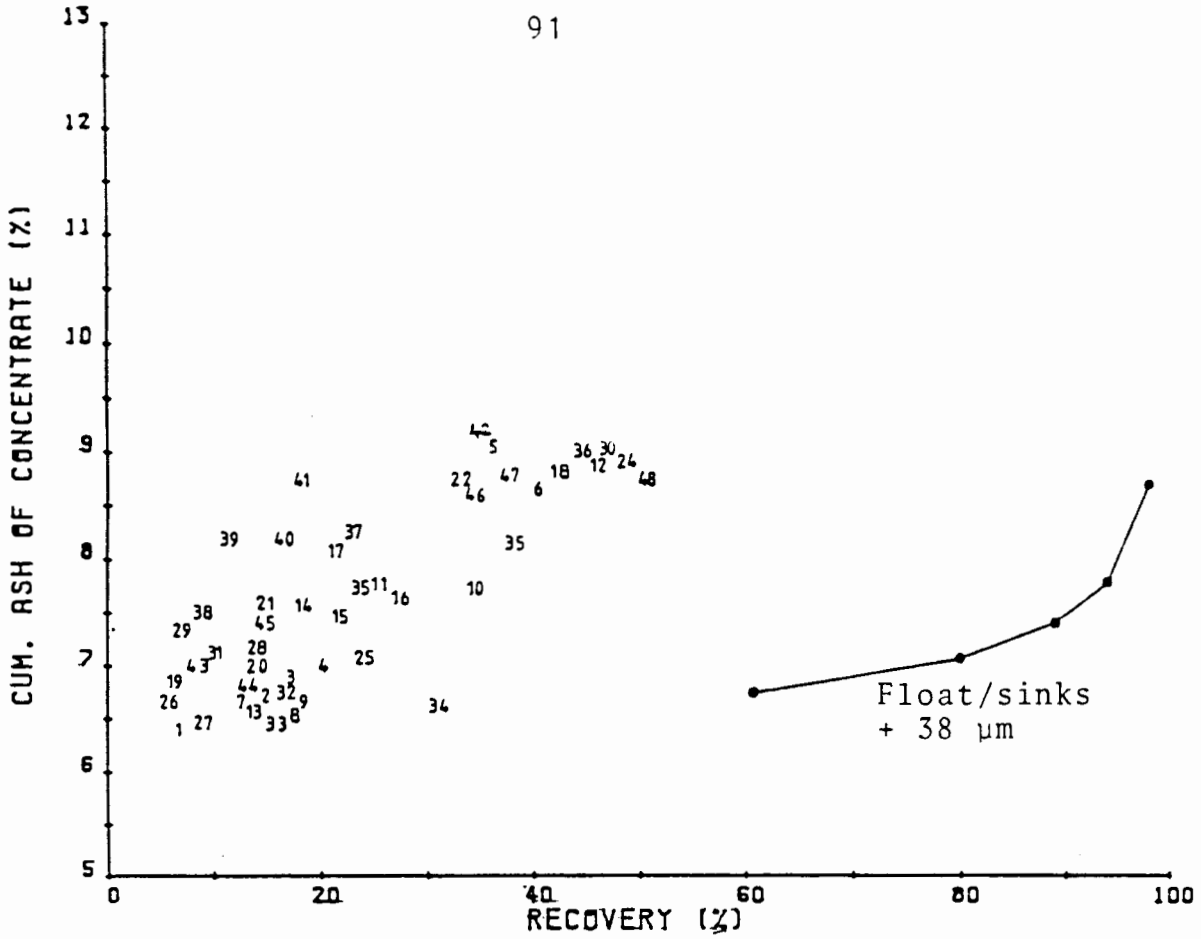

Figure 5.3 Recovery-Ash profile for batch flotation compared to density separation for the Grootegeluk coal sample.

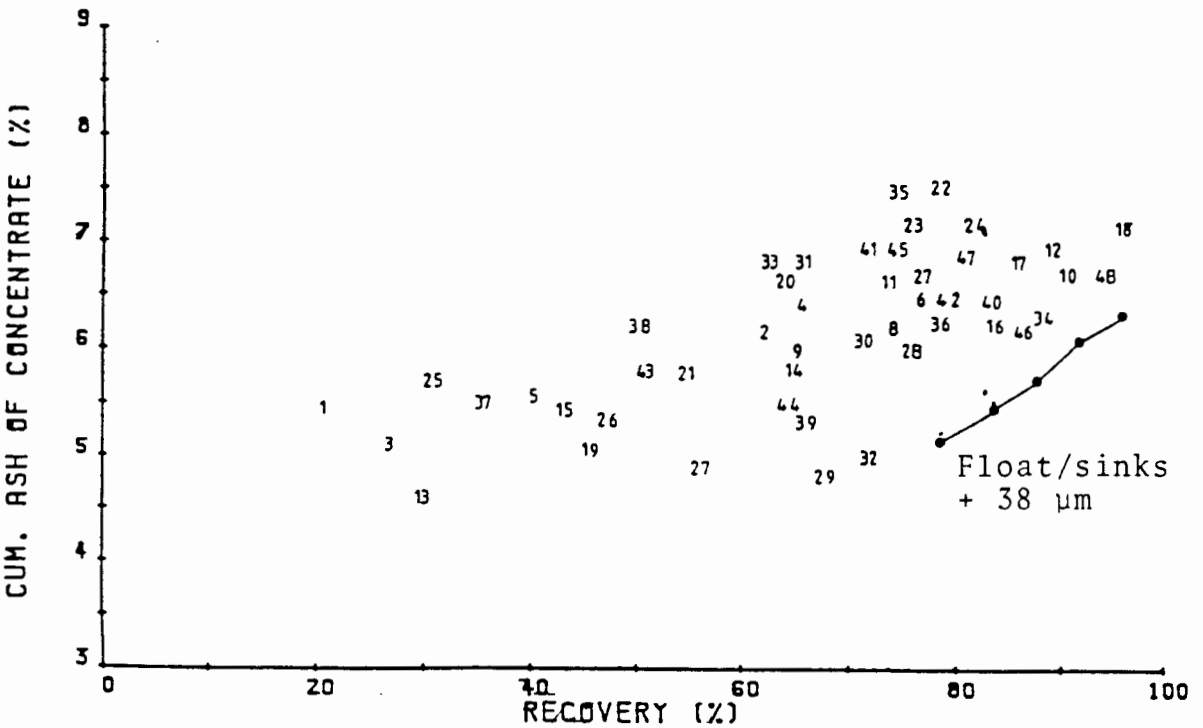

Figure 5.4 Recovery-Ash profile for batch flotation compared to density separation for the Greenside coal sample.

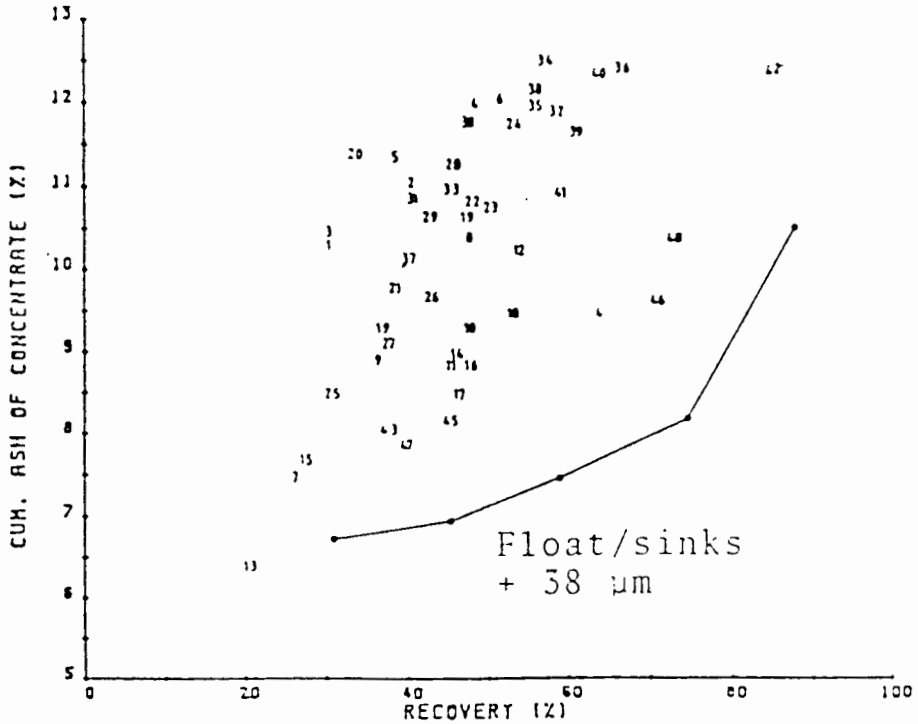

Figure 5.5 Recovery-ash profile for batch flotation compared to density separation for the Nonsana washed coal sample.

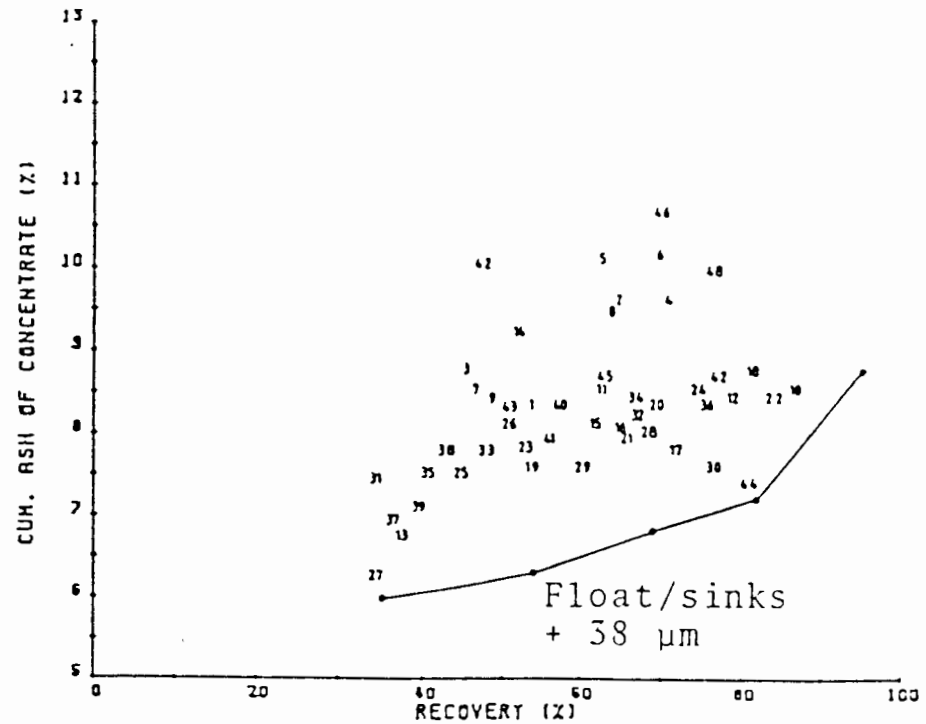

Figure 5.6 Recovery-ash profile for batch flotation compared to density separation for the Longridge washed coal sample.

gangue but the actual liberation sizes should be investigated.

5.1.7 One-at-a-time Experiments

In a series of flotation tests on the Grootegeluk, Nonsana and Longridge coal samples (similar to those done on the Greenside coal sample (Section 4.4)), the paraffin and MIBC concentrations were varied. This aim was to check that the levels of these factors used in the factorial design experiments were of the right orders of magnitude.

These results are plotted in Figures 5.7 to 5.14. The results for the Nonsana and Longridge samples show similar trends to those for the Greenside samples. However, the results of the tests carried out on the Grootegeluk coal sample show that the reagent dosages for the f.d.e. on this sample were low (c f Section 5.1.5).

These figures also show that the natural hydrophobicity of the Grootegeluk coal sample was lower than that of the other coal samples, because there was a much larger variation in flotation recovery and rate constant when the MIBC dosage was varied at different paraffin addition rates. When no collector was added the recovery of Grootegeluk coal was less than 5 per cent, irrespective of the frother dosage, while the other coals exhibited a much higher natural hydrophobicity with higher recoveries when no collector was used. The selectivity also increased when collector was used in the floats.

The paraffin collector appears to have two functions (1) to extend the natural hydrophobicity; (2) to induce hydrophobicity in the coal sample. This might depend on the chemical and physical characteristics of the exposed coal surface.

In general these results show that an optimum reagent

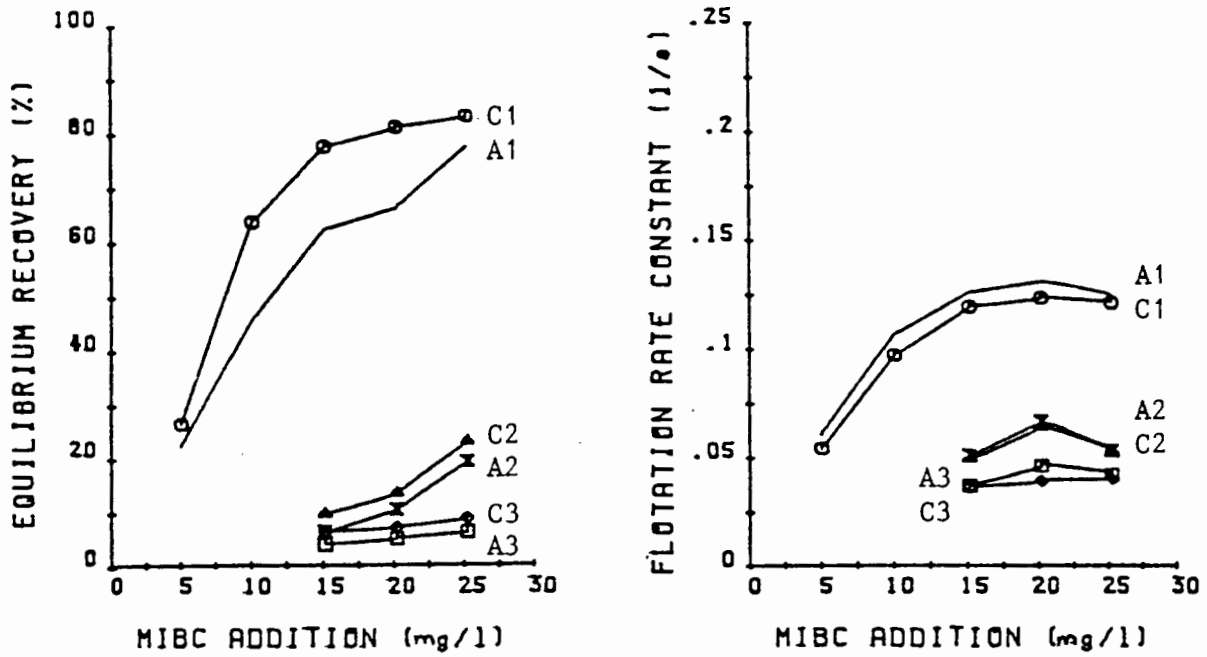

Figure 5.7 The effect of MIBC concentration on the kinetic model constants for the Grootegeluk washed coal sample. (C - coal, A - ash, 1 - 2600 g/t paraffin, 2 - 1200 g/t paraffin, 3 - no paraffin).

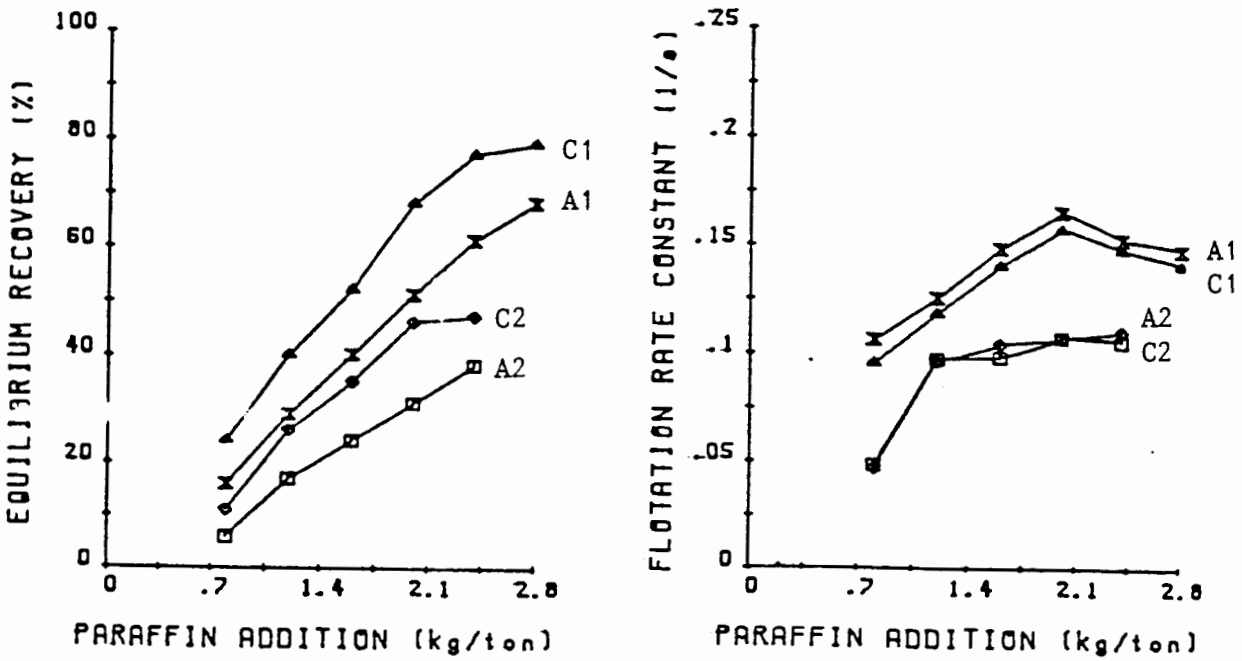

Figure 5.8 The effect of paraffin dosage on the kinetic model constants for the Grootegeluk washed coal sample. (C - coal, A - Ash, (1 - 20 mg/ℓ MIBC, 2 - 15 mg/ℓ MIBC)

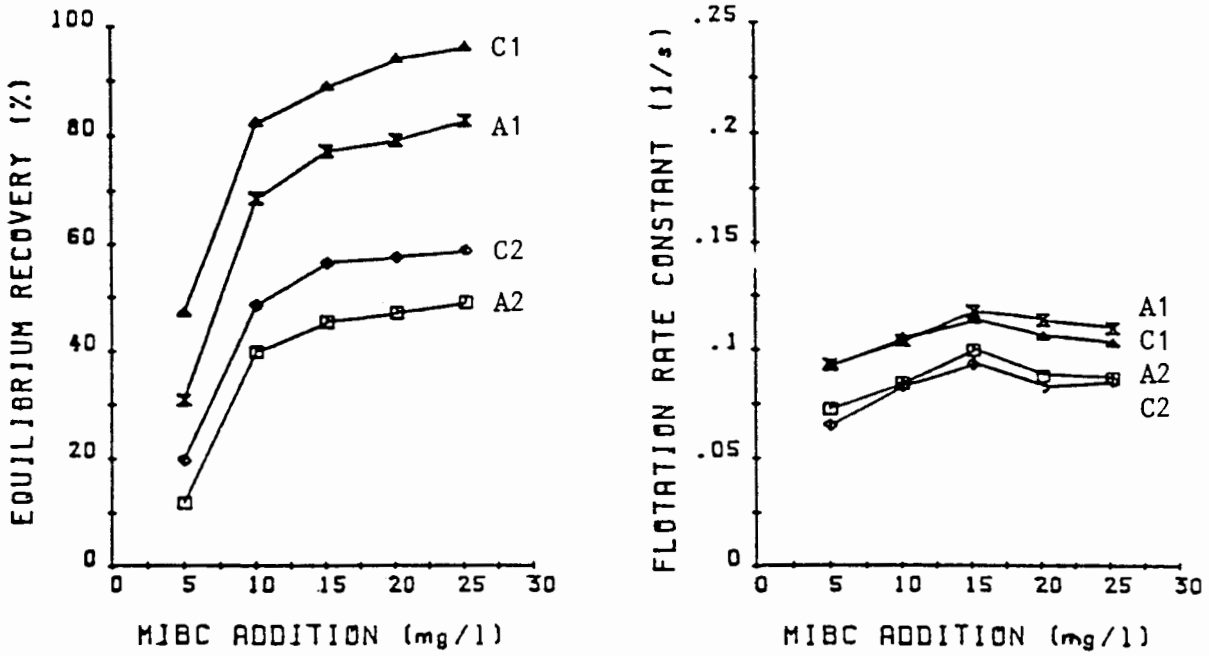

Figure 5.9 The effect of different MIBC concentrations on the kinetic model constants for the Greenside washed coal sample. (C - coal, A - ash; 1 - 1200 g/t paraffin, 2 - no paraffin).

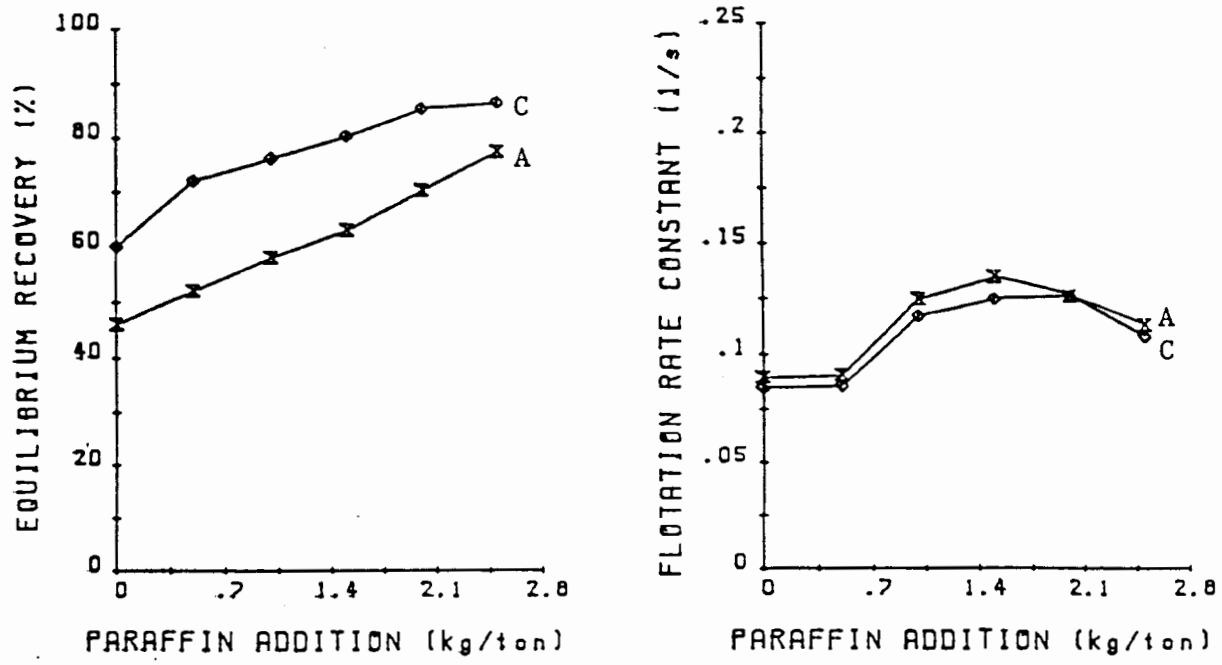

Figure 5.10 The effect of different paraffin dosages on the kinetic model constants for the Greenside washed coal sample. (C - coal, A - ash).

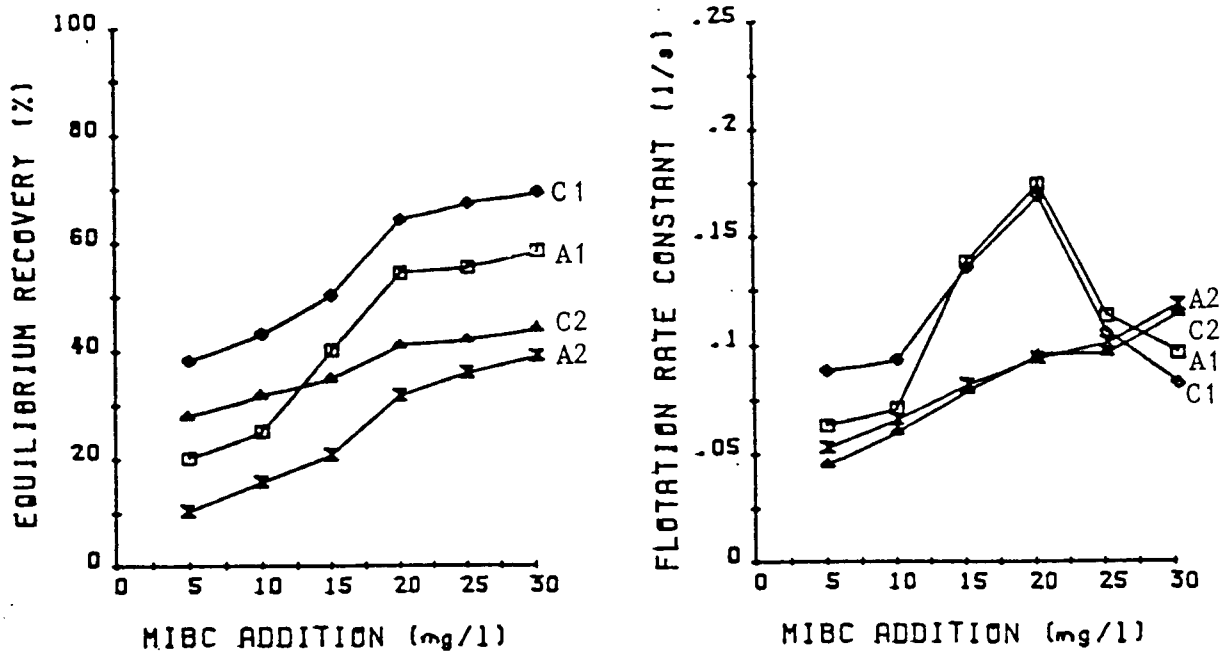

Figure 5.11 The effect of MIBC concentration on the kinetic model constants for the Nonsana washed coal sample.
(1 - 1200 g/t paraffin, 2 - no paraffin).

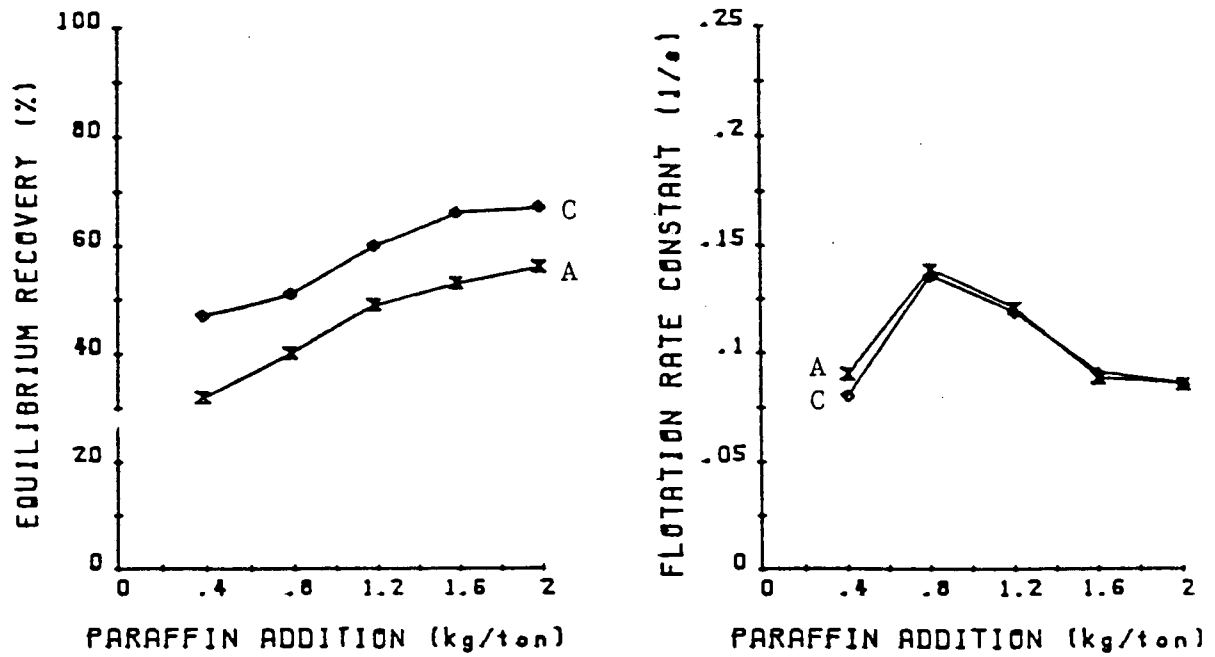

Figure 5.12 The effect of paraffin dosage on the kinetic model constants for the Nonsana washed coal sample. (C - coal, A - ash).

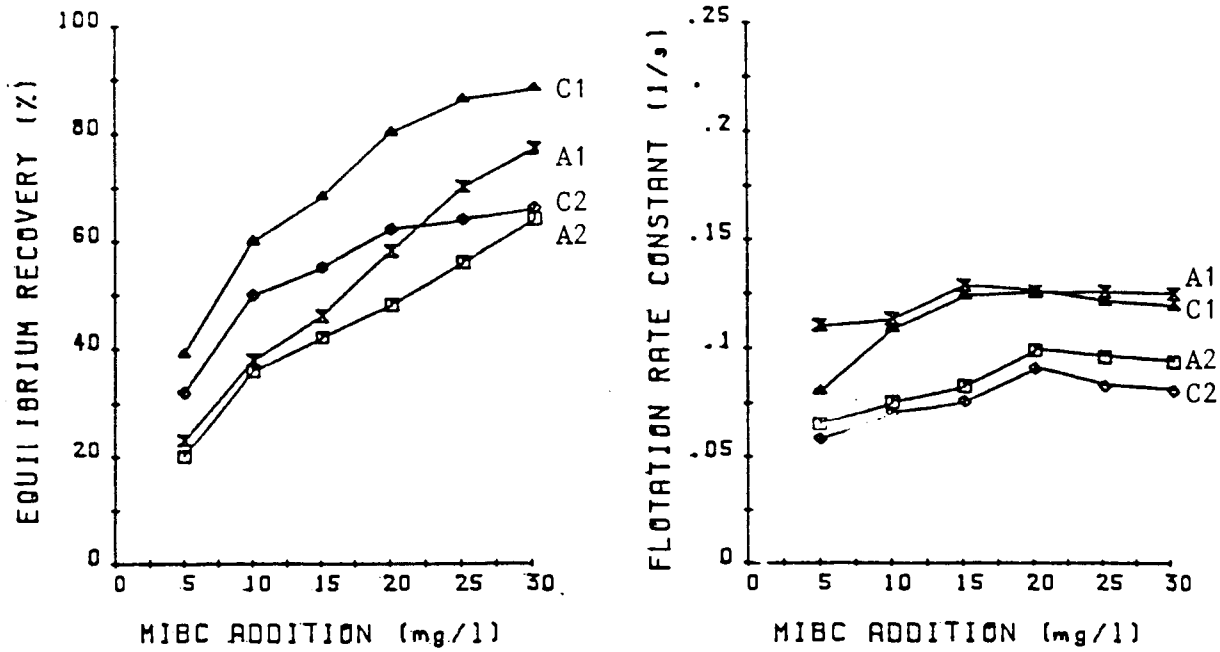

Figure 5.13 The effect of MIBC concentration on the kinetic model constants for the Longridge washed coal sample. (C - coal, A - ash, 1 - 1200 g/t paraffin, 2 - no paraffin)

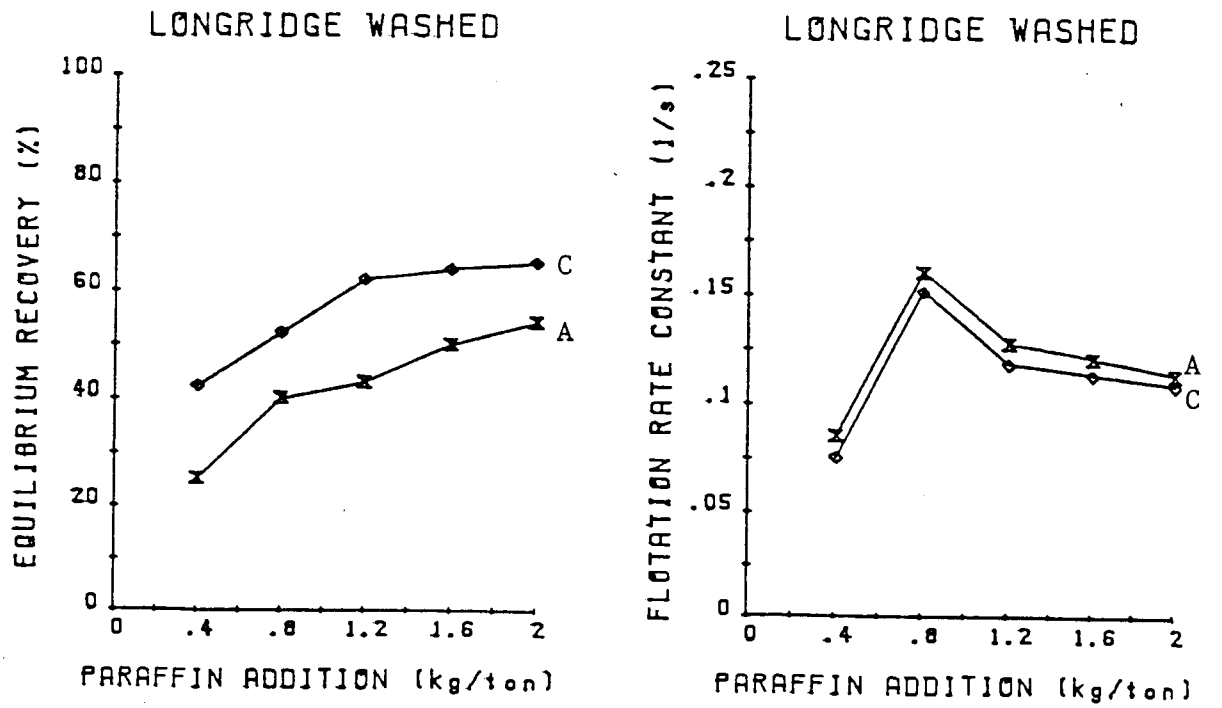

Figure 5.14 The effect of paraffin dosage on the kinetic model constants for the Longridge washed coal sample (C - coal, A - ash).

dosage for efficient flotation should be determined for each coal, and that this is a highly complex function of rank, maceral composition and surface chemistry.

5.2 SECOND STAGE OF THE INVESTIGATION

From the results of Section 5.1 it was possible to identify a narrow range of values containing the optimal flotation conditions. The aim of this part of the work was to apply these conditions to the raw coals in order to probe further into the optimal flotation conditions and to investigate maceral and microlithotype floatability.

5.2.1 Floatability of the Raw Coals

The results of Section 5.1 showed that the following flotation conditions gave high recoveries of coal at acceptably low concentrate ashes for all the four washed coal samples.

Impeller speed	1200 rpm
Particle size	95 per cent minus 300 μm
Paraffin addition	1200 g/t
Aeration rate	4 ℓ/min.

(The aeration rate was set at an intermediate value of 4 ℓ/min to provide a compromise between high or low flotation rate constants).

Another series of tests were carried out, this time with the raw coal samples, at the above flotation conditions and 5 and 10 per cent solids concentration, using 10 and 15 mg/ℓ MIBC addition. Each set of four tests on each raw coal sample was aimed at further identifying optimal flotation conditions. The detailed results are presented in Appendix F and summarised in Table 5.9 and Figures 5.15 to 5.18.

TABLE 5.9

Klimpel Model Constants for the Batch Flotation Tests Carried Out on the Raw Coal Samples

Note: The flotation conditions for the Batch Tests:

1 10 mg MIBC/ℓ : 10 per cent solids
2 15 mg MIBC/ℓ : 10 per cent solids
3 10 mg MIBC/ℓ : 5 per cent solids
4 15 mg MIBC/ℓ : 5 per cent solids

Batch Test	Coal Sample	Equilibrium Recovery (ϕ : %)		Flotation Rate ($k \times 10^3$: s^{-1})	
		Coal	Gangue	Coal	Gangue
1	Grootegeluk	44,35	10,38	77,1	82,8
2		60,26	21,11	104,4	109,8
3		28,3	4,18	59,3	56,5
4		40,52	7,54	102,2	106,3
1	Greenside	77,84	44,75	55,3	57,6
2		89,93	51,84	66,4	70,9
3		43,17	18,9	69,2	71,0
4		67,8	35,91	162,0	174,7
1	Nonsana	46,96	16,18	100,2	96,2
2		57,43	20,42	98,9	92,7
3		41,62	13,26	28,71	344,8
4		42,54	13,18	306,0	363,4
1	Longridge	74,38	36,50	59,4	57,5
2		82,83	43,6	64,1	38,5
3		62,18	29,28	150,1	167,4
4		71,27	35,43	142,4	137,6

Note: *Figures 5.15 to 5.18* show the effects of Percent Solids and MIBC Concentration on the recovery-ash profiles for the raw coal samples.

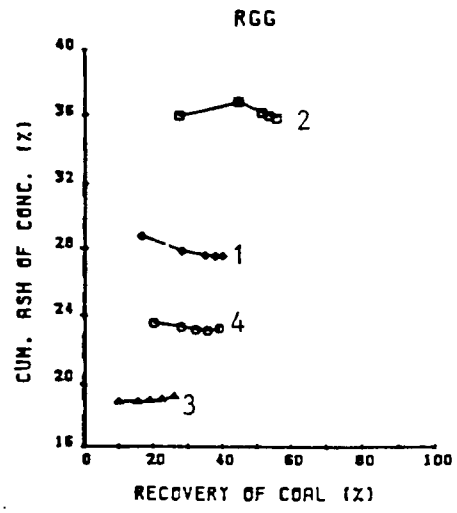

Figure 5.15 Grootegeluk raw coal.

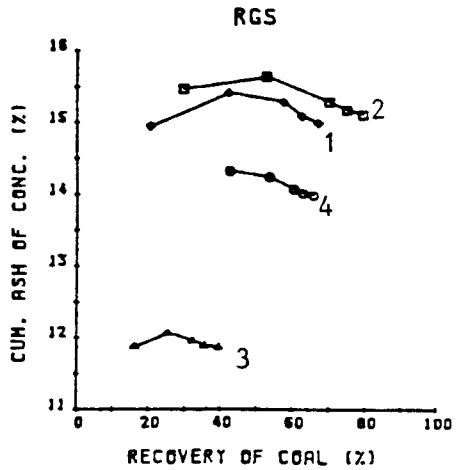

Figure 5.16 Greenside raw coal.

Legend	MIBC (mg/ℓ)	Percent Solids
1	10	10
2	15	10
3	10	5
4	15	5

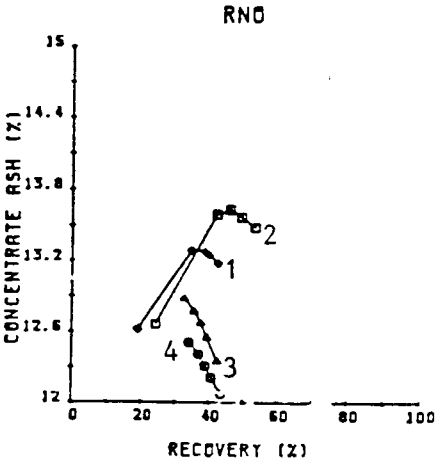

Figure 5.17 Nonsana raw coal.

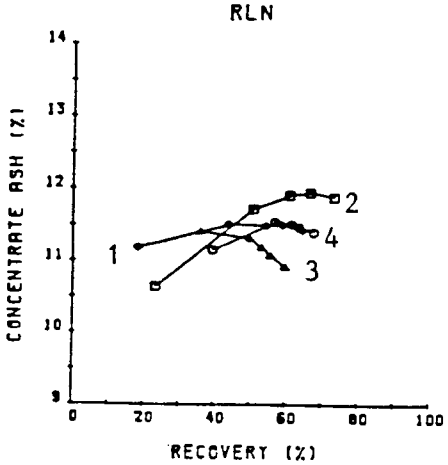

Figure 5.18 Longridge raw coal.

(a) Grootegeluk Sample

As may be seen from Table 5.9 the effect of higher pulp densities and frother dosage was to increase the recovery of coal and gangue. At the lower MIBC addition (10 mg/ℓ) an increase in pulp density increased k, but at the higher frother dosage (15 mg/ℓ) there was no significant change in the rate constant.

Increasing the MIBC dosage or the pulp density increased the ash content of the concentrate (Figure 5.15).

(b) Greenside Sample

The effect of the higher pulp density was to increase the recovery and decrease the rate constant for both coal and gangue (Table 5.9). This produced a higher ash concentrate at both levels of MIBC addition (Figure 5.16).

Increasing the MIBC dosage increased both the recovery and flotation rate for the coal and gangue (Table 5.9) as well as the concentrate ash (Figure 5.16).

(c) Nonsana Sample

Increasing the pulp density from 5 to 10 per cent solids decreased k but increased the recovery of coal and gangue. The MIBC dosage showed no significant effect on the rate constant at either pulp density, or the recovery at lower pulp density. However there was a higher recovery at the upper levels of both factors (Table 5.9).

Neither of these two factors had any

significant effect on the concentrate ash (Figure 5.17).

(d) Longridge Sample

Changing the pulp density and MIBC dosage had no significant effect on the concentrate ash (Figure 5.18). However the higher levels of these factors decreased the rate constant and increased the recoveries of the coal and gangue.

The effect of changing the MIBC dosage and pulp density on the flotation of the raw coals was similar to that found for the washed coals. Although the coals exhibiting the same rank behaved similarly, the overall effect of the parameter changes was the same for all the samples (Table 5.10).

However the effect of changing the pulp density on the concentrate ashes was much more marked for the Greenside and Grootegeluk coal samples. This was thought to be the result of different degrees of liberation of the individual coal samples. Accordingly a series of float and sink tests were carried out on the 300 x 38 micron feed size fraction using zinc chloride as the dense medium. The results are presented in Figure 5.19. This shows that the Grootegeluk coal was less liberated than the other coal samples and had a much higher ash content. The results of this series of float and sink tests show similar trends to those carried out on the washed coal samples (Section 5.1.4.4).

Table 5.11 shows the ratio of the mass of water to concentrate recovered. The mean values and the standard deviations were higher for the bituminous coals than for the anthracite coals. This shows that there was less drainage of water from the concentrate at the higher pulp densities for the Greenside and Grootegeluk than for the

TABLE 5.10

The Relative Effects of Changing the Pulp Density and MIBC Dosage on the Kinetic Model Constants and Concentrate Ashes

(a) Effect of Increasing Pulp Density

Sample	Coal		Ash		Concentrate Ash
	ϕ	k	ϕ	k	
Grootegeluk	+	0	+	0	+
Greenside	+	-	+	-	+
Nonsana	+	-	+	-	0
Longridge	+	-	+	-	0

(b) Effect of Increasing MIBC Concentration

Sample	Coal		Ash		Concentrate Ash
	ϕ	k	ϕ	k	
Grootegeluk	+	0	+	0	+
Greenside	+	+	+	+	+
Nonsana	+	0	0	0	0
Longridge	+	-	+	-	0

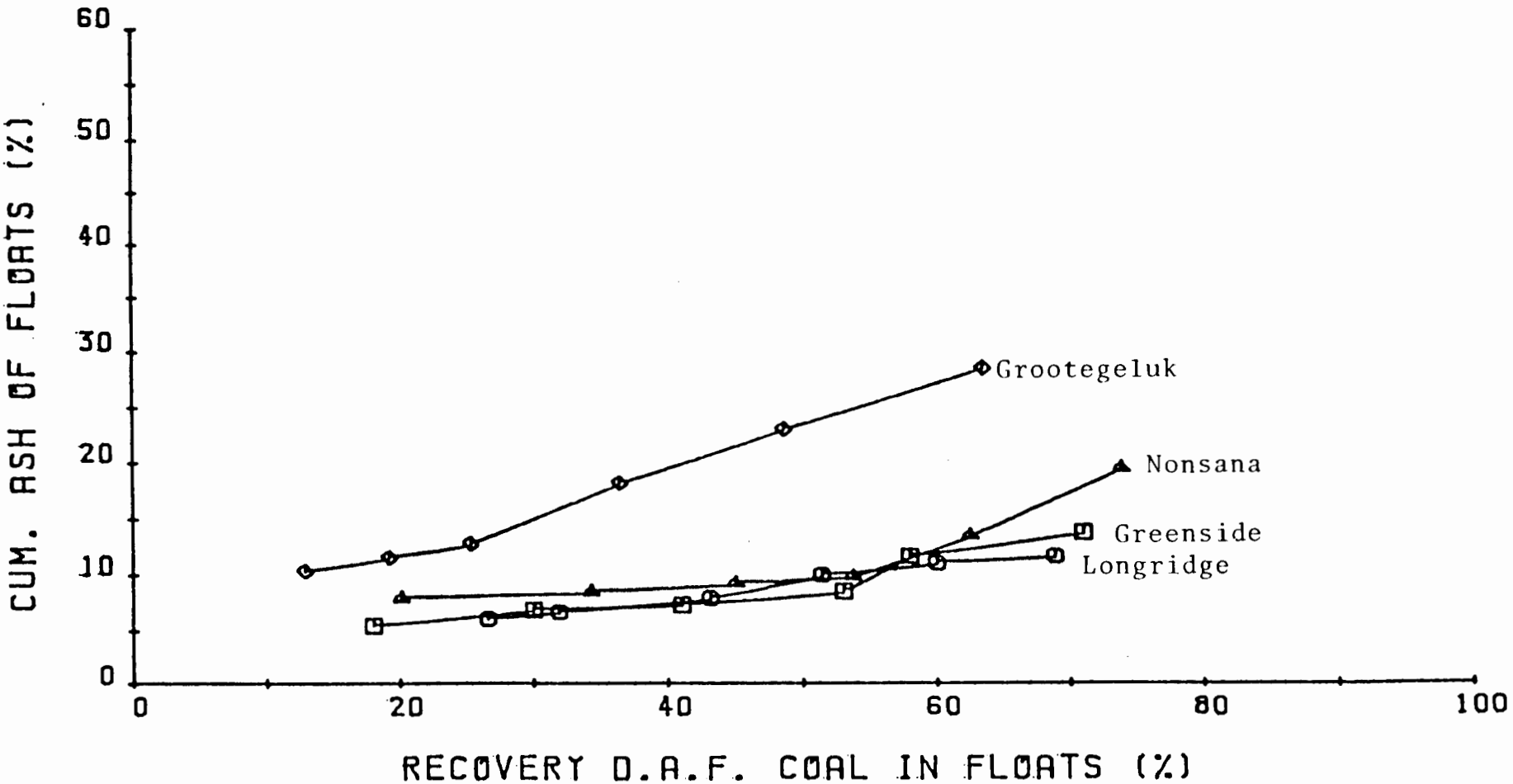

Figure 5.19 Results of the float and sink washability carried out on the +38 µm raw coal samples.

TABLE 5.11

Ratio of Mass of Water to Mass of Concentrate Recovered during the Batch Floats on the Raw Coal Samples

Test Number	Ratio of Mass of Water/Mass of Concentrate			
	Grootegeluk	Greenside	Nonsana	Longridge
1	1,903	2,011	1,378	1,577
2	1,687	1,964	1,398	1,685
3	2,486	2,509	1,340	1,612
4	2,085	2,709	1,304	1,735
Mean	2,04	2,298	1,355	1,632
Standard Deviation	0,339	0,368	0,042	0,071

Nonsana and Longridge samples. This also shows that the increase in concentrate ash at the higher pulp densities and MIBC concentrations could have been caused by entrainment.

5.2.2 Floatability of the Macerals

On the basis of the tests on the raw coal samples the following flotation conditions were chosen as those yielding high recoveries and low ashes (i.e. corresponding to batch test 2 in Table 5.9)

Impeller Speed	1200 rpm
Aeration Rate	4 ℓ/min
Paraffin Addition	1200 g/t
MIBC Addition	15 mg/ℓ
Particle Size	95 per cent minus 300 μm
Solid Concentration	10 per cent

A series of four identical batch flotation tests was carried out on each of the washed and raw coal samples. The feed, concentrates and tailings were sieved and the individual size fractions were ashed. They were also analysed for maceral and microlithotype composition. The petrographic analysis is presented in Appendix G.

Although the results of these tests were highly repeatable they are not directly comparable with the results of the earlier tests because the mill speed was changed when the mill was serviced. This yielded a different feed size distribution which was not noticed until after the results had been analysed. However the ash distribution did not vary significantly.

Figures 5.20 to 5.27 show the mass and ash distribution of the concentrates, feed and tailings with respect to particle size for the four washed and raw coal samples.

(a) Grootegeluk Samples

The ash of the raw and washed coal samples was higher in the finest and coarsest particle sizes of the feed and tailings but was evenly distributed through the concentrates. At longer flotation times the concentrates contained greater proportions of fine particles (Figures 5.20 and 5.21).

(b) Greenside Samples

Although the ash component was evenly distributed through the feed and concentrates, the intermediate size fractions contained lower ash material. The finest size fraction of the tailings exhibited the highest ash content. At longer residence times the concentrates contained greater proportions of fine particles (Figures 5.22 and 5.23).

(c) Nonsana Samples

The lowest ash material for both raw and washed coal samples was found in the coarsest size fractions and was evenly distributed through the range of concentrate particle sizes (Figures 5.24 and 5.25). The concentrates contained higher proportions of coarse material than the feed.

(d) Longridge Samples

In the washed coal sample feed, the -38 micron material exhibited the highest ash content; however the gangue was evenly distributed though the range of particle sizes of the raw coal feed.

The different particle size ranges of the concentrates showed very little ash variation, although the finer particles (-53 microns) of the tailings had a high ash content. Except for the final concentrate, the concentrates contained larger proportions of coarse material than the feed (Figures 5.26 and 5.27).

Panopoulous and King (1983) working on Durnacol and Landau naturally arising fines, reported that the ash content increased as the particles became smaller. In the present work, the uniform nature of the ash distribution with particle size may be accounted for by the method by which the samples were prepared.

The kinetic model constants, for each particle size range, are plotted in Figures 5.28 to 5.34.

Except for the Grootegeluk raw coal sample the recovery of coal is generally not affected by particle size, but the recovery of ash decreases in the finer sizes (-75 microns). In the -75 micron fraction the Grootegeluk raw coal sample exhibits a higher recovery of coal and ash; these might be related phenomena (Figures 5.28 to 5.31).

The flotation rate constant decreased with particle size probably because as the particles became smaller the surface area per unit mass of coal increases and hence the mass of coal per bubble decreases (Figures 5.32 to 5.35).

The results of the petrographic analyses on the eight batch floats are summarised in Tables 5.12 to 5.15. Figures 5.36 to 5.39 show the recovery-time profiles of the coal, ash and various petrographic components.

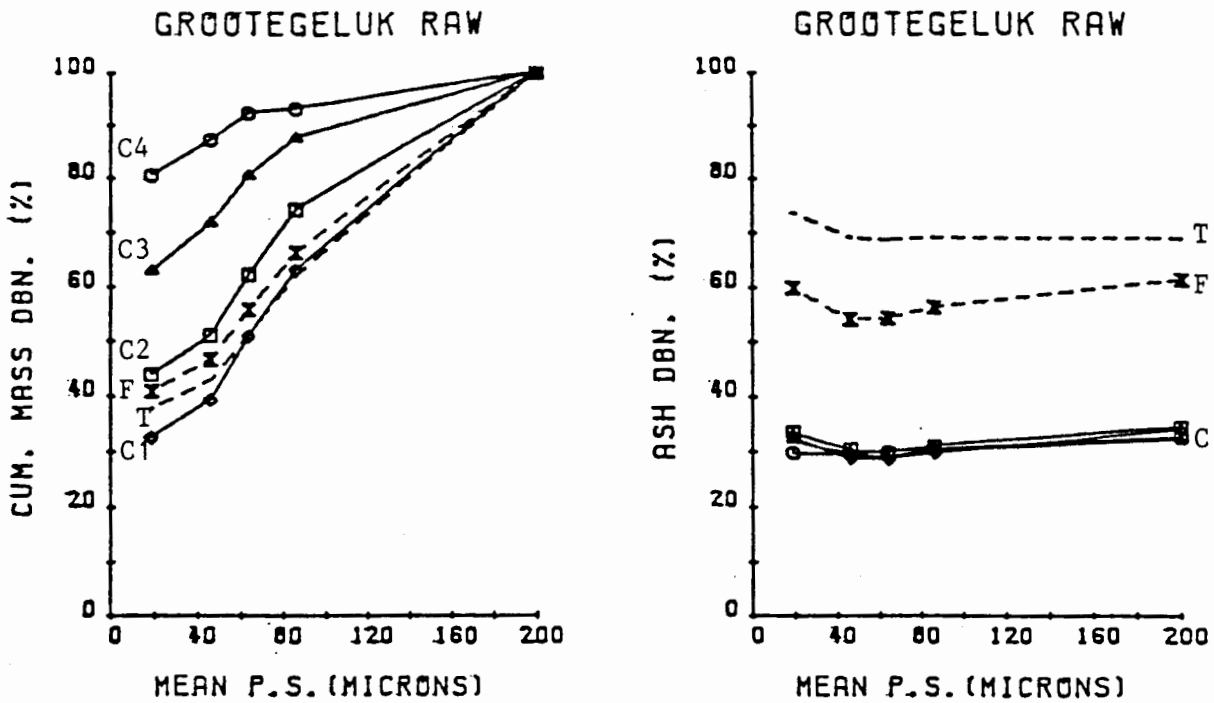

Figure 5.20 Sieve analysis of feed, concentrate and tailing samples for the petrographic batch floats on the Grootegeluk raw coal sample.

Legend	
F	Feed
T	Tailings
C	Concentrate

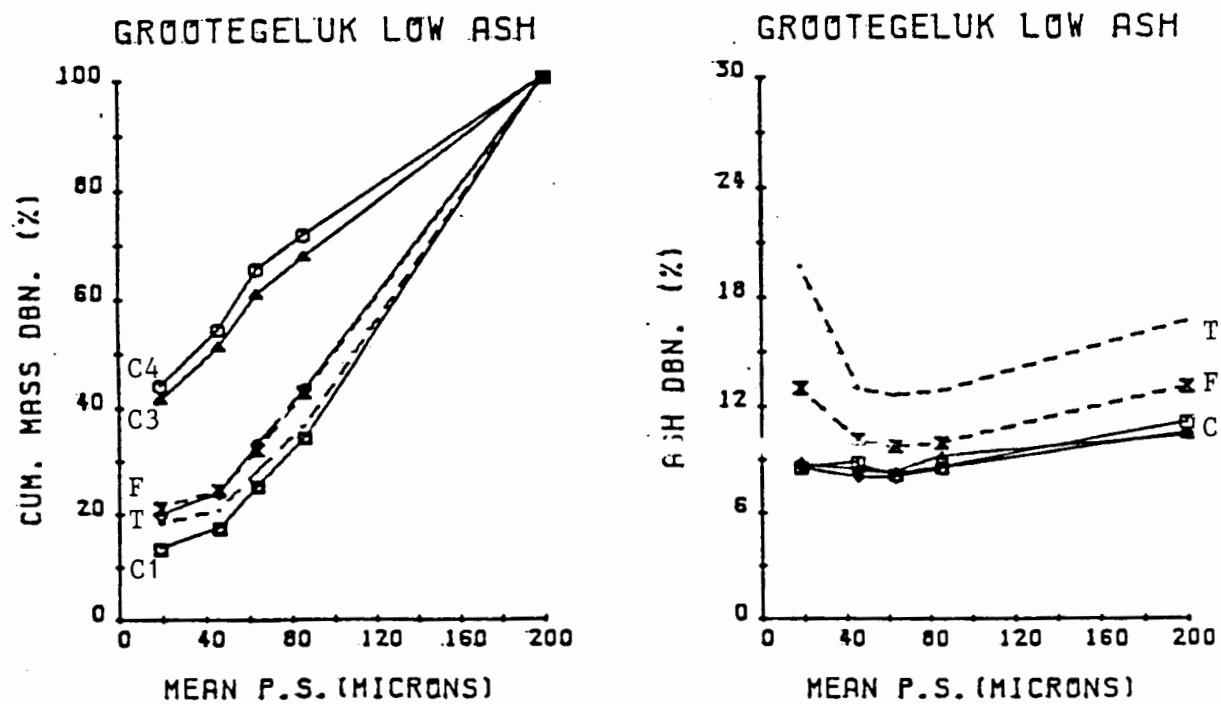

Figure 5.21 Sieve analysis of the feed, concentrate and tailing samples for the petrographic floats on the Grootegeluk washed coal sample.

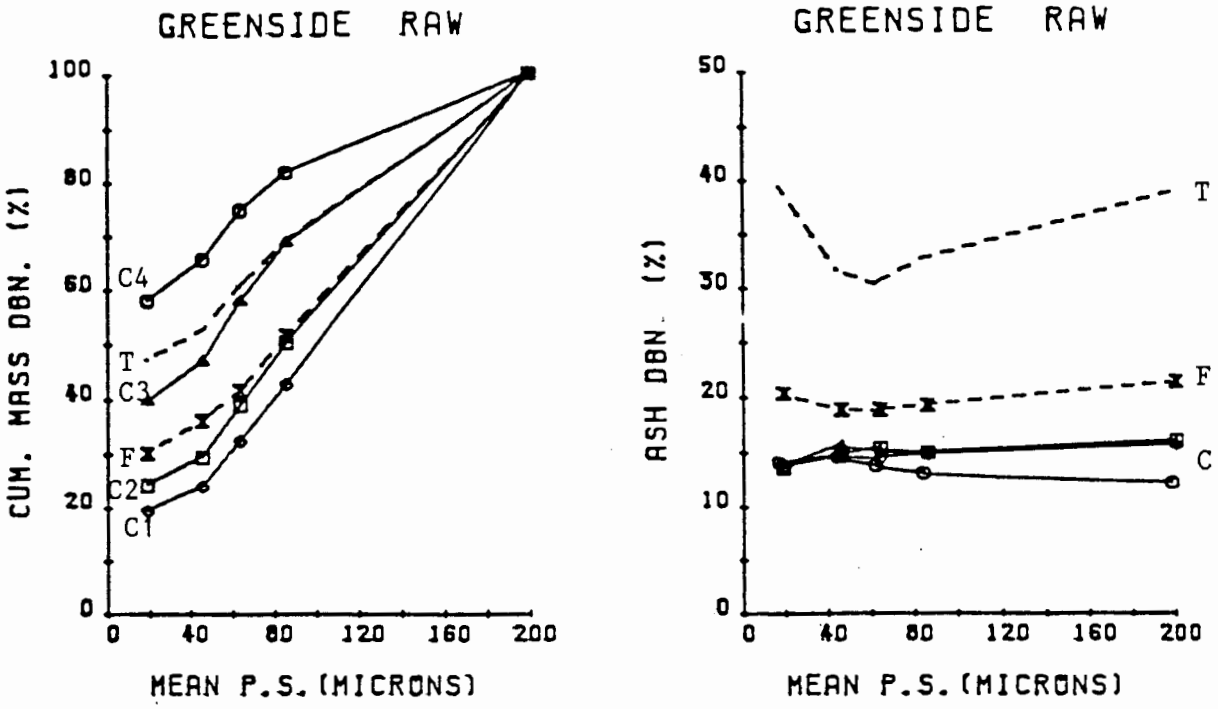

Figure 5.22 Sieve analysis of the feed, concentrate and tailings for the petrographic floats on the Greenside raw coal sample.

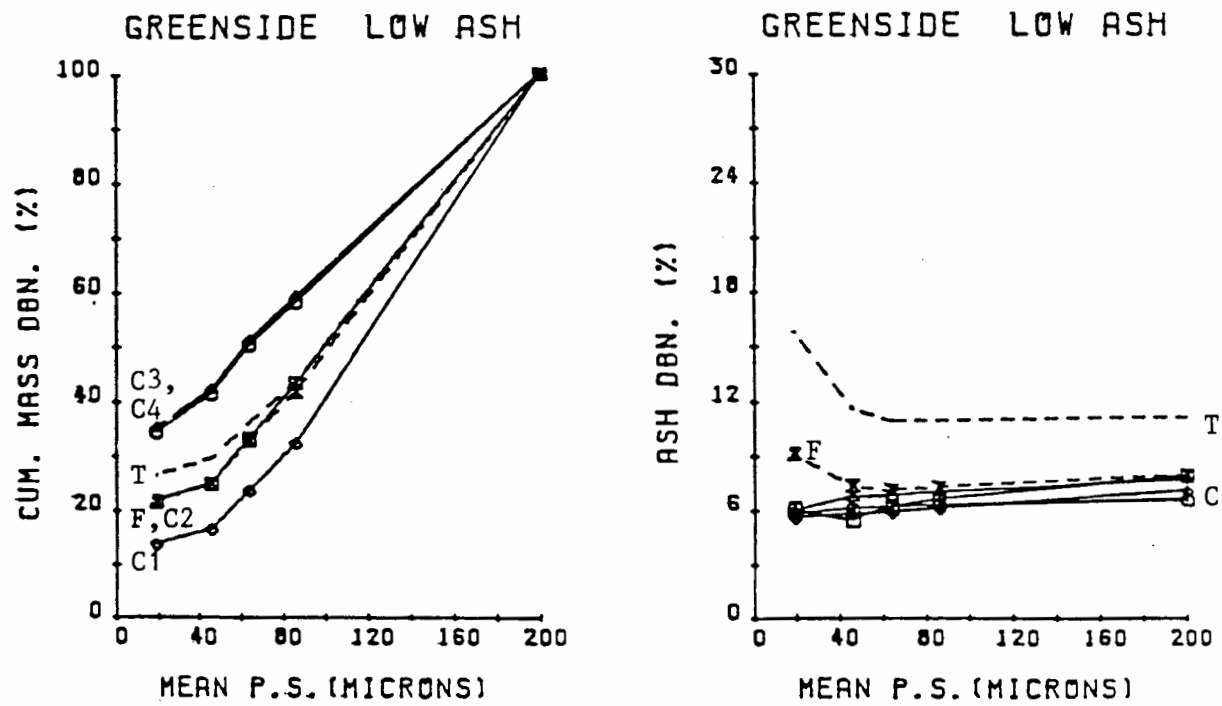

Figure 5.23 Sieve analysis of the feed, concentrate and tailings for the petrographic floats on the Greenside washed coal sample.

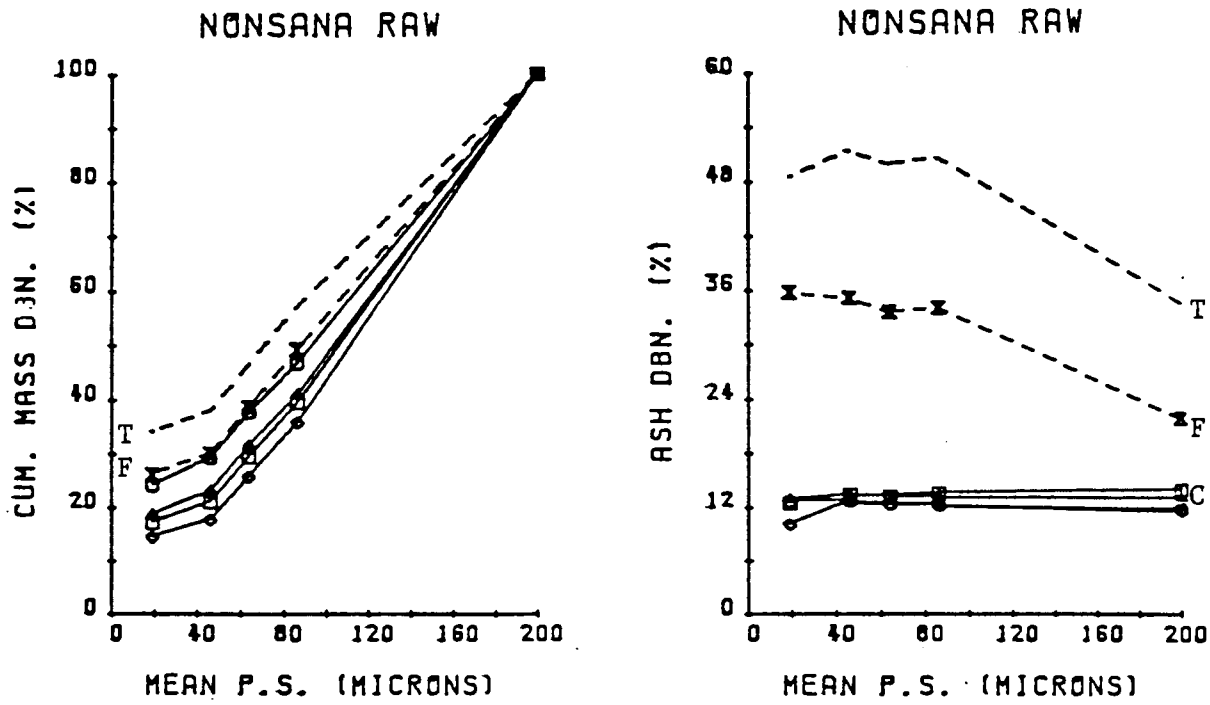

Figure 5.24 Sieve analysis of the feed, concentrates and tailings for the petrographic floats on the Nonsana raw coal sample.

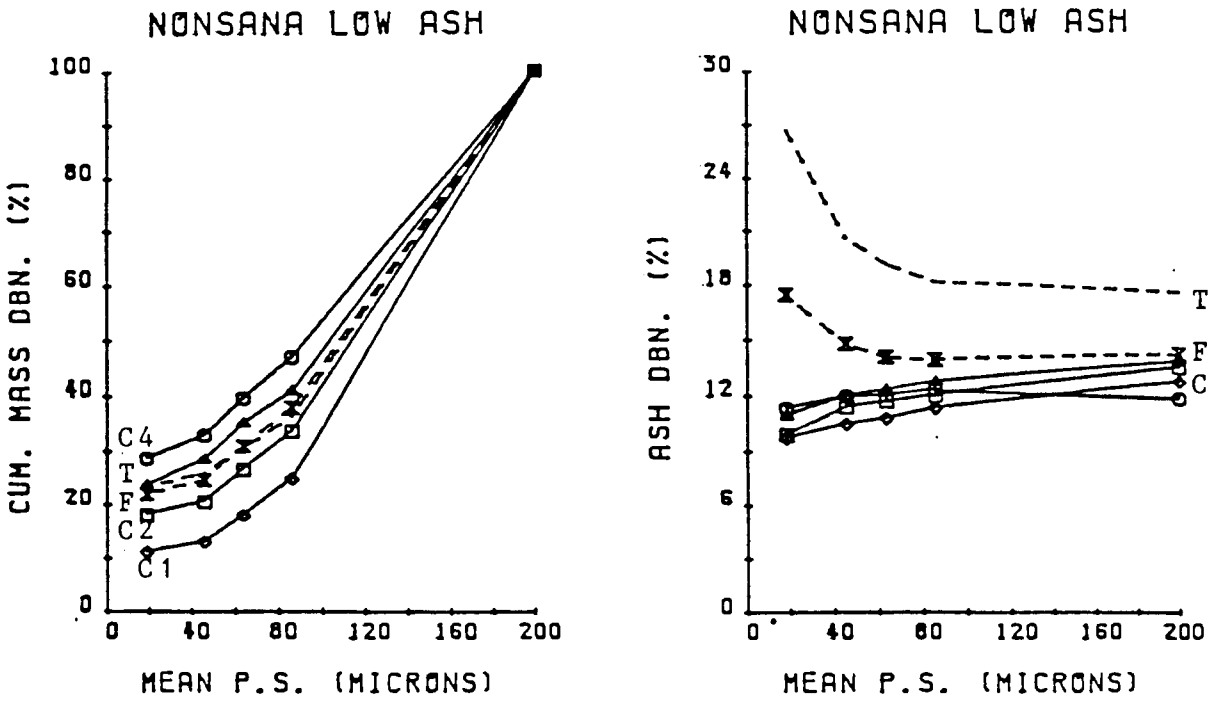

Figure 5.25 Sieve analysis of the feed, concentrates and tailings for the petrographic floats on the Nonsana washed coal sample.

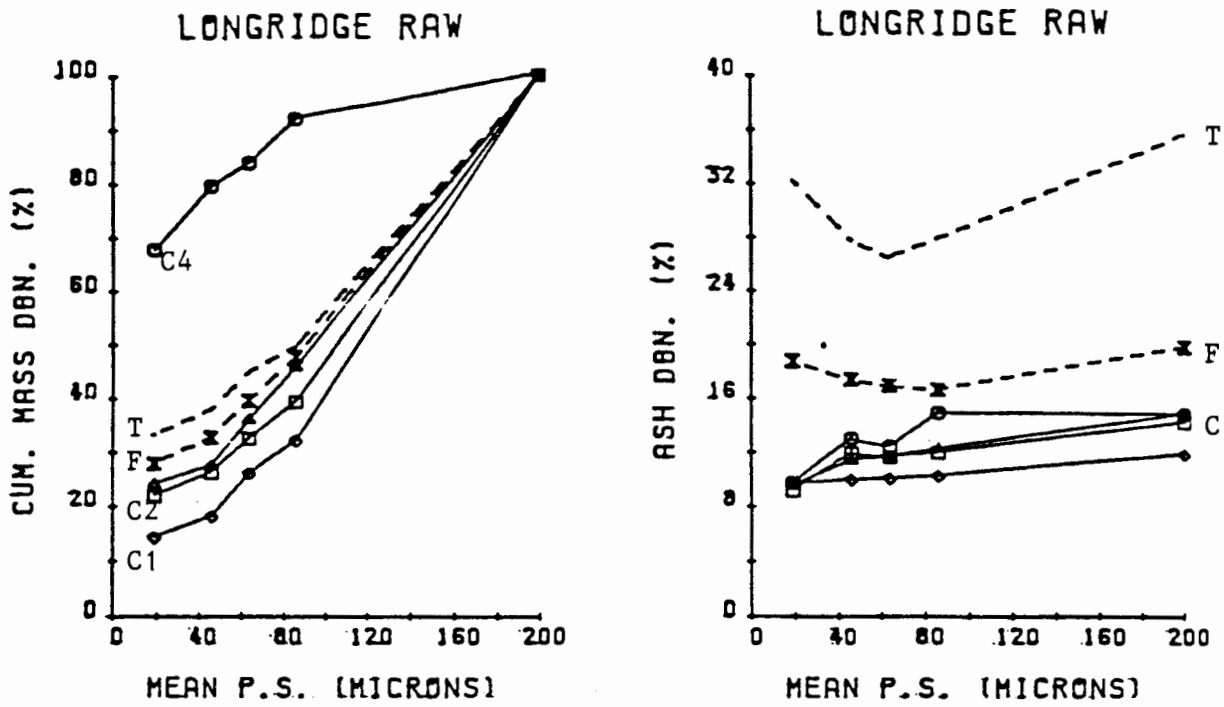

Figure 5.26 Sieve analysis of the feed, concentrates and tailings for the petrographic floats on the Longridge raw coal sample.

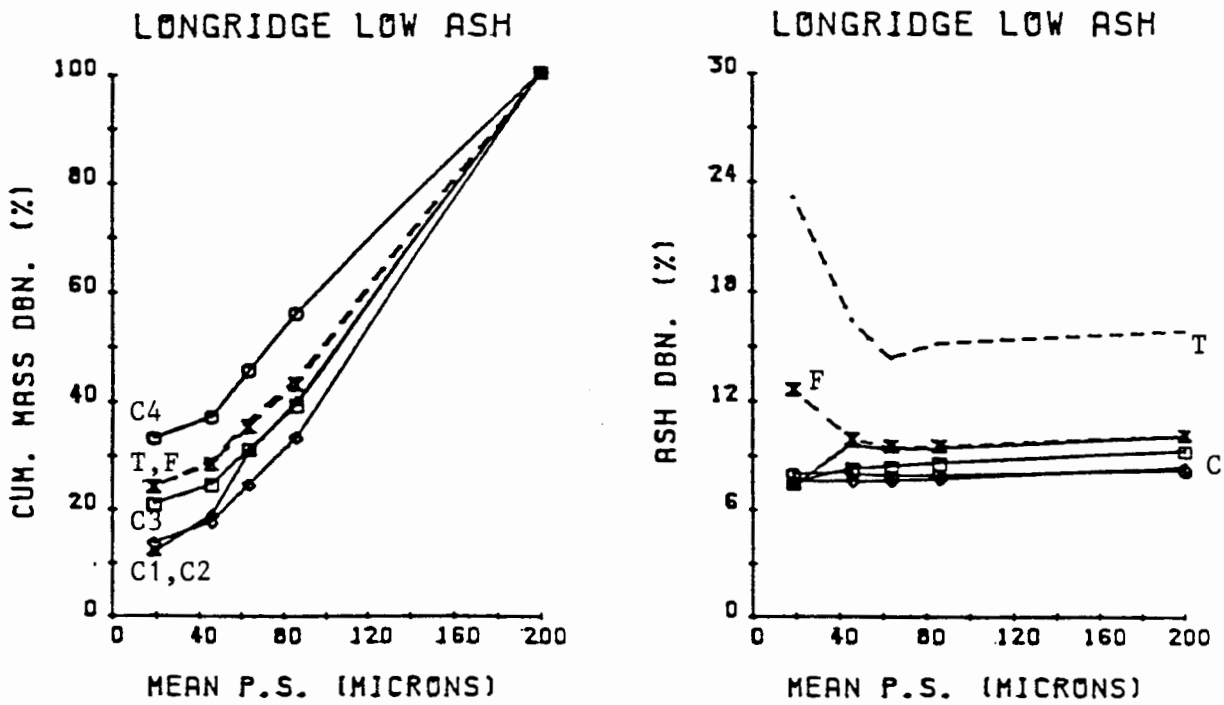

Figure 5.27 Sieve analysis of the feed, concentrates and tailings for the petrographic floats on the Longridge washed coal sample.

Note: *Figures 5.28 to 5.31* show the flotation rate constant calculated for the individual size fractions of the ash and coal constituents in both the raw and washed samples.

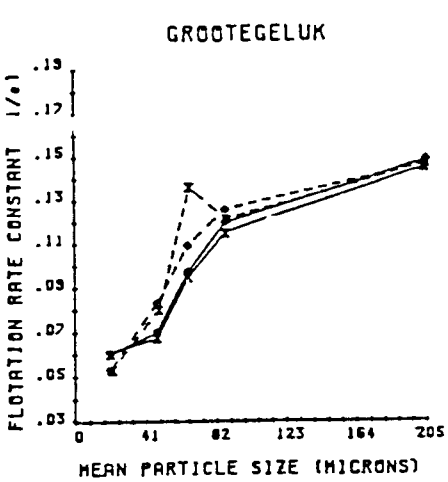

Figure 5.28 Grootegeluk coal samples

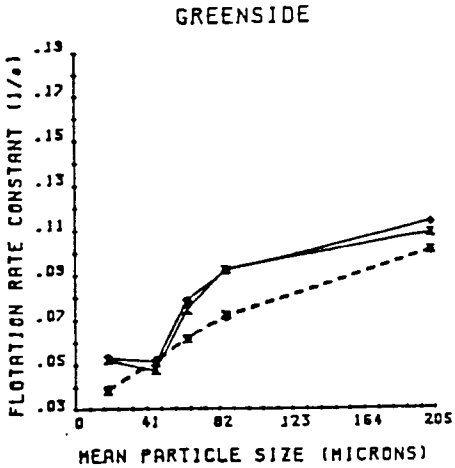

Figure 5.29 Greenside coal samples

Legend	
---x---	Ash : Raw coal sample
---◇---	Coal : Raw coal sample
——x——	Ash : Washed coal sample
——◇——	Coal : Washed coal sample

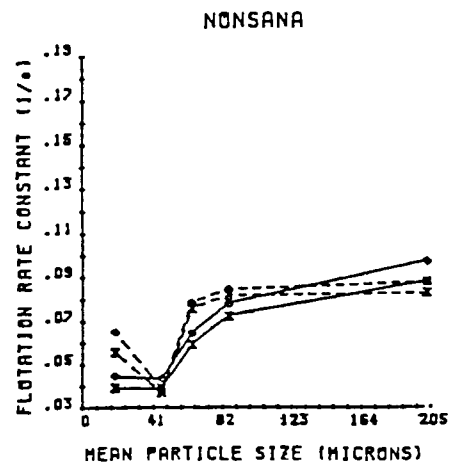

Figure 5.30 Nonsana coal samples

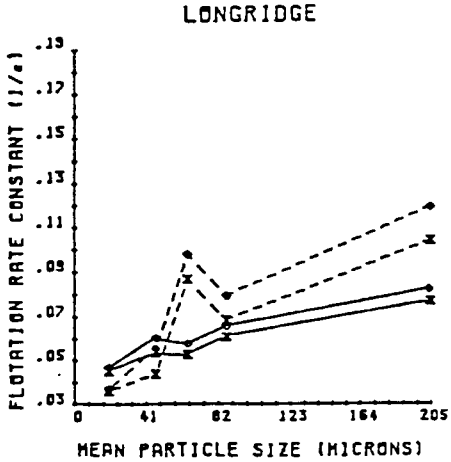

Figure 5.31 Longridge coal samples

Note: *Figures 5.32 to 5.35* show the equilibrium recovery calculated for the individual size fractions of the ash and coal constituents in both the raw and washed samples.

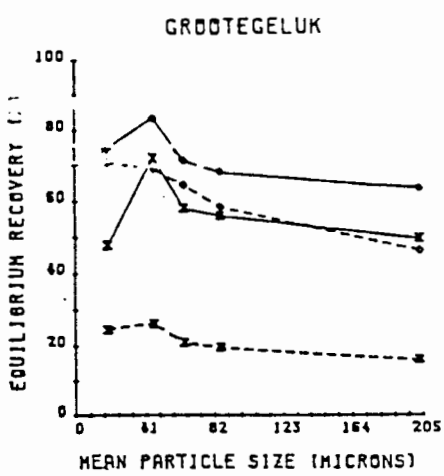

Figure 5.32 Grootegeluk samples

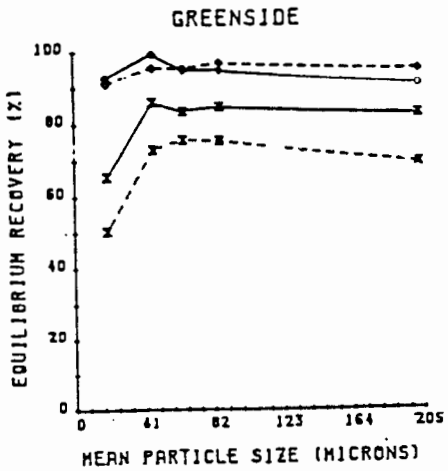

Figure 5.33 Greenside samples

Legend	
---⧖---	Ash : Raw sample
---◇---	Coal : Raw sample
——⧖——	Ash : Washed sample
——◆——	Coal : Washed sample

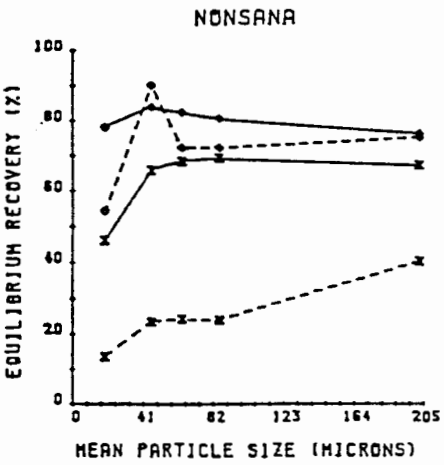

Figure 5.34 Nonsana samples

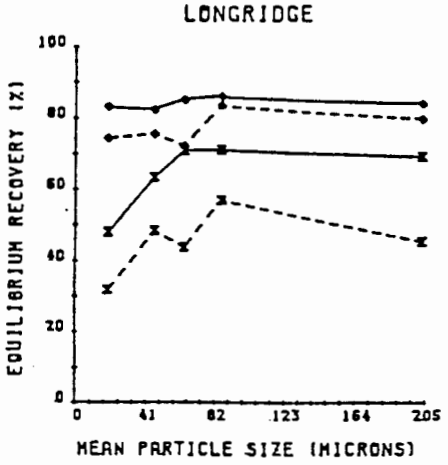

Figure 5.35 Longridge samples

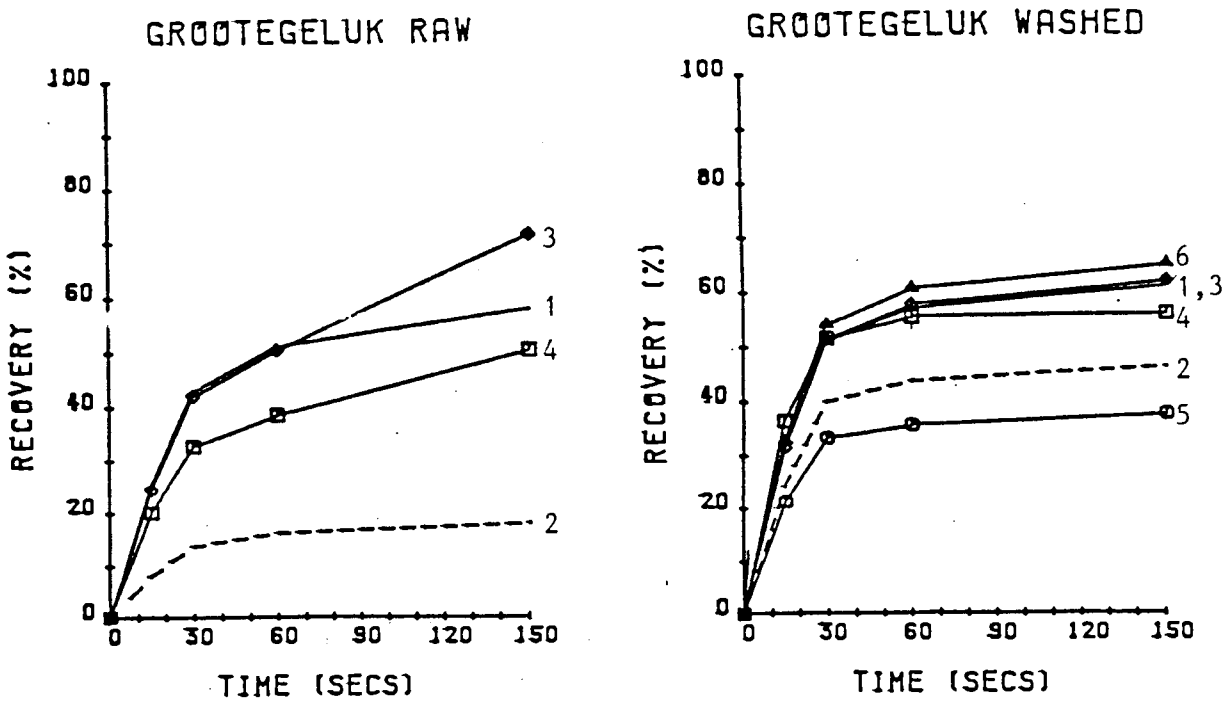

Figure 5.36 Recovery-time profiles of the macerals and microlithotypes for the Grootegeluk coal samples

Legend			
1	Coal	5	Vitrites
2	Ash	6	Inertites
3	Vitrinite	7	Intermediates
4	Inertinite		

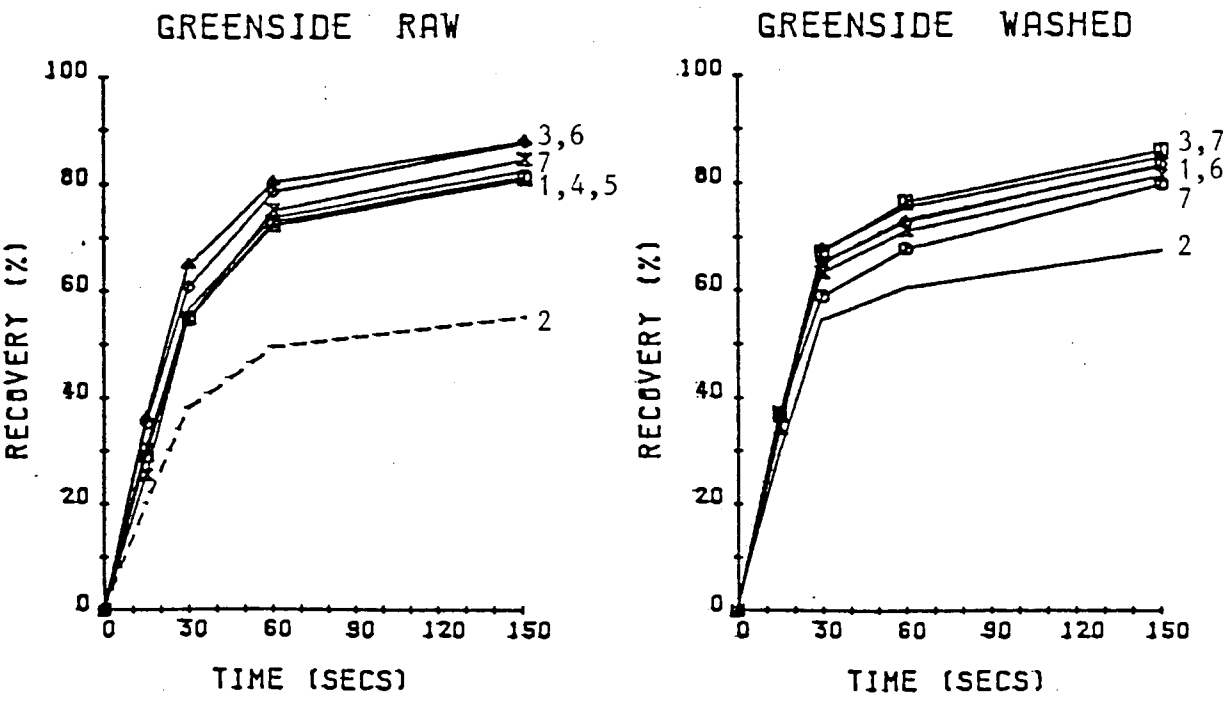

Figure 5.37 Recovery-time profiles for the macerals and microlithotypes for the Greenside coal samples

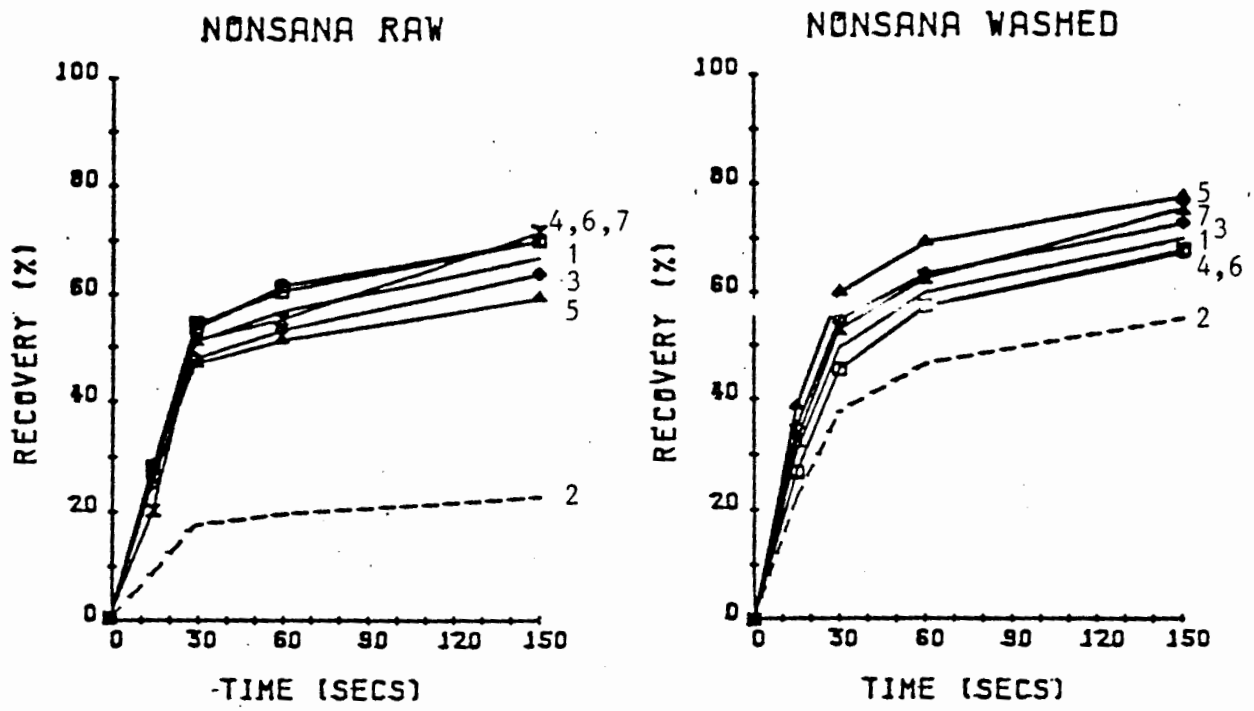

Figure 5.38 Recovery-time profile of the maceral and microlithotypes for the Nonsana coal sample.

Legend			
1	Coal	5	Vitrite
2	Ash	6	Inertite
3	Vitrinite	7	Intermediates
4	Inertinite		

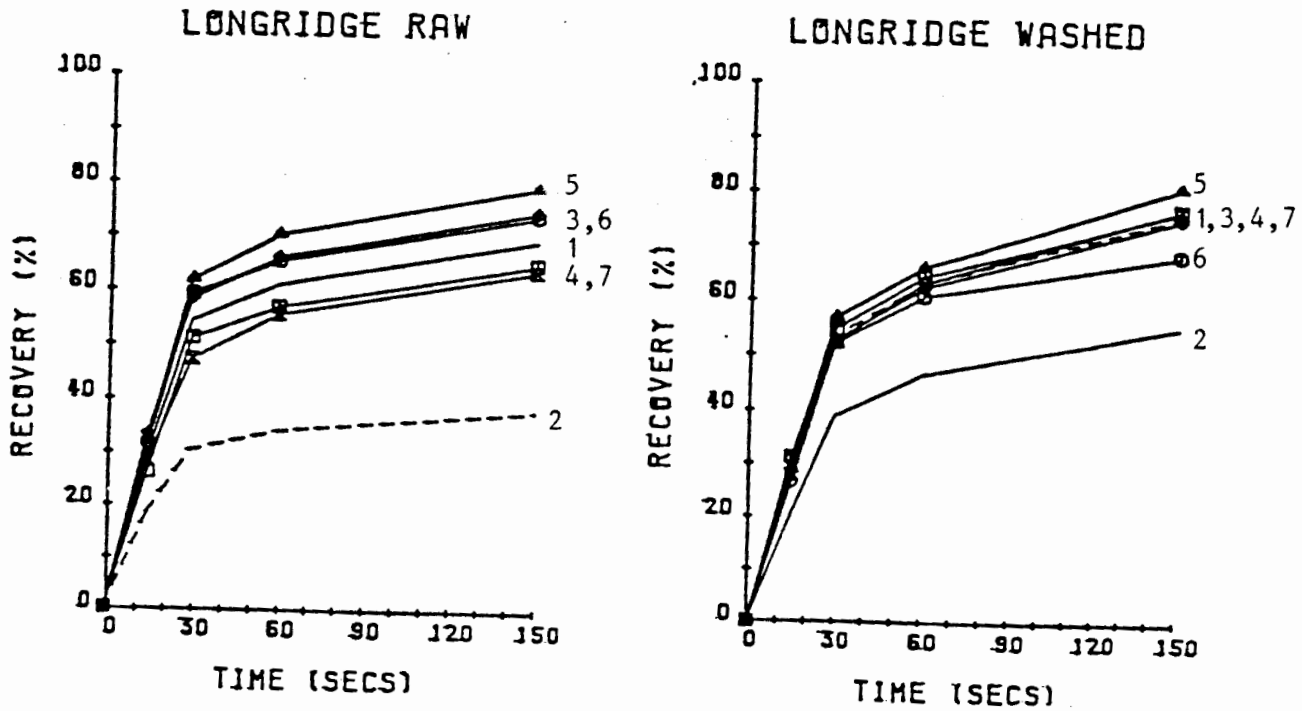

Figure 5.39 Recovery-time profile of the maceral and microlithotypes for the Longridge coal samples.

Although the Klimpel model constants were obtained using the data from only four concentrates, the difference between the predicted and experimental recoveries was very small. In some cases there was a large discrepancy between calculated and assayed feed grades, but this was because the analysis involved the conversion of two dimensional analysis (area) into three (volume), when calculating the relative amounts of the different organic components.

The limited accuracy with which the percentage of the various organic components can be determined using microscopic measurements, precludes the inclusion of data, for components constituting less than 10 per cent of the sample.

(a) Grootegeluk Coal Samples

The fine particle size of the third and fourth concentrates obtained from the raw coal sample did not permit microlithotype analysis. Table 5.12 shows that vitrinite floats better than inertinite and is beneficiated into the concentrate. The vitrites and intermediates behave similarly to the vitrinite.

The washed coal sample was mainly composed of reactive macerals (vitrinite 92,7 per cent) and hence the relative floatability and separation of the macerals could not be determined with any accuracy.

(b) Greenside Coal Samples

The reactive macerals and microlithotypes exhibited a better floatability than the inerts and were slightly beneficiated into the concentrate. However the separation

of macerals was poor and they showed very little change in their rate constants and equilibrium recoveries.

(c) Nonsana Coal Samples

Under the flotation conditions used in the batch tests, a high degree of selectivity for the reactive or inert macerals was not obtained, although there was a higher percentage of reactives in the concentrate than the feed.

(d) Longridge Coal Samples

Again there was no highly significant upgrading of the macerals or microlithotypes into the concentrate. Both reactive and inert species exhibited similar f.r.c's and equilibrium recoveries.

There is no significant separation of, or selectivity for, any specific maceral or microlithotypes under the froth flotation conditions chosen for these batch tests. This is probably the result of insufficient liberation of the macerals, or non-specific adsorption of the collector.

This and earlier results show that there are two highly important areas of coal flotation that ought to be investigated in depth. They are different chemicals to provide selectivity for coal and also for the individual macerals; and to identify the liberation size for the coal gangue and individual macerals.

TABLE 5.12

Recovery - Grade of Macerals and Microlithotypes for the Grootegeluk Coal Sample

Valuable Component	Cum Conc. Grade (%)	Equili-brium Recovery (%)	Flotation Rate Constant (s^{-1})	Calc Feed Grade (%)	Assayed Feed Grade (%)	Assayed Tailings Grade (%)
(a) Raw Coal						
Coal		63,44	0,0848			
Ash	32,0	19,71	0,0877	60,2	60,2	74,6
Vitrinite	68,9	79,87	0,0509	60,8	47,5	47,0
Inertinite	26,8	54,4	0,0668	33,6	45,5	45,0
Vitrites	54,3				42,0	13,3
Intermediates	15,8				4,0	1,7
Inertites	15,2				15,5	1,7
(b) Washed Coals						
Coal		66,67	0,1111			
Ash	9,7	50,65	0,1181	12,4	12,4	16,3
Vitrinite	93,9	67,55	0,1099	92,7	91,0	90,7
Inertinite					5,3	5,3
Vitrites	91,3	70,94	0,1092	86,0	89,5	77,5
Intermediates					1,0	0,5
Inertites					0,5	0,5

TABLE 5.13

Recovery-Grade of Macerals and Microlithotypes for the Longridge Coal Sample

Valuable Component	Cum Conc. Grade (%)	Equilibrium Recovery (%)	Flotation Rate Constant (s^{-1})	Calc Feed Grade (%)	Assayed Feed Grade (%)	Assayed Tailings Grade (%)
(a) Raw Coal						
Coal		75,55	0,0876			
Ash	12,6	40,86	0,1017	20,8		34,4
Vitrinite	48,8	80,91	0,0912	45,3	48,3	37,6
Inertinite	51,2	71,06	0,0844	54,7	51,7	62,4
Vitrites	33,7	87,26	0,0831	29,4	28,7	20,0
Intermediates	21,0	68,95	0,0860	22,9	6,0	27,0
Inertites	38,8	80,46	0,0903	36,3	50,5	31,0
(b) Washed Coals						
Coal		83,73	0,0707			
Ash	8,48	61,45	0,0699	11,26		18,9
Vitrinite	57,9	83,99	0,0668	58,3	62,0	59,8
Inertinite	42,1	83,24	0,0766	41,6	38,0	40,2
Vitrites	47,8	89,86	0,0669	44,7	31,3	35,0
Intermediates	29,4	85,58	0,0659	28,9	39,0	27,5
Inertites	20,5	74,71	0,0893	22,7	26,0	29,5

TABLE 5.14

Recovery-Grade of Macerals and Microlithotypes for the Greenside Coal Sample

Valuable Component	Cum Conc. Grade (%)	Equilibrium Recovery (%)	Flotation Rate Constant (s^{-1})	Calc Feed Grade (%)	Assayed Feed Grade (%)	Assayed Tailings Grade (%)
(a) Raw Coal						
Coal		93,58	0,0683			
Ash	15,2	62,65	0,0678	21,2		40,7
Vitrinite	33,9	98,35	0,0730	31,9	26,4	22,7
Inertinite	63,8	92,34	0,0658	65,0	74,6	70,6
Vitrites	24,4	98,22	0,0799	22,9	21,6	16,0
Intermediates	23,7	98,24	0,0569	22,1	23,0	20,5
Inertites	47,8	92,76	0,0665	48,4	51,7	51,0
(b) Washed Coals						
Coal		90,87	0,0888			
Ash	6,6	74,09	0,0918	8,0		14,3
Vitrinite	48,0	90,54	0,0911	47,8	57,6	46,7
Inertinite	47,1	94,49	0,0874	45,5	46,0	37,7
Vitrites	39,2	93,9	0,0843	38,4	37,7	34,7
Intermediates	39,0	87,97	0,0935	39,8	45,6	43,7
Inertites	14,2	85,27	0,0900	14,8	13,7	17,7

TABLE 5.15

Recovery-Grade of Macerals and Microlithotypes for the Nonsana Coal Sample

Valuable Component	Cum Conc. Grade (%)	Equilibrium Recovery (%)	Flotation Rate Constant (s^{-1})	Calc Feed Grade (%)	Assayed Feed Grade (%)	Assayed Tailings Grade (%)
(a) Raw Coal						
Coal		72,71	0,0804			
Ash	12,5	24,54	0,0786	30,1		49,8
Vitrinite	48,3	69,22	0,0779	50,5	50,8	55,0
Inertinite	51,7	76,27	0,0827	49,5	49,2	45,0
Vitrites	32,6	62,87	0,0999	36,8	39,8	45,0
Intermediates	25,6	79,87	0,0590	23,9	19,0	20,5
Inertites	37,2	77,73	0,0764	35,5	37,2	32,0
(b) Washed Coals						
Coal		74,54	0,0797			
Ash	12,4	60,97	0,0716	15,2		21,2
Vitrinite	45,5	78,24	0,0950	43,8	42,1	39,7
Inertinite	54,5	75,53	0,0688	56,2	57,9	60,3
Vitrites	26,8	83,58	0,1014	24,2	20,0	18,0
Intermediates	25,1	81,82	0,0789	23,2	23,5	19,0
Inertites	45,8	74,68	0,0707	47,7	53,0	52,0

CHAPTER 6

OVERALL DISCUSSION AND CONCLUSIONS

This chapter consolidates the findings of a broad investigation into the froth flotation of South African coals into a number of brief and salient points. From the results presented and discussed in Chapter 5, a number of factors are identified and discussed that warrant further study in an ongoing research programme to optimise the efficiency of fine coal beneficiation by froth flotation in South Africa.

6.1 EFFICIENCY OF FLOTATION

High process efficiencies may be obtained relative to the ideal float/sink washability curve, provided that the flotation parameters are controlled within narrow limits of the optimum conditions. This was found to be true for all but the Grootegeluk coal sample, for which it was shown that the collector concentration chosen for the tests was too low.

6.2 HYDROPHOBICITY

Natural hydrophobicity does not appear to correspond to maceral or microlithotype composition, rank, fixed carbon or moisture content, as has been proposed by a number of investigators, but appears to be a complex function of these variables. The order of increasing hydrophobicity was found to be Grootegeluk, Nonsana, Longridge and Greenside coal.

Flotation selectivity improves when a collector (paraffin) is used.

6.3 LIBERATION AND SELECTIVITY

Process inefficiencies are due to the combined effects of insufficient liberation and unselective process conditions. A significant feature of the results is that any change in the recovery and flotation rate constant for coal was generally accompanied by a corresponding change for the gangue.

6.4 MACERAL SEPARATION

There was no significant beneficiation of any of the macerals or microlithotypes at the flotation conditions used in the batch tests. This may be the result of insufficient liberation of the macerals and/or unselective adsorption of the collector on the individual organic coal components.

6.5 MAIN EFFECTS

6.5.1 MIBC Addition

At higher MIBC dosages the recovery and flotation rate constant of coal and gangue increased, approaching a maximum value. The concentrate ash also increased.

The primary effect of MIBC is to lower the surface tension of the pulp thus producing smaller and more numerous bubbles. This would increase the probability of fruitful particle/bubble collision and therefore the flotation rate.

The increase in recovery is thought to be a pseudo-effect because at the lower dosages, MIBC addition (i.e. froth stability) is the limiting factor.

6.5.2 Paraffin Addition

Paraffin induced hydrophobicity on the Grootegeluk sample and increased the natural hydrophobicity of the other coals. It also increased selectivity, when added, indicating that there was some degree of selective adsorption although at the higher levels selectivity decreased.

Although oily collectors decrease surface tension, they also increase the proportion of fast floating species in the pulp and these result in a larger flotation rate constant.

6.5.3 Aeration Rate

The higher aeration rate increased the flotation rate constant because of the higher probability of particle/bubble contact. However it has no significant effect on the recovery.

6.5.4 Pulp Density

Increasing the pulp density raised the recoveries and lowered the flotation rate constant. The increase in recovery was probably a pseudo-effect caused by entrainment. The drop in the flotation rate constant would be the result of particle crowding as the ratio of hydrophobic particles to bubbles increases.

6.5.5 Particle Size

The flotation rate constant of the constituent particle size fractions decreased with mean size because as the particles become smaller, the ratio of mass of coal recovered to bubble surface area decreases.

Except for the raw Grootegeluk coal sample, the recovery of coal does not vary with particle size. But

the recovery of ash is lower at the smaller particle sizes indicating improved liberation of coal and gangue.

6.6 INTERACTIONS

The interactions between some of the factors are as important as the factors themselves. The magnitude of the interactions was dependant on the coal sample and the same interaction might cause the measured response to increase or decrease depending on where on the response surface the flotation conditions were situated.

The combined factor of paraffin addition and impeller speed exhibited the most significant interactions resulting in increased recovery and a larger flotation rate constant.

6.7 RAW VERSUS WASHED COALS

For each coal, the raw and washed samples exhibited very similar flotation characteristics, and the recovery of ash was lower for the raw coal samples because the proportion of liberated gangue increased.

Nonsana and Longridge had similar liberation characteristics in both raw and washed coal samples. However the raw Grootegeluk and Greenside samples were less liberated than their corresponding washed coal samples.

This shows that for the same liberation, the efficiency of froth flotation is determined by the process conditions which are in turn controlled by a complex function of the rank and maceral composition of the coal.

CHAPTER 7

RECOMMENDATIONS FOR FURTHER WORK

The previous chapter has highlighted a number of areas in coal flotation which require a more in-depth investigation.

7.1 LIBERATION

The degree of liberation of the coal and gangue or the macerals will set limits on the efficiency of any beneficiation operation. This factor should be investigated and quantified before any detailed study of the viability of froth flotation of any coal is attempted.

7.2 COLLECTORS

Further research is required to identify collectors that selectively adsorb on the coal surface. The most efficient way of approaching this, would be to study and compare the adsorption kinetics of different chemicals with the results of batch flotation tests. This could be extended to comparing the adsorption of different collectors on the individual macerals and microlithotypes.

7.3 FROTHERS

A range of different frothers should be tested in batch floats because they exert the single most important method of changing the froth characteristics.

7.4 INTERACTIONS

The interactions between the parameters involved in flotation should be studied as these provide a valuable insight into the mechanisms involved in flotation.

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APPENDIX A

Sample Calculation of Recovery and Ash Content

Recovery (R) is defined as the mass of a component recovered in the concentrate divided by the mass of the component in the feed, expressed as a percentage.

Therefore

the recovery of coal (d.a.f.) $Rc = \frac{C\,(100 - c)\,100}{F\,(100 - f)}$

$$= \frac{C\,(100 - c)\,100}{C\,(100 - c) + T\,(100 - t)}$$

and the recovery of gangue $Rg = \frac{Cc\,100}{Ff}$

$$= \frac{cC\,100}{cC + tT}$$

where C, F and T refer to the mass of the concentrate, feed and tailing respectively

and c, f and t are the analysed percent ash of the concentrate, feed and tailing, respectively.

The percent ash of the concentrate $A_c = \frac{Cc\,100}{C}$

given a set of raw data, the cumulative recoveries and concentrate ashes can be calculated as follows:

Concentrate Number	Mass of Concentrate (g)	Analysed Ash of Concentrate (%)
1	33,3	8,99
2	17,6	9,56
3	7,5	7,94
4	2,6	7,11
5	1,8	7,22

The mass of ash in the concentrate is the product calculated of the mass of concentrate and the percent ash. The mass of coal in the concentrate is the difference between the mass of concentrate and the mass of ash in the concentrate.

Mass of Ash (g)	Mass of Coal (g)
2,99 = (33,3 x 8,49/100)	(33,3 - 2,99) = 30,31
1,68	15,92
0,60	6,90
0,18	2,42
0,13	1.67

and therefore the cumulative masses of ash and coal can be calculated.

Cumulative Mass in the Concentrate	
Ash (g)	Coal (g)
2,99	30,31
4,67	46,23
5,27	53,12
5,45	55,55
5,58	57,22

The mass of ash in the feed = $150 \times \frac{12,26}{100} = 18,399$

and the mass of coal in the feed = 150 - 18,39 = 131,61g

The cumulative recoveries of ash (gangue) and coal can be calculated.

Cumulative Recovery		Cumulative Ash of Concentrate (%)
Ash (%)	Coal (%)	
$\frac{2,99}{18,39} \times 100 = 16,28$	$\frac{30,31}{131,61} \times 100 = 23,03$	$\frac{2,99}{(2,99 + 30,31)} \times 100 = 8,99$
25,39	35,13	9,19
28,66	40,36	9,03
29,64	42,21	8,94
30,34	43,48	8,89

APPENDIX B

Summary of the results obtained during the preliminary f.d.e. on the Grootegeluk Washed Coal Sample.

TABLE B 1

Preliminary Factorial Designed Experiments - Grootegeluk Coal Sample

Flotation Rate of Coal (K_{Coal} : s^{-1})

		I1		I2	
		P1	P2	P1	P2
AE1	M1	0,0899	0,1052	0,0958	0,1082
	M2	0,1455	0,1314	0,1154	0,0975
AE2	M1	0,0921	0,0704	0,0590	0,0724
	M2	0,1178	0,1697	0,2046	0,1050

TABLE B 2

Preliminary Factorial Designed Experiments - Grootegeluk Coal Sample

Equilibrium Recovery of Coal (ϕ_{Coal} : %)

		I1		I2	
		P1	P2	P1	P2
AE1	M1	33,6	35,4	35,7	61,8
	M2	71,6	72,9	69,8	70,6
AE2	M1	33,5	40,5	58,4	47,6
	M2	49,2	73,2	68,8	45,1

TABLE B.3

Analysis of Variance of Flotation Rate of Grootegeluk Coal - Preliminary Factorial Designed Experiments

Treatment Combination	Effect x 10^4	Degrees of Freedom	Mean Square (x 10^4)	Significance Level (%)
I	-0,0080	1	2,57	5,0
P	-0,0075	1	2,27	-
AE	0,0003	1	0,00	-
M	0,0492	1	96,97	0,5
I x P	-0,0305	1	9,15	0,5
I x AE	0,0115	1	1,33	-
I x M	-0,0049	1	0,24	-
P x AE	-0,0129	1	1,67	-
P x M	-0,0248	1	6,14	0,5
AE x M	0,0531	1	28,22	0,5
Residual		5	12,31	
ERROR	-	32	6,57	

Significance Level (%)	Effect or Interaction Mean Square (x 10^4)
5,0	2,38
2,5	3,17
1,0	4,3
0,5	5,23

APPENDIX C

Summary of the Results obtained for the Factorial Designed Experiments Completed on the Four Washed Coal Samples Given as Klimpel Model Constants for Coal and Gangue.

Notation Used:

1. MIBC Addition
   - M1 5 mg/ℓ
   - M2 10 mg/ℓ
   - M3 15 mg/ℓ

2. Paraffin Addition/Impeller Speed
   - PA1 800 g/t : 900 rpm
   - PA2 1200 g/t : 1200 rpm

3. Particle Size
   - PS1 95 per cent minus 500 μm
   - PS2 95 per cent minus 300 μm

4. Aeration Rate
   - AE1 3 ℓ/min
   - AE2 6 ℓ/min

5. Solids Concentration
   - PD1 5 per cent
   - PD2 19 per cent

TABLE C.1

Grootegeluk Coal Sample

Equilibrium Recovery of Coal (ϕ_{Coal} : %)

			M1		M2		M3	
			PA1	PA2	PA1	PA2	PA1	PA2
PS1	AE1	PD1	5,3	15,3	21,0	22,0	37,9	43,2
		PD2	17,7	29,5	21,7	39,0	30,1	54,7
	AE2	PD1	6,9	20,5	16,7	27,0	24,1	42,7
		PD2	7,5	14,6	16,8	34,8	35,5	50,9
PS2	AE1	PD1	8,4	6,5	20,6	14,2	9,4	47,0
		PD2	17,8	21,2	20,3	34,5	26,0	42,2
	AE2	PD1	7,6	9,4	12,8	16,6	19,6	34,5
		PD2	8,0	14,5	15,8	35,9	38,2	51,3

TABLE C.2

Grootegeluk Coal Sample

Flotation Rate of Coal (K_{Coal} : s^{-1})

			M1		M2		M3	
			PA1	PA2	PA1	PA2	PA1	PA2
PS1	AE1	PD1	0,0236	0,0531	0,0308	0,0880	0,1295	0,1142
		PD2	0,0277	0,0231	0,0347	0,0708	0,0435	0,0609
	AE2	PD1	0,0347	0,0410	0,0536	0,0905	0,0665	0,1830
		PD2	0,0413	0,0793	0,0696	0,1710	0,0697	0,1752
PS2	AE1	PD1	0,0217	0,0311	0,0391	0,0760	0,0362	0,1134
		PD2	0,0270	0,0381	0,0427	0,0769	0,0599	0,0781
	AE2	PD1	0,0361	0,0530	0,0518	0,1307	0,0625	0,2491
		PD2	0,0419	0,0860	0,0650	0,1967	0,0771	0,1706

TABLE C.3

Grootegeluk Coal Sample

Equilibrium Recovery of Ash (ϕ_{Ash} : %)

			M1		M2		M3	
			PA1	PA2	PA1	PA2	PA1	PA2
PS1	AE1	PD1	3,0	6,4	0,2	10,9	25,0	27,4
		PD2	8,9	11,2	10,7	22,3	17,4	36,4
	AE2	PD1	3,3	13,3	8,2	16,5	15,8	30,0
		PD2	4,0	7,6	9,6	23,1	24,2	35,1
PS2	AE1	PD1	63,1	3,5	9,4	7,6	25,4	31,8
		PD2	9,3	10,9	10,2	17,5	15,9	26,4
	AE2	PD1	4,2	5,4	8,1	10,5	13,5	24,9
		PD2	4,2	7,5	8,9	23,9	24,8	35,2

TABLE C.4

Grootegeluk Coal Sample

Flotation Rate of Ash (K_{Ash} : s^{-1})

			M1		M2		M3	
			PA1	PA2	PA1	PA2	PA1	PA2
PS1	AE1	PD1	0,0241	0,0554	0,0306	0,0930	0,1431	0,1176
		PD2	0,0260	0,0329	0,0344	0,0729	0,0444	0,0643
	AE2	PD1	0,0277	0,0300	0,0533	0,0886	0,0669	0,1880
		PD2	0,0384	0,0754	0,0673	0,1845	0,0676	0,1800
PS2	AE1	PD1	0,0220	0,0292	0,0404	0,0760	0,0356	0,1188
		PD2	0,0270	0,0371	0,0416	0,0722	0,0582	0,0833
	AE2	PD1	0,0331	0,0506	0,0517	0,1266	0,0632	0,2473
		PD2	0,0414	0,0787	0,0617	0,2050	0,0767	0,1656

TABLE C.5

Greenside Coal Sample

Equilibrium Recovery of Coal (ϕ_{Coal} : %)

			M1		M2		M3	
			PA1	PA2	PA1	PA2	PA1	PA2
PS1	AE1	PD1	24,2	64,7	29,0	70,6	44,6	82,6
		PD2	52,6	86,0	72,8	89,5	83,2	91,2
	AE2	PD1	34,0	67,6	45,9	85,1	89,2	87,2
		PD2	48,1	69,9	69,4	79,8	81,5	85,0
PS2	AE1	PD1	31,3	49,4	58,2	81,7	75,2	74,1
		PD2	61,2	77,2	68,3	85,2	80,2	84,5
	AE2	PD1	38,4	51,4	69,9	80,2	75,2	78,3
		PD2	52,6	71,9	77,0	81,2	84,3	86,3

TABLE C.6

Greenside Coal Sample

Flotation Rate of Coal (K_{Coal} : s^{-1})

			M1		M2		M3	
			PA1	PA2	PA1	PA2	PA1	PA2
PS1	AE1	PD1	0,0668	0,1146	0,1032	0,1192	0,0809	0,1016
		PD2	0,0461	0,0597	0,0781	0,0828	0,0683	0,0797
	AE2	PD1	0,0323	0,1572	0,1154	0,2554	0,1640	0,2022
		PD2	0,1564	0,0857	0,1183	0,2106	0,1504	0,2393
PS2	AE1	PD1	0,1624	0,1138	0,1617	0,1624	0,0730	0,1851
		PD2	0,0736	0,1079	0,0921	0,0985	0,0798	0,1026
	AE2	PD1	0,1159	0,1553	0,1348	0,2549	0,1533	0,2867
		PD2	0,1207	0,0926	0,1437	0,2194	0,1784	0,2576

TABLE C.7

Greenside Coal Sample

Equilibrium Recovery of Ash (ϕ_{Ash} : %)

			M1		M2		M3	
			PA1	PA2	PA1	PA2	PA1	PA2
PS1	AE1	PD1	16,6	49,9	18,8	57,8	32,3	68,1
		PD2	37,5	67,7	56,3	71,1	69,1	76,2
	AE2	PD1	20,6	50,6	32,0	67,9	66,6	77,6
		PD2	29,7	59,4	60,7	78,9	79,6	89,5
PS2	AE1	PD1	22,7	33,3	36,1	62,1	43,7	58,2
		PD2	44,6	47,9	68,4	78,3	67,5	70,3
	AE2	PD1	27,0	41,4	47,5	70,2	66,7	65,3
		PD2	38,1	49,6	67,9	73,9	73,8	69,6

TABLE C.8

Greenside Coal Sample

Flotation Rate of Ash (K_{Ash} : s^{-1})

			M1		M2		M3	
			PA1	PA2	PA1	PA2	PA1	PA2
PS1	AE1	PD1	0,0747	0,1329	0,1170	0,1274	0,0826	0,1058
		PD2	0,0539	0,0639	0,0835	0,1197	0,0821	0,0835
	AE2	PD1	0,0781	0,1536	0,1238	0,2754	0,1540	0,2415
		PD2	0,2106	0,0874	0,1262	0,2116	0,1545	0,1505
PS2	AE1	PD1	0,1121	0,1346	0,1555	0,1145	0,0995	0,1915
		PD2	0,0827	0,1152	0,1072	0,1189	0,0825	0,1162
	AE2	PD1	0,1263	0,1635	0,1482	0,2669	0,1616	0,2992
		PD2	0,1316	0,0962	0,1643	0,2239	0,1765	0,2652

TABLE C.9

Nonsana Coal Sample

Equilibrium Recovery of Coal (ϕ_{Coal} : %)

			M1		M2		M3	
			PA1	PA2	PA1	PA2	PA1	PA2
PS1	AE1	PD1	29,9	42,3	28,7	51,5	38,0	53,3
		PD2	26,2	52,5	36,1	53,0	46,2	58,8
	AE2	PD1	21,4	44,7	27,2	47,3	44,7	50,8
		PD2	36,4	46,0	38,2	49,1	45,8	53,0
PS2	AE1	PD1	33,1	47,9	40,7	50,0	44,8	51,3
		PD2	46,3	69,8	50,3	69,1	72,3	75,5
	AE2	PD1	43,9	55,6	59,5	65,6	59,6	87,1
		PD2	35,3	67,7	46,9	77,8	41,9	80,1

TABLE C.10

Nonsana Coal Sample

Flotation Rate of Coal (K_{Coal} : s^{-1})

			M1		M2		M3	
			PA1	PA2	PA1	PA2	PA1	PA2
PS1	AE1	PD1	0,1762	0,1157	0,2258	0,1147	0,2001	0,1308
		PD2	0,1378	0,0763	0,1227	0,0792	0,1139	0,0747
	AE2	PD1	0,0499	0,2287	0,0873	0,2158	0,2236	0,2438
		PD2	0,0908	0,1134	0,1022	0,1189	0,1477	0,1096
PS2	AE1	PD1	0,0447	0,0485	0,0475	0,0746	0,0969	0,0872
		PD2	0,0476	0,6439	0,0568	0,0426	0,0345	0,0565
	AE2	PD1	0,0499	0,1439	0,1182	0,1307	0,1081	0,1583
		PD2	0,0635	0,0758	0,0713	0,0715	0,0572	0,0766

TABLE C.11

Nonsana Coal Sample

Equilibrium Recovery of Ash (ϕ_{Ash} : %)

			M1		M2		M3	
			PA1	PA2	PA1	PA2	PA1	PA2
PS1	AE1	PD1	20,1	30,8	19,6	40,8	27,3	42,7
		PD2	15,4	44,3	29,3	38,8	32,6	49,2
	AE2	PD1	10,6	32,6	17,2	44,4	31,0	39,8
		PD2	27,7	44,0	30,7	42,4	39,3	50,0
PS2	AE1	PD1	17,9	28,7	23,9	38,1	31,9	41,6
		PD2	29,4	48,6	31,7	49,6	48,5	58,8
	AE2	PD1	25,2	39,0	40,7	47,2	57,0	63,7
		PD2	22,5	45,7	26,0	53,9	22,6	59,7

TABLE C.12

Nonsana Coal Sample

Flotation Rate of Ash (K_{Ash} : s^{-1})

			M1		M2		M3	
			PA1	PA2	PA1	PA2	PA1	PA2
PS1	AE1	PD1	0,1965	0,1209	0,2538	0,1205	0,2267	0,1256
		PD2	0,1408	0,0682	0,1364	0,0784	0,1255	0,0705
	AE2	PD1	0,0584	0,2215	0,0877	0,2158	0,1755	0,2049
		PD2	0,0908	0,126	0,1029	0,1103	0,1579	0,1156
PS2	AE1	PD1	0,0506	0,0599	0,0554	0,0745	0,1052	0,0879
		PD2	0,0494	0,0448	0,0622	0,0420	0,0365	0,0569
	AE2	PD1	0,0511	0,1483	0,1340	0,1308	0,1167	0,1426
		PD2	0,0618	0,0744	0,0793	0,0690	0,0676	0,0797

TABLE C.13

Longridge Coal Sample

Equilibrium Recovery of Coal (ϕ_{Coal} : %)

			M1		M2		M3	
			PA1	PA2	PA1	PA2	PA1	PA2
PS1	AE1	PD1	53,2	69,9	46,2	73,7	66,8	74,8
		PD2	51,4	76,5	53,6	98,9	69,5	101,7
	AE2	PD1	37,0	54,4	64,1	71,1	72,5	84,1
		PD2	57,3	80,6	76,0	88,6	55,6	85,1
PS2	AE1	PD1	44,4	56,6	41,6	76,5	59,5	85,0
		PD2	38,6	89,2	65,3	73,2	47,7	95,3
	AE2	PD1	37,7	41,5	39,8	58,2	54,5	78,2
		PD2	54,4	89,8	64,2	72,3	44,4	79,3

TABLE C.14

Longridge Coal Sample

Flotation Rate of Coal (K_{Coal} : s^{-1})

			M1		M2		M3	
			PA1	PA2	PA1	PA2	PA1	PA2
PS1	AE1	PD1	0,1418	0,1026	0,1316	0,1048	0,1254	0,0951
		PD2	0,0707	0,0578	0,0940	0,0670	0,0787	0,0363
	AE2	PD1	0,1315	0,2725	0,1255	0,1154	0,2463	0,1312
		PD2	0,1125	0,0595	0,2216	0,1005	0,1399	0,0727
PS2	AE1	PD1	0,1434	0,0919	0,0365	0,0670	0,1623	0,0816
		PD2	0,660	0,0404	0,0325	0,0292	0,0277	0,0311
	AE2	PD1	0,1616	0,2263	0,1829	0,2041	0,3579	0,2714
		PD2	0,0821	0,1097	0,1306	0,1237	0,2236	0,1250

TABLE C.15

Longridge Coal Sample

Equilibrium Recovery of Ash (ϕ_{Ash} : %)

			M1		M2		M3	
			PA1	PA2	PA1	PA2	PA1	PA2
PS1	AE1	PD1	34,3	52,5	31,5	55,3	53,3	60,5
		PD2	29,9	53,9	32,7	60,3	42,0	64,7
	AE2	PD1	18,1	38,4	37,3	42,4	41,7	54,4
		PD2	34,0	54,0	47,6	68,1	34,2	58,1
PS2	AE1	PD1	29,1	36,3	20,0	46,5	35,5	51,4
		PD2	20,4	53,1	33,4	57,7	25,4	56,8
	AE2	PD1	19,9	24,8	20,4	37,1	33,3	52,4
		PD2	34,7	30,4	24,5	61,2	24,5	56,3

TABLE C.16

Longridge Coal Sample

Flotation Rate of Ash (K_{Ash} : s^{-1})

			M1		M2		M3	
			PA1	PA2	PA1	PA2	PA1	PA2
PS1	AE1	PD1	0,1146	0,0977	0,1361	0,1043	0,1140	0,0769
		PD2	0,0998	0,0536	0,0842	0,0703	0,0864	0,3155
	AE2	PD1	0,1452	0,1888	0,1325	0,1046	0,2640	0,1364
		PD2	0,1242	0,0548	0,2337	0,1145	0,1461	0,0657
PS2	AE1	PD1	0,1189	0,0885	0,0382	0,0749	0,2051	0,0837
		PD2	0,0719	0,0431	0,0357	0,0437	0,0673	0,0427
	AE2	PD1	0,1875	0,2551	0,1085	0,2120	0,3704	0,2671
		PD2	0,0890	0,0989	0,0783	0,1272	0,3134	0,1232

APPENDIX D

Sample Calculation of Variance and a Summary of the ANOVA Tables

If the effects of two factors are studied, with one at three levels, the other at two levels (Figure D.1), it is possible to characterise their effects and interactions by calculating five contrasts or effects. These points can be represented graphically and are shown in Figure D.1. Obviously three and higher factor interactions cannot be represented graphically. Appendix E shows the difference between positive and negative effects and interactions.

$$A = a_2b_2 + a_2b_1 + a_2b_0 - a_1b_2 - a_1b_1 - a_1b_0$$

$$B1 = a_2b_2 - a_2b_0 + a_1b_2 - a_1b_0$$

$$B2 = a_2b_2 - 2a_2b_1 + a_2b_0 + a_1b_2 - 2a_1b_1 + a_1b_0$$

$$AB1 = a_2b_2 - a_2b_0 - a_1b_2 + a_1b_0$$

$$AB2 = a_2b_2 - 2a_2b_1 + a_2b_0 - a_1b_2 + 2a_1b_1 - a_1b_0$$

The contrasts A, B1 and AB1 measure the linear effects of the factor. The other two measure the deviations of linearity, because if the points are colinear these quantities equal zero. The method of subdividing the sum of square with two degrees of freedom into single components is given in Table D.2.

An example is taken from the results of the f.d.e. on the Greenside coal sample with the equilibrium recovery of coal as the measured response.

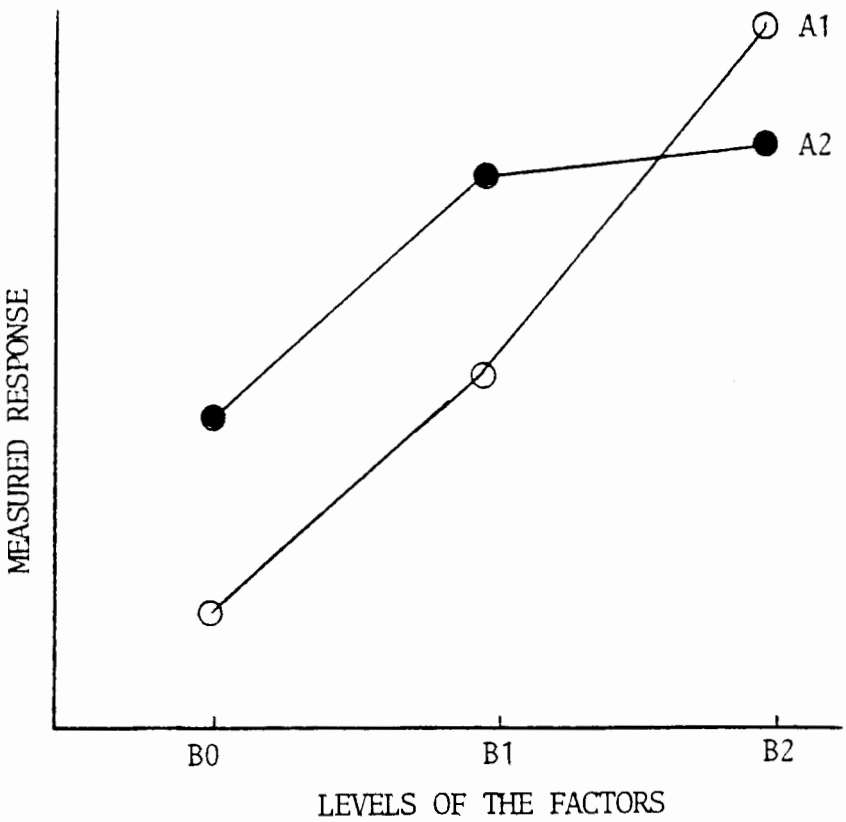

Figure D.1 The effect of two factors, one at two levels the other at three, at equal intervals.

TABLE D.1

Equilibrium Recovery of Coal (Greenside Sample).

Note: The individual cell values represent the total of eight responses obtained from the major Table C.5

	M1	M2	M3	
PA1	342,4	490,5	613,4	1446,3
PA2	528,7	657,4	669,2	1865,3
	881,1	1147,9	1282,6	3311,6

TABLE D.2

A Breakdown of the Treatment Calculations

Contrast	Contrast Value ($\hat{z}$)	Divisor D x r	Sum of Squares
PA	419,0	6 x 8	3657,5
M1	401,5	4 x 8	5037,6
M2	-132,1	12 x 8	181,8
PA x M1	-140,5	4 x 8	616,9
PA x M2	-81,7	12 x 8	69,5
			9563,3

Check this total sum of squares with the one calculated

$$\frac{342,4^2 + 490,5^2 + 613,4^2 + 538,7^2 + 657,4^2 + 669,2^2}{8} - \frac{3311,6^2}{48}$$

$$= 9563,3$$

The contrast values are calculated using the equations described earlier. The divisor is the product of the number of replications (r) and the square of the coefficients of the contrast equations.

The variance calculations for the two way tables is similar to those shown for the initial f.d.e. on the Grootegeluk coal.

The following tables are a summary of the analysis of variance (ANOVA) of the f.d.e. on the four coals.

TABLE D.3

Summarised ANOVA Table of Klimpel Model Constants for Grootegeluk Coal Sample

Note: The 5% significance level is attained for all interactions and effects whose mean square is greater than 4,26 x error mean square.

Contrast	Mean Square			
	Equilibrium Recovery		Flotation Rate x 10^3	
	Coal	Gangue	Coal	Gangue
M1	4940,2	2876,6	332,0	372,4
M2	66,0	103,8	5,4	6,5
PA	1379,2	588,0	332,6	350,2
PS	140,4	30,7	1,5	0,3
AE	80,9	0,0	190,3	164,7
PD	604,2	160,6	0,1	0,1
PA x M1	173,0	110,3	63,0	66,0
PA x M2	5,2	0,1	8,0	10,5
PS x M1	1,1	0,5	0,0	0,3
PS x M2	0,1	0,6	1,6	1,4
AE x M1	12,5	0,7	19,5	22,5
AE x M2	1,4	8,8	2,5	6,2
PD x M1	0,0	0,0	26,2	33,0
PD x M2	11,6	22,6	24,3	27,6
PA x PS	49,4	21,1	9,5	6,7
PA x AE	20,2	34,0	87,8	86,9
PA x PD	133,7	52,7	0,0	0,3
PS x AE	5,7	0,0	8,2	11,1
PS x AD	18,6	1,0	2,1	1,5
AE x PD	0,1	9,2	27,6	34,5
ERROR	33,3	37,2	0,19	0,16

TABLE D.4

Summarised ANOVA Table of Klimpel Model Constants for Greenside Coal Sample

Note: The 5% significance level is attained for all interactions and effects whose mean square is greater than 4,25 x error mean square.

Contrast	Mean Square			
	Equilibrium Recovery		Flotation Rate x 10^3	
	Coal	Gangue	Coal	Gangue
M1	5037,6	5981,4	17,2	12,4
M2	181,8	356,9	4,2	5,2
PA	3657,5	3519,2	24,1	19,6
PS	38,5	2,3	8,5	6,5
AE	118,4	466,3	52,4	55,6
PD	2235,9	3211,1	5,8	5,9
PA x M1	616,9	239,3	4,9	4,6
PA x M2	69,5	118,1	0,9	1,1
PS x M1	2,0	8,5	0,0	1,7
PS x M2	228,5	388,4	0,1	0,5
AE x M1	130,8	358,5	14,9	7,3
AE x M2	13,9	0,0	0.1	0,3
PD x M1	249,2	0,7	0,2	0,3
PD x M2	10,3	97,4	0,7	0,0
PA x PS	478,2	666,0	0,0	0,4
PA x AE	184,1	39,6	7,3	0,1
PA x PD	227,9	366,3	3,6	6,9
PS x AE	15,0	23,8	0,7	0,0
PS x PD	37,1	35,4	1,4	0,1
AE x PD	575,5	290,1	3,7	0,4
ERROR	33,3	37,2	0,19	0,16

TABLE D.5

Summarised ANOVA Table of Klimpel Model Constants for Nonsana Coal Sample

Note: The 5% significance level is attained for all interactions and effects whose mean square is greater than 4,26 x error mean square

Contrast	Mean Square			
	Equilibrium Recovery		Flotation Rate x 10^3	
	Coal	Gangue	Coal	Gangue
M1	1303,1	1433,8	5,4	37,3
M2	4,3	8,5	52,2	0,6
PA	3440,9	3288,5	5,4	0,4
PS	2566,7	470,6	467,1	433,9
AE	70,1	115,0	77,9	37,8
PD	503,1	340,8	266,9	263,8
PA x M1	43,7	29,3	17,2	24,3
PA x M2	0,0	1,3	1,3	2,4
PS x M1	14,9	53,6	2,8	0,5
PS x M2	13,1	0,5	2,0	0,3
AE x M1	12,3	9,5	12,6	5,9
AE x M2	15,5	3,3	2,5	0,8
PD x M1	9,5	66,1	43,6	13,8
PD x M2	0,3	13,8	0,7	0,1
PA x PS	32,3	0,0	14,8	21,7
PA x AE	36,1	14,6	161,7	182,8
PA x PD	62,1	83,5	28,1	21,9
PS x AE	139,4	28,1	15,9	39,2
PS x PD	21,9	43,5	20,1	13,9
AE x PD	373,0	190,8	6,7	0,0
ERROR	33,3	37,2	0,19	0,16

TABLE D.5

Summarised ANOVA Table of Klimpel Model Constants for Nonsana Coal Sample

Note: The 5% significance level is attained for all interactions and effects whose mean square is greater than 4,26 x error mean square

Contrast	Mean Square			
	Equilibrium Recovery		Flotation Rate x 10^3	
	Coal	Gangue	Coal	Gangue
M1	1303,1	1433,8	5,4	37,3
M2	4,3	8,5	52,2	0,6
PA	3440,9	3288,5	5,4	0,4
PS	2566,7	470,6	467,1	433,9
AE	70,1	115,0	77,9	37,8
PD	503,1	340,8	266,9	263,8
PA x M1	43,7	29,3	17,2	24,3
PA x M2	0,0	1,3	1,3	2,4
PS x M1	14,9	53,6	2,8	0,5
PS x M2	13,1	0,5	2,0	0,3
AE x M1	12,3	9,5	12,6	5,9
AE x M2	15,5	3,3	2,5	0,8
PD x M1	9,5	66,1	43,6	13,8
PD x M2	0,3	13,8	0,7	0,1
PA x PS	32,3	0,0	14,8	21,7
PA x AE	36,1	14,6	161,7	182,8
PA x PD	62,1	83,5	28,1	21,9
PS x AE	139,4	28,1	15,9	39,2
PS x PD	21,9	43,5	20,1	13,9
AE x PD	373,0	190,8	6,7	0,0
ERROR	33,3	37,2	0,19	0,16

TABLE D.6

Summarised ANOVA Table of Klimpel Model Constants for Longridge Coal Sample.

Note: The 5% significance level is attained for all interactions and effects whose mean square is greater than 4,26 x error mean square.

Contrast	Mean Square			
	Equilibrium Recovery		Flotation Rate x 10^3	
	Coal	Gangue	Coal	Gangue
M1	1519,4	1020,4	45,6	235,5
M2	15,9	19,9	25,2	133,1
PA	6491,4	4580,6	104,8	61.6
PS	648,3	955,0	6,3	2,0
AE	96,9	163,9	843,4	571,2
PD	1476,3	360,3	518,3	218,3
PA x M1	27,2	54,3	1,8	50,8
PA x M2	-55,7	58,1	297,8	32,3
PS x M1	43,0	1,5	58,5	14,5
PS x M2	46,6	0,9	64,1	81,3
AE x M1	12,1	13,1	60,1	14,8
AE x M2	73,7	88,7	0,4	0,7
PD x M1	602,9	187,7	0,1	15,9
PD x M2	97,0	245,4	88,7	44,4
PA x PS	47,6	6,0	8,2	0,6
PA x AE	243,0	61,0	0,0	24,9
PA x PD	449,0	269,3	12,4	0,6
PS x AE	50,0	0,3	92,3	184,7
PS x PD	4,2	3,0	13,3	99,5
AE x PD	36,1	144,6	29,0	70,7
ERROR	33,3	37,2	0,19	0,16

APPENDIX E

Statistical Notation

During factorial analysis (Appendix D) the factors and their interactions were tested for significant effects. It was also possible to test three level factors for linearity and any deviations. These results are shown graphically in Figure E.1 where M12 and M13 represent the effect of a factor M at the lower levels. The overall effect of M can be obtained by the average of M over all levels of the individual factors investigated. In this case A.

Figures E.2 to E.5 show some selected cases where the interactions and effects are not significant, or positive, or negative.

Figure E.2 shows that both the effects and interactions are not significant.

Figure E.3 shows that M12 and M23 are positive significant effects as are M12A and M23A.

Figure E.4 shows that the effect M12 and interaction M12A are positive and significant while the rest are not significant.

Figure E.5 shows that the effect and interaction M12 is zero and the effect and interaction M23 is positive.

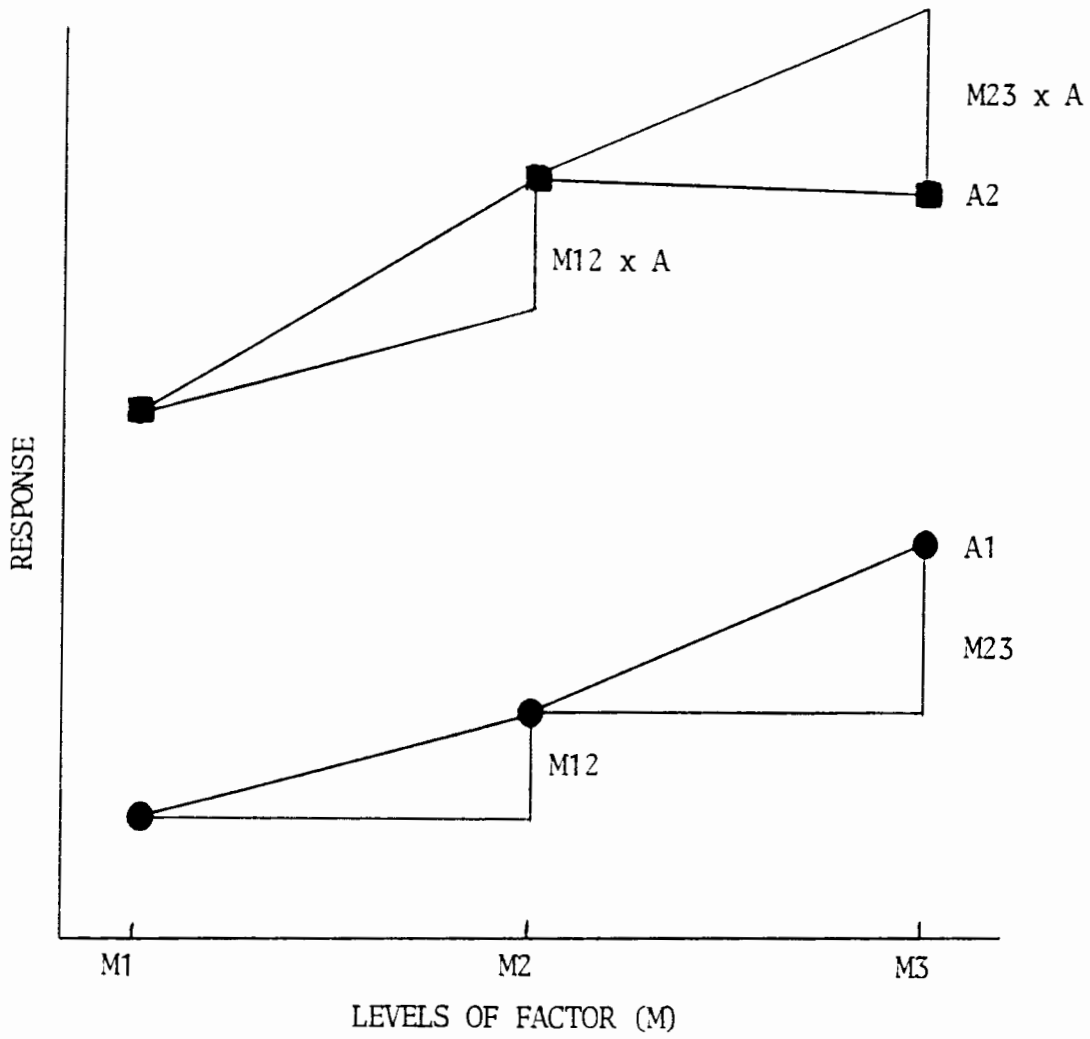

Figure E.1 A graphical representation of the notation.

Note: (i) The main effects M12, M23 and the interaction M12 x A are positive

(ii) The interaction M23 is negative.

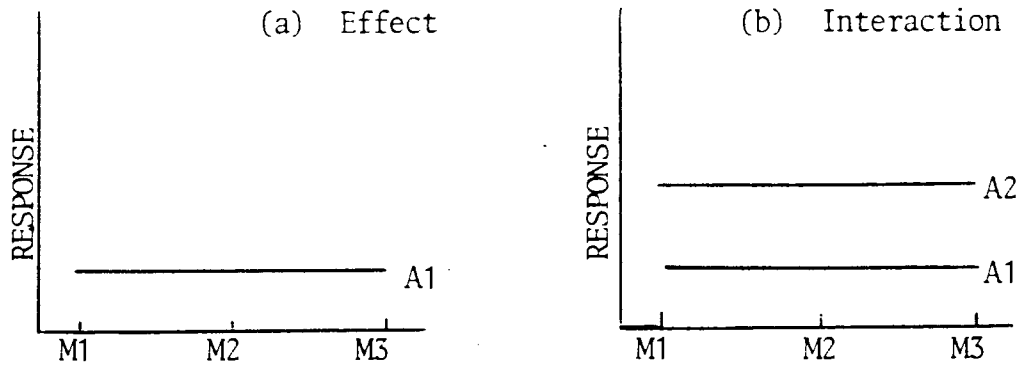

Figure E.2

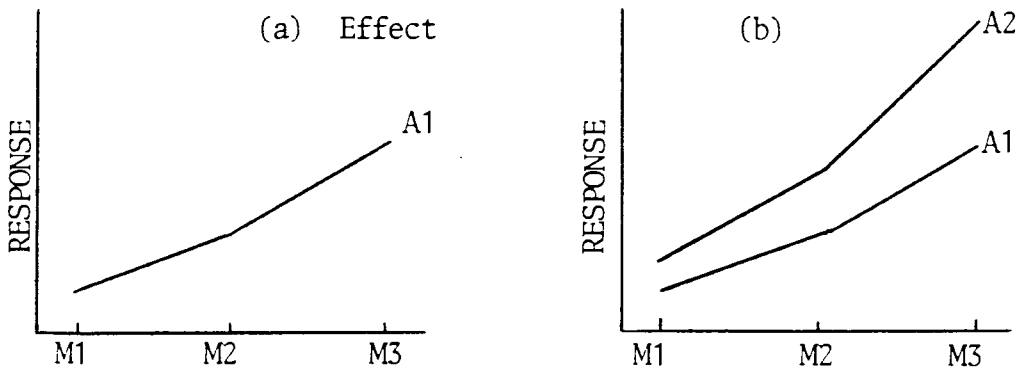

Figure E.3

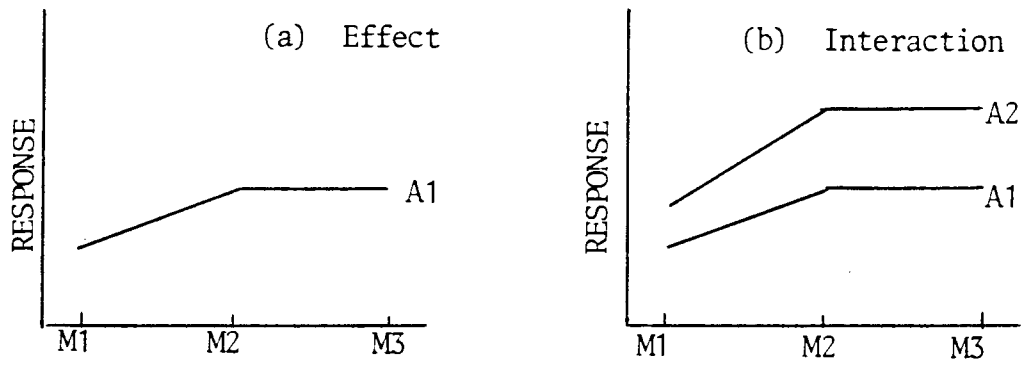

Figure E.4

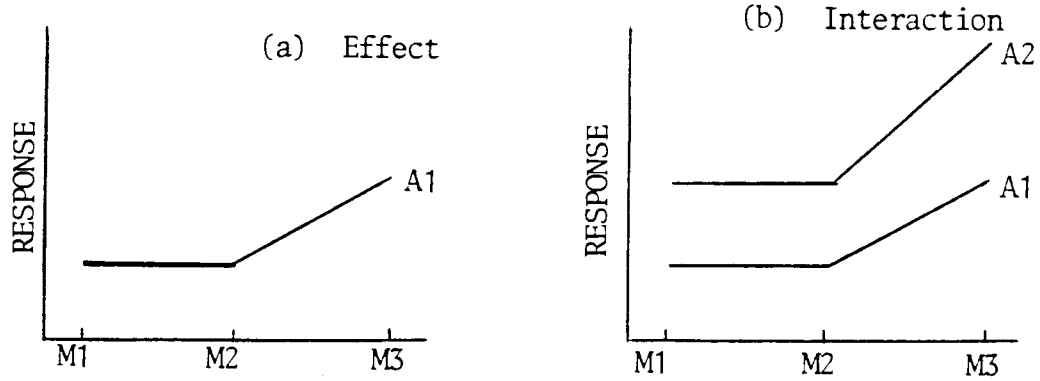

Figure E.5

APPENDIX F

Results of the Batch Flotation Tests Carried Out on the Raw Coals.

TABLE F.1

Flotation conditions varied during batch floats on the raw coals.

Batch Test Number	MIBC Addition (mg/ℓ)	Per cent Solids
1	10	10
2	15	10
3	10	5
4	15	5

TABLE F.2

Results of Batch Flotation Tests carried out on the Grootegeluk Raw Coal Sample

	Batch Test 1		Batch Test 2		Batch Test 3		Batch Test 4	
Time (s)	Mass (g)	Ash (%)	Mass (g)	Ash (%)	Mass (g)	Ash (%)	Mass (g)	Ash (%)
15	27,1	28,7	49,5	36,0	7,2	18,7	15,3	23,4
30	18,1	26,4	31,9	38,2	3,9	18,7	5,8	22,5
60	10,6	26,5	11,3	31,4	2,5	19,1	3,1	22,1
90	4,5	26,4	3,5	30,8	2,5	19,9	2,6	22,3
150	3,5	28,0	3,7	32,2	2,5	20,2	2,6	24,4
Tailings	236,2	70,8	200,1	74,4	131,4	67,6	120,6	70,9
Feed	300	61,6	300	61,6	150	61,6	150	61,6

TABLE F.3

Results of Batch Flotation Tests carried out on the Greenside Raw Coal Sample

	Batch Test 1		Batch Test 2		Batch Test 3		Batch Test 4	
Time (s)	Mass (g)	Ash (%)	Mass (g)	Ash (%)	Mass (g)	Ash (%)	Mass (g)	Ash (%)
15	55,8	15,0	80,6	15,5	21,3	11,9	57,2	14,3
30	58,5	15,9	62,7	15,8	11,9	12,4	14,6	13,9
60	41,2	14,9	46,2	14,2	9,0	11,6	8,9	12,7
90	13,3	12,8	12,7	13,5	4,3	11,2	3,4	12,5
150	11,9	13,6	12,0	14,0	5,2	11,6	3,6	13,4
Tailings	119,3	36,3	85,7	44,4	98,3	29,6	62,3	36,8
Feed	300	23,5	300	23,5	150	23,5	150	23,5

TABLE F.4

Results of Batch Flotation Tests carried out on the Nonsana Raw Coal Sample

	Batch Test 1		Batch Test 2		Batch Test 3		Batch Test 4	
Time (s)	Mass (g)	Ash (%)	Mass (g)	Ash (%)	Mass (g)	Ash (%)	Mass (g)	Ash (%)
15	47,1	12,6	59,3	12,7	39,1	12,9	40,4	12,5
30	38,0	14,1	43,5	14,9	3,5	11,6	3,4	11,3
60	9,5	13,2	9,6	14,1	2,3	10,6	2,2	10,2
90	2,8	12,4	7,7	12,7	2,1	10,1	2,0	10,1
150	6,5	12,2	9,4	12,3	3,5	9,7	3,4	10,0
Tailings	196,1	40,2	170,5	44,0	99,5	40,2	98,6	40,6
Feed	300	30,8	300	30,8	150	30,8	150	30,8

TABLE F.5

Results of Batch Flotation Tests carried out on the Longridge Raw Coal Sample

	Batch Test 1		Batch Test 2		Batch Test 3		Batch Test 4	
Time (s)	Mass (g)	Ash (%)	Mass (g)	Ash (%)	Mass (g)	Ash (%)	Mass (g)	Ash (%)
15	51,3	11,2	64,1	10,7	49,1	11,4	53,9	11,2
30	68,8	11,8	74,8	12,7	18,7	11,1	23,5	12,5
60	28,1	11,6	27,8	12,9	4,4	9,2	6,4	11,2
90	11,9	11,2	15,3	12,4	3,3	8,7	2,7	10,3
150	15,1	10,6	17,7	11,3	5,3	8,7	5,3	10,3
Tailings	124,8	33,9	100,3	38,4	69,2	32,3	58,2	35,5
Feed	300	20,7	300	20,7	150	20,7	150	20,7

APPENDIX G

MACERAL AND MICROLITHOTYPE COMPOSITIONS OF FEED, TAILINGS AND CONCENTRATES FOR THE RAW AND WASHED COALS

TABLE G.1

Notation used in Tables G.2 to G.9

VIT	Vitrinite
S/R/C	Sporinite/Resinite/Cutinite
ALG	Alginite
RSF	Reactive Semifusinite
ISF	Inert Semifusinite
F/SCL	Fusinite/Sclerotinite
MIC	Micrinite
MAC	Reactive Macrinite
INRT	Inertodetrinite
REACTS	Reactives
CALC	Calculated
VIS	Visual

TABLE G.2

Maceral Compositions for Petrographic Float
Performed on the Grootegeluk Raw Coal Sample

Sample No.	Vit.	Exinite		Inertinite							Reacts	
		S/R/C	ALG.	RSF.	ISF.	F/SCL	MIC.	MAC.	INRT	TOTAL	CALC	VIS
Feed	47,5	7,0	,0	1,5	19,0	4,5	,0	,5	20,0	45,5	N/A	56.5
Tailings	47,0	8,0	,0	,0	21,0	1,0	,0	1,0	22,0	45,0	N/A	56.0
Conc. 1	66,0	4,0	,0	2,0	16,7	7,0	,0	,0	4,3	30,0	N/A	71,0
Conc. 2	67,0	6,7	,0	2,0	13,0	9,0	,3	,0	2,0	26,3	N/A	75,7
Conc. 3	70,4	2,3	,0	2,0	16,7	4,3	,3	,0	4,0	27,3	N/A	74,7
Conc. 4	74,0	3,0	,0	1,0	13,0	6,0	,0	,0	3,0	23,0	N/A	78,0

TABLE G.3

Maceral Compositions for the Petrographic Float Performed on the Greenside Raw Coal Sample.

Sample No.	Vit.	Exinite		Inertinite							Reacts	
		S/R/C	ALG.	RSF.	ISF.	F/SCL	MIC.	MAC.	INRT	TOTAL	CALC	VIS
Feed	26,4	3,0	,0	12,0	27,3	5,0	,3	10,0	16,0	70,6	N/A	51,4
Tailings	22,7	2,7	,0	11,0	37,0	4,3	,3	9,7	12,3	74,6	N/A	46,1
Conc. 1	36,4	2,7	,0	13,0	22,0	5,0	,3	9,3	11,3	60,9	N/A	61,4
Conc. 2	32,0	1,7	,0	10,0	25,0	3,0	,3	6,0	22,0	66,3	N/A	49,7
Conc. 3	32,7	3,0	,0	6,0	31,7	7,7	,6	2,7	15,6	64,3	N/A	44,4
Conc. 4	33,5	1,5	,0	12,0	22,0	3,0	,0	10,0	18,0	65,0	N/A	57,0

TABLE G.4

Maceral Composition for the Petrographic Float
Performed on the Nonsana Raw Coal Sample

Sample No.	Vit.	Exinite		Inertinite							Reacts	
		S/R/C	ALG.	RSF.	ISF.	F/SCL	MIC.	MAC.	INRT	TOTAL	CALC	VIS
Feed	50,8	,0	,0	,0	46,7	,3	,0	,0	2,2	49,2	N/A	50,8
Tailings	55,0	,0	,0	,3	40,7	1,7	,0	,0	2,3	45,0	N/A	55,3
Conc. 1	47,7	,0	,0	,0	43,3	1,3	,0	1,0	6,7	52,3	N/A	48,7
Conc. 2	46,8	,0	,0	,0	46,3	1,3	,0	,3	5,3	53,2	N/A	47,1
Conc. 3	47,7	,0	,0	,3	45,3	3,7	,0	,3	2,7	52,3	N/A	48,3
Conc. 4	53,6	,0	,0	,0	35,7	2,0	,0	,0	8,7	46,4	N/A	53,6

TABLE G.5

Maceral Composition for the Petrographic Float
Performed on the Longridge Raw Coal Sample

Sample No.	Vit.	Exinite		Inertinite							Reacts	
		S/R/C	ALG.	RSF.	ISF.	F/SCL	MIC.	MAC.	INRT	TOTAL	CALC	VIS
Feed	48,3	,0	,0	,0	42,0	3,7	,3	,7	5,0	51,7	N/A	49,0
Tailings	37,6	,0	,0	,0	42,0	3,7	,7	,0	16,0	62.4	N/A	37,6
Conc. 1	51,3	,0	,0	,7	35,3	4,0	,7	,3	7,7	48,7	N/A	52,3
Conc. 2	45,4	,0	,0	,7	38,0	3,3	,3	,0	12,3	54,6	N/A	46,1
Conc. 3	52,7	,0	,0	,7	40,3	3,0	,0	,0	3,3	47,3	N/A	53,4
Conc. 4	47,0	,0	,0	,0	45,7	4,3	,0	,0	3,0	53,0	N/A	47,0

TABLE G.6

Maceral Composition for the Petrographic Float Performed on the Grootegeluk Washed Coal Sample

Sample No.	Vit.	Exinite		Inertinite							Reacts	
		S/R/C	ALG.	RSF.	ISF.	F/SCL	MIC.	MAC.	INRT	TOTAL	CALC	VIS
Feed	91,0	3,7	,0	,7	,7	3,6	,3	,0	,0	5,3	N/A	95,4
Tailings	90,7	4,0	,0	,0	,3	4,7	,0	,0	,3	5,3	N/A	94,7
Conc. 1	93,5	1,0	,0	1,0	2,0	2,5	,0	,0	,0	5,5	N/A	95,5
Conc. 2	93,0	3,5	,0	,0	1,5	2,0	,0	,0	,0	3,5	N/A	96,5
Conc. 3	96,0	1,0	,0	,5	1,5	1,0	,0	,0	,0	3,0	N/A	97,5
Conc. 4	98,5	1,0	,0	,0	,0	,5	,0	,0	,0	,5	N/A	99,5

TABLE G.7

Maceral Composition for the Petrographic Float
Performed on the Greenside Washed Coal Sample

Sample No.	Vit.	Exinite		Inertinite							Reacts	
		S/R/C	ALG.	RSF.	ISF.	F/SCL	MIC.	MAC.	INRT	TOTAL	CALC	VIS
Feed	57,6	4,7	,0	7,7	7,0	9,3	2,3	3,7	7,7	37,7	N/A	73,7
Tailings	46,7	7,3	,0	8,7	12,0	11,2	1,0	6,3	6,7	46,0	N/A	69,0
Conc. 1	49,3	4,3	,0	7,7	17,0	8,0	1,7	4,0	8,0	46,4	N/A	65,3
Conc. 2	46,1	6,3	,0	8,7	9,3	11,3	4,3	3,7	10,3	47,6	N/A	64,8
Conc. 3	44,0	5,0	,0	9,5	14,0	12,5	3,5	3,5	8,0	51,0	N/A	62,0
Conc. 4	52,3	3,0	,0	5,7	13,7	12,3	2,0	4,7	6,3	44,7	N/A	65,7

TABLE G.8

Maceral Composition for the Petrographic Float
Performed on the Nonsana Washed Coal Sample

Sample No.	Vit.	Exinite		Inertinite							Reacts	
		S/R/C	ALG.	RSF.	ISF.	F/SCL	MIC.	MAC.	INRT	TOTAL	CALC	VIS
Feed	42,1	,0	,0	2,3	48,0	1,3	,0	3,0	3,3	57,9	N/A	47,4
Tailings	39,7	,0	,0	1,7	46,2	3,0	,0	3,7	5,7	60,3	N/A	45,1
Conc. 1	50,5	,0	,0	1,0	34,5	5,5	,5	2,0	6,0	49,5	N/A	53,5
Conc. 2	45,3	,0	,0	3,3	41,7	1,7	,0	1,0	7,0	54,7	N/A	49,6
Conc. 3	35,3	,0	,0	1,5	46,7	4,7	,0	5,5	6,3	64,7	N/A	42,3
Conc. 4	41,3	,0	,0	1,7	45,3	4,0	,0	,7	7,0	58,7	N/A	43,7

TABLE G.9

Maceral Composition for the Petrographic Floats
Performed on the Longridge Washed Coal Sample

Sample No.	Vit.	Exinite		Inertinite							Reacts	
		S/R/C	ALG.	RSF.	ISF.	F/SCL	MIC.	MAC.	INRT	TOTAL	CALC	VIS
Feed	62,0	,0	,0	1,7	35,0	1,0	,0	,3	,0	38,0	N/A	64,0
Tailings	59,8	,0	,0	1,7	25,5	,8	,0	,0	2,2	40,2	N/A	61,5
Conc. 1	54,2	,0	,0	1,7	36,7	1,7	,0	,0	5,7	45,8	N/A	55,9
Conc. 2	61,0	,0	,0	1,0	35,0	2,0	,0	,0	1,0	39,0	N/A	62,0
Conc. 3	58,4	,0	,0	,0	33,3	1,3	,0	,0	7,0	41,6	N/A	58.4
Conc. 4	59,5	,0	,0	3,0	34,0	2,5	,0	,0	1,0	40,5	N/A	62,5

TABLE G.10

Notation used in Tables G.11 to G.18

VITS	Vitrites
INTM	Intermediates
FSSF	Fusite/Semi-fusite/Sclerotite
MINM	Macroite/Inertodetrite/Micrite
CBMN	Carbominerites
CCBC	Cannel/Boghead Coal
O/SH	Oil Shale
CL/E	Clarite E
TR/E	Trimacerite E
DU/E	Durite E

TABLE G.11

Microlithotype Compositions for the Petrographic Floats Performed on the Grootegeluk Raw Coal Sample

SAMPLE NO	VITS	INTM	INERTITES			CBMN	LIPTITES					
			FSSF	MINM	TOTAL		CCBC	O/SH	CL/E	TR/E	DU/E	TOTAL
Feed	42,0	4,0	11,0	4,5	15,5	38,5	,0	,0	,0	,0	,0	,0
Tailings	13,3	1,7	1,7	,0	1,7	83,3	,0	,0	,0	,0	,0	,0
Conc. 1	52,0	17,7	14,3	1,7	16,0	14,3	,0	,0	,0	,0	,0	,0
Conc. 2	57,5	13,0	12,0	2,0	14,0	13,5	,0	,0	2,0	,0	,0	2,0
Conc. 3		,0	,0	,0	,0	,0	,0	,0	,0	,0	,0	,0
Conc. 4		,0	,0	,0	,0	,0	,0	,0	,0	,0	,0	,0

TABLE G.12

Microlithotype Composition for the Floats Performed on the Greenside Raw Coal Sample

SAMPLE NO	VITS	INTM	INERTITES			CBMN	LIPTITES					
			FSSF	MINM	TOTAL		CCBC	O/SH	CL/E	TR/E	DU/E	TOTAL
Feed	21,0	23,0	19,0	32,7	51,7	4,0	,0	,0	,3	,0	,0	,3
Tailings	16,0	20,5	13,0	38,0	51,0	12,5	,0	,0	,0	,0	,0	,0
Conc. 1	27,0	19,0	15,0	32,0	47,5	6,5	,0	,0	,0	,0	,0	,0
Conc. 2	25,5	26,5	15,0	31,0	46,0	2,0	,0	,0	,0	,0	,0	,0
Conc. 3	20,3	27,0	12,7	38,3	51,0	1,7	,0	,0	,0	,0	,0	,0
Conc. 4		,0	,0	,0	,0	,0	,0	,0	,0	,0	,0	,0

TABLE G.13

Microlithotype Composition for the Floats
Performed on the Nonsana Raw Coal Sample

SAMPLE NO	VITS	INTM	INERTITES			CBMN	LIPTITES					
			FSSF	MINM	TOTAL		CCBC	O/SH	CL/E	TR/E	DU/E	TOTAL
Feed	39,8	19,0	34,0	3,2	37,2	4,0	,0	,0	,0	,0	,0	,0
Tailings	45,0	20,5	32,0	,0	32,0	2,5	,0	,0	,0	,0	,0	,0
Conc. 1	39,5	18,0	29,0	6,0	35,0	7,5	,0	,0	,0	,0	,0	,0
Conc. 2	27,5	30,5	37,5	2,0	39,5	2,5	,0	,0	,0	,0	,0	,0
Conc. 3	29,0	16,0	43,5	6,0	49,5	5,5	,0	,0	,0	,0	,0	,0
Conc. 4	29,0	39,0	30,0	,5	30,5	1,5	,0	,0	,0	,0	,0	,0

TABLE G.14

Microlithotype Composition for the Floats Performed on the Longridge Raw Coal Sample

SAMPLE NO	VITS	INTM	INERTITES			CBMN	LIPTITES					
			FSSF	MINM	TOTAL		CCBC	O/SH	CL/E	TR/E	DU/E	TOTAL
Feed	28,7	27,0	25,3	5,7	31,0	13,3	,0	,0	,0	,0	,0	,0
Tailings	20,0	6,0	48,0	2,5	50,5	23,5	,0	,0	,0	,0	,0	,0
Conc. 1	32,0	22,0	37,0	2,0	39,0	7,0	,0	,0	,0	,0	,0	,0
Conc. 2	35,0	17,5	39,5	1,0	40,5	7,0	,0	,0	,0	,0	,0	,0
Conc. 3	33,5	28,5	29,0	3,0	32,0	6,0	,0	,0	,0	,0	,0	,0
Conc. 4	32,5	22,5	33,0	5,0	38,0	7,0	,0	,0	,0	,0	,0	,0

TABLE G.15

Microlithotype Composition for the Floats
Performed on the Grootegeluk Washed Coal Sample

SAMPLE NO	VITS	INTM	INERTITES			CBMN	LIPTITES					
			FSSF	MINM	TOTAL		CCBC	O/SH	CL/E	TR/E	DU/E	TOTAL
Feed	89,5	1,0	,5	,0	,5	9,0	,0	,0	,0	,0	,0	,0
Tailings	75,5	,5	,5	,0	,5	23,5	,0	,0	,0	,0	,0	,0
Conc. 1	90,5	5,0	1,0	,0	1,0	3,5	,0	,0	,0	,0	,0	,0
Conc. 2	91,5	5,5	,5	,0	,5	2,5	,0	,0	,0	,0	,0	,0
Conc. 3	94,5	1,5	1,0	,0	1,0	3,0	,0	,0	,0	,0	,0	,0
Conc. 4		,0	,0	,0	,0	,0	,0	,0	,0	,0	,0	,0

TABLE G.16

Microlithotype Compositions for the Petrographic Floats Performed on the Greenside Washed Coal Sample

SAMPLE NO	VITS	INTM	INERTITES			CBMN	LIPTITES					
			FSSF	MINM	TOTAL		CCBC	O/SH	CL/E	TR/E	DU/E	TOTAL
Feed	37,7	45,6	7,7	6,0	13,7	,0	,0	,0	2,0	1,0	,0	3,0
Tailings	34,7	43,7	7,7	10,0	17,7	1,0	,0	,0	1,3	1,3	,3	2,9
Conc. 1	36,0	41,4	6,0	9,3	15,3	,0	,0	,0	5,0	2,0	,3	7,3
Conc. 2	44,5	35,5	8,5	2,5	11,0	1,5	,0	,0	5,5	2,0	,0	7,5
Conc. 3	37,0	38,0	9,0	7,0	16,0	,0	,0	,0	7,5	1,5	,0	9,0
Conc. 4	36,5	42,0	9,0	9,5	18,5	,0	,0	,0	2,5	,5	,0	3,0

TABLE 9.17

Microlithotype Composition for the Petrographic Floats
Performed on the Nonsana Washed Coal Sample

SAMPLE NO	VITS	INTM	INERTITES			CBMN	LIPTITES					
			FSSF	MINM	TOTAL		CCBC	O/SH	CL/E	TR/E	DU/E	TOTAL
Feed	20,0	23,5	27,0	26,0	53,0	3,5	,0	,0	,0	,0	,0	,0
Tailings	18,0	19,0	32,0	20,0	52,0	11,0	,0	,0	,0	,0	,0	,0
Conc. 1	31,0	25,0	38,5	4,0	42,0	1,5	,0	,0	,0	,0	,0	,0
Conc. 2	26,5	25,0	30,5	15,0	45,5	3,0	,0	,0	,0	,0	,0	,0
Conc. 3	21,5	21,0	45,0	9,0	54,0	3,5	,0	,0	,0	,0	,0	,0
Conc. 4	20,3	30,2	32,5	15,3	47,8	1,7	,0	,0	,0	,0	,0	,0

TABLE G 18

Microlithotype Compositions for the Petrographic Floats
Performed on the Longridge Washed Coal Sample

SAMPLE NO	VITS	INTM	INERTITES			CBMN	LIPTITES					
			FSSF	MINM	TOTAL		CCBC	O/SH	CL/E	TR/E	DU/E	TOTAL
Feed	31,3	39,0	15,3	,7	26,0	3,7	,0	,0	,0	,0	,0	,0
Tailings	35,0	27,5	29,0	,5	29,5	8,0	,0	,0	,0	,0	,0	,0
Conc. 1	45,0	28,5	23,5	1,0	24,5	2,0	,0	,0	,0	,0	,0	,0
Conc. 2	50,0	28,0	19,0	,5	19,5	2,5	,0	,0	,0	,0	,0	,0
Conc. 3	43,0	33,0	19,0	1,0	20,0	4,0	,0	,0	,0	,0	,0	,0
Conc. 4	53,0	31,5	13,0	1,0	14,0	1,5	,0	,0	,0	,0	,0	,0