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GRADUATE SCHOOL OF NATURAL AND APPLIED SCIENCES

THE ENRICHMENT OF ZONGULDAK FINE COAL BY FLOTATION

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by

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İZMİR

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SUMMARY

In the present work, the enrichment of Zonguldak fine coal by flotation is investigated.

This study covers the following headlines:

- Coal-A fossil fuel
- Formation of coal
- Properties of coal and its impurities
- Coal preparation
- General information about coal flotation
- Flotation machines of coal
- Flotation circuits of Coal
- Description of Zonguldak hard coal region

Cleaning coal fines, which in the past at best marginally profitable, is now an essential part of coal preparation strategy for a number of reasons:

- 1- Percentage of sizes finer than 28 mesh in the plant feed has increased dramatically as result of intensive mechanization and by blending and storage.
- 2- The quality of raw coal particles smaller than 28 mesh has decreased significantly through wide spread application of full seam, non-selective mining systems.
- 3- Black water problem which has to be controlled from an environmental point of view.
- 4- With substantial increase in the value of coal and its cost of production, the fines previously wasted become a very valuable product.

Froth flotation is the only established technique for treating coals below 0.5 mm in size.

In this thesis, Zonguldak fine coal is treated by flotation. In order to examine the flotation characteristics of fine coals and to find out the effect of various reagents on the coal flotation and to determine the most proper flotation conditions, various experiments have been done. The aim of experiments has been to obtain low-ash concentrates from high ash levels (approximately 50%).

The flotation experiments have been carried out in two steps. In the first step, the parameters determined are collector type, frother type, conditioning time, solid-liquid ratio, depressant type and amount of depressant.

In the second step, an attempt has been made to study the effect of some operating variables in the coal flotation process on the efficiency index. The operating variables studied are, the collector dosage (kerosene), the frother dosage (isocotanol) and the time of flotation. From the results obtained in relation with the efficiency index, the amount of reagent consumption, for predetermined flotation periods has been calculated by utilizing third-degree regression equation. The highest efficiency index values were obtained at flotation time of 180 seconds. At this flotation time, the values obtained were $C=1.82$ kg/t, $F=0.179$ kg/t and $E_{max}=282.45$.

OZET

Bu çalışmada Zonguldak ince kömürünün flotasyon yöntemi ile zenginleştirilmesi araştırılmıştır. Öncelikle kömürün oluşumu, özellikleri, iri ve ince kömür zenginleştirme yöntemleri, kömür flotasyonu hakkında genel bilgiler ve Zonguldak taşkömürü havzası ile ilgili genel bilgileri içeren kaynak araştırması yapılmıştır.

Günümüzde ince kömürün zenginleştirilmesi şu nedenlerden dolayı kömür hazırlama stratejisinin önemli bir parçasını oluşturmaktadır:

- 1- Yıkama ünitelerine beslenen malzemedeki ince kömür miktarı (-0.5 mm), mekanizasyon, eleme, harmanlama ve depolama gibi işlemlerin sonucu olarak belirgin bir şekilde artmıştır.
- 2- Tam damar ve selektif olmayan üretim yöntemlerinin yaygın şekilde uygulanması sonucu, tüvenan kömürdeki 0.5 mm den daha ince malzemenin kalitesi azalmıştır.
- 3- Kirli su probleminin çevresel açıdan kontrol altına alınması gereksinimi artmıştır.
- 4- Kömürün değerli hale gelmesi ve üretim maliyetinin pahalılaşması nedeniyle, ince kömürler değerli bir ürün haline gelmiştir.

Flotasyon 0.5 mm den daha ince kömürlerin zenginleştirilmesinde kullanılan tek yöntemdir.

Bu çalışmada, Zonguldak ince kömürünün flotasyon özelliklerini ortaya koymak, değişik reaktiflerin kömür flotasyonu üzerindeki etkilerini incelemek ve en uygun flotasyon şartlarını saptamak için flotasyon deneyleri yapılmıştır. Böylece yüksek kül içerikli (%50) kömürlerden, düşük küllü konsantreler elde edilmiştir.

Deneyler iki kademede yürütülmüştür. Birinci kademede incelenen parametreler tane iriliği, kollektör cinsi, köpürtücü cinsi, kondisyon süresi, katı-sıvı oranı, bastırıcı cinsi ve miktarı, emülgatör etkisidir.

Deney sonuçlarına göre tane boyutunun küçültülmesi konsantre kül içeriğini arttırmaktadır. Kollektör cinsi olarak sırasıyla gazyağı, mazot, light neutral seçilmiştir. Optimum kollektör miktarı 1.5-2 kg/t dur. Etil alkol ve izo oktanolun köpürtücü olarak kullanımı uygun görülmüştür. Optimum köpürtücü miktarı 200 gr/t dur. Optimum kondisyon süresi 3-5 dakika ve katı-sıvı oranı % 15 olarak tesbit edilmiştir.

Deneylerde kullanılan tri-polyphosphate in kömür flotasyonunu olumlu yönde, sodyum silikatın ise olumsuz yönde etkidiği gözlenmiştir. Non-ionik emülgatörlerin kömür flotasyonu üzerinde herhangi bir etkisi yoktur.

ikinci kademedeki verim indeksine bağı olarak çeşitli parametrelerin kömür flotasyonu üzerindeki etkileri incelenmiştir. Bu parametreler 1) Kollektör (gazyağı) miktarı 2) Köpürtücü (izo oktanol) miktarı 3) Flotasyon zamanı

Bu parametrelerin saptanması için 3. derece regresyon denklemi kullanılmıştır. Bu denklem şu şekildedir:
$$E = a_1 + a_2 F + a_3 C + a_4 F^2 + a_5 C^2 + a_6 CF + a_7 C^3 + a_8 F^3 + a_9 C^2F + a_{10} CF^2$$

Burada $a_1 - a_{10}$ katsayıları flotasyon zamanına bağı katsayılarıdır. Bu denklem LOTUS bilgisayar programı (DATA/REGRESSION) kullanılarak %9 luk bir standart sapma ile çözülmüştür. Bu çözüme göre 150 sn flotasyon zamanında kollektör dozajı $C = 1.82$ kg/t ve köpürtücü dozajı $F = 0.170$ kg/t olarak bulunmuştur. 180 saniye flotasyon zamanında $E_{max} = 292.70$, $C = 1.87$ kg/t, $F = 0.189$ kg/t dur. Deneysel sonuçlar ile elde edilen sonuçlar istatistiksel uyum içerisindedir.

Elde edilen optimal flotasyon koşullarına göre:

- i) Oktanöl kullanımı ile % 22.67 kül içerikli ve % 90.21 verimde temiz konsantre elde edilmiştir.
- ii) Etil alkol kullanımı ile %9.61 kül ve % 48.94 verim değerinde temiz konsantre elde edilmiş, bu kademenin artığına oktanöl ilave edilerek % 38.10 küllü ara ürün % 41.01 verimle elde edilmiştir.

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1. COAL- A FOSSIL FUEL

Next to the mineral oil and kerogen, coal is the most important reservoir of carbon in the world. Carbon is an indispensable necessity of life as well as the main source of energy (12).

As seen in table 1.1, coal is the world's most abundant fossil fuel. World total resources of coal are currently estimated at about 11.75×10^6 tonnes and, in terms of energy content, are over 20 times the total resources of crude oil and over 1.5 times the total resources of crude oil and all other fossil fuels combined (22).

Table 1.1- World total resources in situ of fossil fuels (22)

Fuel	Quantity ($\text{tx } 10^6$, except for natural gas)	Energy Content	
		($\text{Jx}10^6$)	(% Of world t.)
Crude oil	257500	11134	2.8
Natural gas	$281743 \times 10^9 \text{ m}^3$	10509	2.6
Oil shale	2 511600 (as oil)	108602	26.9
Tar sand	364155 (as bitumen)	15746	3.9
Peat	222378	1861	0.5
Coal	11 769779	255246	63.3
Total		403098	100.0

It is now widely agreed that the availability of oil in international trade is likely to diminish over the next two decades. This is to some extent partly reflected by the steady growth in coal exportation which has occurred in the past 20 years. Fig 1.1 shows the gradual expansion which is occurring in coal consumption in terms of oil equivalent.

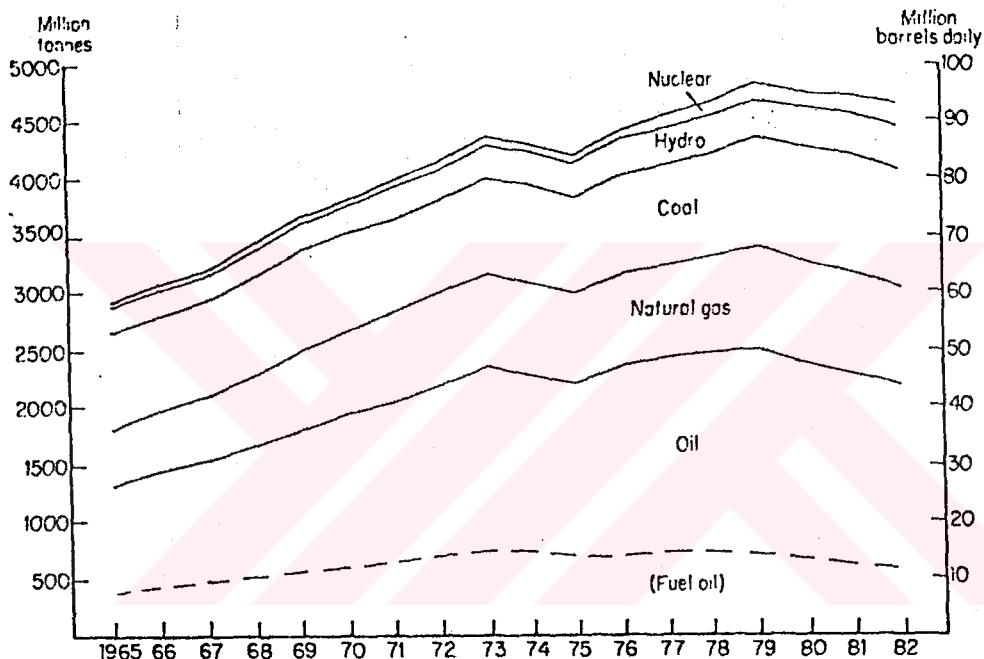


Fig 1.1- Non-communist world primary energy consumption (22)

Vigorous conversation, the development and rapid implementation of programmes for nuclear power, unconventional sources of oil and gas, solar energy and new technologies will not be sufficient to meet the growing energy needs of the world. Coal will, therefore become increasingly utilized to meet this shortfall.

2. FORMATION OF COAL

Coal can be defined as a sedimentary rock composed principally of macerals and subordinantly of minerals, containing water and gases in submicroscopic pores. Coals originated as peat deposits formed in swamps. Wood was the dominant plant material in typical peat swamps. Wood as the term is used here, is highly complex ;that is, the structural elements of the plants are referred to as wood, including the leaf tissues, roots, bark and trunk-stem structures. The chemical constituents comprising wood are predominantly classifiable as celluloses (including hemicelluloses) and lignin, which are variable in their combustion and molecular structures. Lignocellulosic material is the source both of vitrinite and of most inertite (18).

The transformation of plant substances into coal is generally considered to involve two stages :

1. The biochemical (peat) stage, which is instrumental in establishing coal type.
2. The dynamochemical (geochemical, meteamorphic) stage, which establishes coal rank.

For a coal seam to exist, the plant substances from which it formed had to be preserved from oxidative and microbial destruction. The preservation and accumulation took place in swamps where relatively and quiescent parts. In most areas topographic subsidence maintained the water level at the

surface of the peat. Not only did the water level have to balance peat accumulation had to be active growth stage, but peat accumulation had to be terminated by relatively rapid burial under waterborne sediments to seal the peat from oxidative or erosional destruction.

The biochemical stage was essentially terminated when the peat was buried to a depth of 10 meter or more. Both nitrogen and sulphur appear to be fixed during the biochemical stage, the nitrogen originating from plant and microbial proteins, the sulphur from dissolved SO_2 .

Physical conditions, principally water depth and related water movement affected chemical conditions in the accumulating peat. The chemical conditions, especially hydrogen ion concentration (pH) and oxidation/ reduction (REDOX) potential, in turn affected the microbiological activities. These interrelated physical, chemical and microbiological conditions interacted with the indigenous plant materials to form specific types of peat, hence specific coal types.

Following the biochemical stage of active peat accumulation, additional physical and chemical changes took place during the metamorphic stage. Only for agencies have been seriously considered as possible causes of coal metamorphism, namely pressure, heat, tectonism and gamma-ray irradiation. The fundamental cause is the increased temperature associated

with increased depth seems to be well established. Geothermal gradients typical of most areas in the earth ($1^{\circ}\text{C}/25\text{ m}$ of depth) were adequate to create high rank coals. About 50-200°C, chemical changes occurred. Heat and pressure converted the organic material to substances more and more similar to graphite. Coals are ranked according to how severely they were metamorphosed hence where they are located along this continuum from peat to graphite.

Minerals can be classified, on the basis of their origin, as detrital, syngenetic and epigenetic. Detrital minerals are those that were transported into the swamp as particles borne either by water or wind. Syngenetic minerals were crystallized in place from dissolved ions at the same time that the peat was deposited. Pyrite is the most common syngenetic mineral. Epigenetic minerals are those that were formed after the peat accumulated. Calcite, dolomite, kaolinite and some pyrite are the most common epigenetic minerals.

Table 3-2 Classification of Coals by Rank (ASTM D-388) (22)

Class	Group	Fixed carbon limits, % (dry mineral- matter-free basis)		Volatile matter limits, % (dry mineral- matter-free basis)		Calorific value limits, Btu per lb (moist mineral- matter-free basis)		Agglomerating character
		Equal or greater than	Less than	Greater than	Equal or less than	Equal or greater than	Less than	
I. Anthracite	1. Meta-anthracite	98	...	...	2	...	...	...
	2. Anthracite	92	98	2	8	...	...	...
	3. Semianthracite	86	92	8	14	...	...	Nonagglomer- ating†
II. Bituminous	1. Low volatile bituminous coal	78	86	14	22	...	...	Commonly ag- glomerating #
	2. Medium volatile bituminous coal	69	78	22	31	14,000\$	...	
	3. High volatile A bituminous coal	...	69	31	...	13,000\$	14,000	
	4. High volatile B bituminous coal	...	...	...	...	{11,500	13,000	Agglomerating
	5. High volatile C bituminous coal	...	...	...	...	10,500	11,500	
III. Subbituminous	1. Subbituminous A coal	...	...	...	...	10,500	11,500	Nonagglomer- ating
	2. Subbituminous B coal	...	...	...	...	9,500	10,500	
	3. Subbituminous C coal	...	...	...	...	8,300	9,500	
IV. Lignite	1. Lignite A	...	...	...	...	6,300	8,300	Nonagglomer- ating
	2. Lignite B	...	...	...	...	...	6,300	

*This classification does not include a few coals, principally nonbanded varieties, which have unusual physical and chemical properties and which come within the limits of fixed carbon or calorific value of the high volatile bituminous and subbituminous ranks. All these coals either contain less than 48% dry mineral-matter-free fixed carbon or have more than 15,500 moist mineral-matter-free Btu per pound (816 per pound $\times 2.325 =$ kilojoules per kilogram).

†Moist refers to coal containing its natural inherent moisture but not including visible water on the surface of the coal.

‡If agglomerating, classify in low volatile group of the bituminous class.

\$Coals having 69% or more fixed carbon on the dry mineral-matter-free basis shall be classified according to fixed carbon, regardless of calorific value.

#It is recognized that there may be nonagglomerating varieties in these groups of the bituminous class, and there are notable exceptions in the high volatile C bituminous group.

3. PROPERTIES OF COAL AND ITS IMPURITIES

3.1 Rank Classification

Of the several ways to classify coals, rank is generally the most common. Fig 2.1 shows a simplified classification of coals by rank (22).

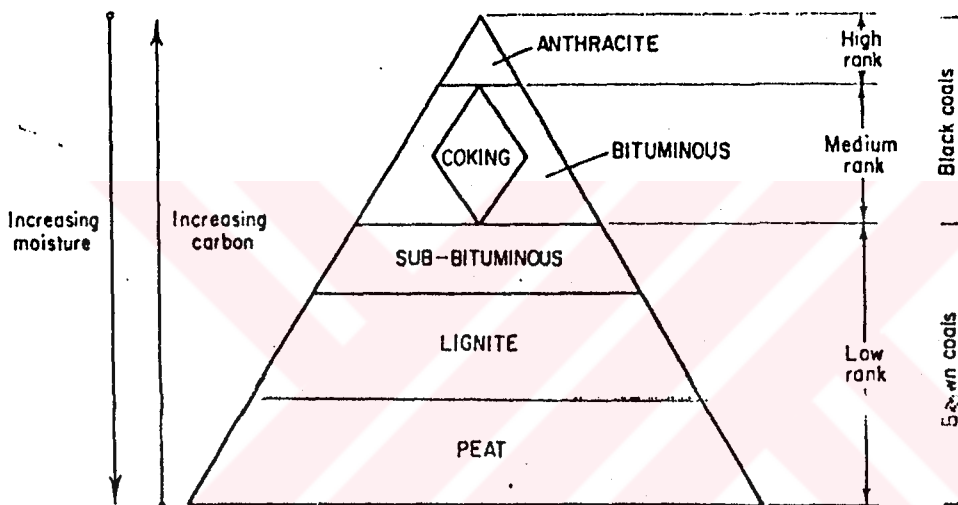


Fig 3.1- Broad classification of coals by rank (22)

Table 3.1 shows the American classification system developed by the American Society for Testing and Materials (ASTM). For the anthracite and low and medium volatile bituminous coals, the most mineral matter-free calorific value changes very little hence the fixed carbon criterion is used. For high volatile bituminous, subbituminous and lignite coals, the moist mineral matter free calorific value is used, since the fixed carbon is almost the same for all classifications (12).

Table 3.1 Some Representative Properties of coal Lithotypes (22)

Lithotypes	General appearance	Specific gravity ¹⁰	Ash	Representative analysis of lithotypes from an Illinois high volatile bituminous coal ¹¹		Relative hardness ¹² (based on power required to crush to -140 mesh). † Eusain (softest lithotype) = 1	
				Volatile matter	Fixed carbon	Btu*	
Vitrain	Uniform shiny black bands	About 1.3	0.5-1% (mainly plant ash)	35.1	64.9	14,438	2
Clarain	Laminated: composed of shiny and dull bands	About 1.3	0.5-2%	40.3	59.7	14,600	3
Durain	Dull, non-reflecting; poorly laminated	1.25-1.45	1-5%, mostly 3-5%, occasionally higher (much extraneous ash)	53.8	46.2	15,371	7.5
Fusain	Charcoal-like fragments	Soft fusain, 1.35-1.45; hard fusain, 1.6 and above	5-10% or more	9.5	90.5	15,632	1

* British thermal unit per pound X 2.326 = kilojoules per kilogram.
† 140 mesh = 0.106 mm.

3.2 Petrographic Composition Of Coal

Coal is an organic rock. Just as an inorganic rock is composed of petrological components known as minerals, coal consists of petrological components called macerals. There is a widespread classification of coals into four types which are vitrain, clarain, durain and fusain (9,10,22,24,26) .

Table 3.2 shows some representative properties of coal lithotypes.

Flotation characteristics of group macerals and lithotypes are different. The bright types of coal (vitrain and durain) show high flotation activity and contain little foreign matter. The dull coals (clarain and fusain) are not so readily floated as the brights.

Some experiments were done by Ozbayoglu to determine the minerologic and petrographic composition of Zonguldak bituminous coal seams namely Cay, Acilik, Piric (23).

Ozbayoglu found that vitrain had the greatest effect on the floatability of the parent coal due to its high percentage and found that Cay coal seam had a lower floatability than the other seams (table 3.3)

Table 3.3 Petrographic analysis of Zonguldak coal (23)

Maceral % volume	SAMPLES					
	PIRIC	ACILIK	CAY	VITRAIN	DURAIN	FUSAIN
Vitrinite	88.8	75.4	71.0	95.4	20.2	7.4
Exinite	3.8	3.0	3.8	1.0	10.0	0.3
Inertite	4.0	17.8	21.2	2.3	66.2	92.0
Shale	2.2	0.8	3.6	1.3	2.6	-
Pyrite	0.2	-	-	-	-	-
Other m.m	1.0	3.0	0.4	-	1.0	0.3
Reflectivity (vitrinite)	1.22	1.11	1.07	-	-	-

3.3 Physical Properties

3.3.1 Moisture

Fresh coal at a dry place in a mine is saturated with moisture although it appears dry and dusty when crushed. The percentage of moisture present, called bed moisture, is more or less constant throughout a given mine and is a characteristics of rank. Such moisture varies from 1,2 or 3 % bituminous coal to 45 % or more in lignite.

Moisture is present in coal on the surface and inside cracks and large capillaries (free or surface water) or in the internal pore structure of the coal (bound or inherent water). Free water has a normally vapor pressure and the same physical properties as ordinary water (12,13).

3.3.2 Volatile Matter

Volatile matter, or that matter which distills as coal is heated to about 950°C, consists mainly of the combustible gases- hydrogen, carbon monoxide, methane, tar vapors, and some incombustible gases such as carbon dioxide and water vapor. The composition of volatile matter varies greatly for different ranks of coal, with the proportion of incombustible gases increasing as the rank increases (12).

3.3.3 Density and Prosimity

The true density of coal increases with increasing rank. The density is an important quantity has to be considered when examining the structure of coal. Exinites have a lower and micrinite a higher density than vitrinite of the same rank.

Coals contain two pore systems, one with a mean pore diameter of 500 Å and second measuring 5-15 Å diameter. The latter pores have a small volume but large internal surface.

3.3.4 Specific Gravity

Specific gravity of coal ranges from 1.23-1.72 depending on rank, moisture and ash content. The specific gravity of coal increases with rank. All coal cleaning methods in general use are methods of gravity concentration with the exception of froth flotation.

3.3.5 Grindability

A general relationship exists between grindability and rank. Coals that are easiest to grind are found in the medium and low volatile groups. the effect of grindability with respect to increased carbon is opposite to that of hardness and reaches a maximum at 90 % carbon (12,15).

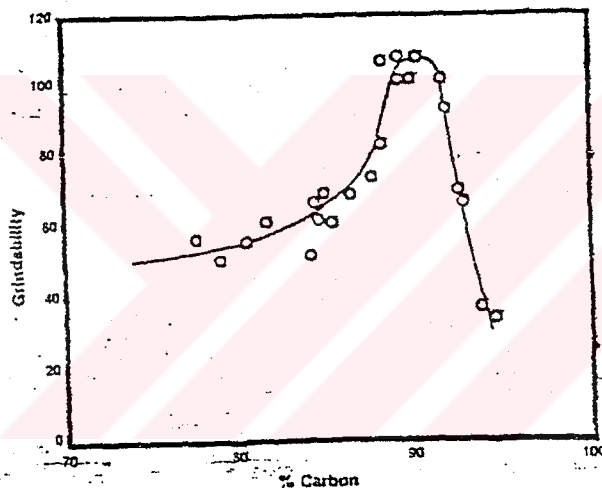


Fig 3.2- Hardgrove grindability index of coal (12)

The other physical properties of coal are optical, electrical, magnetic, hardness, breakage.

3.4 Analytical Chemistry of Coal

To collect data on the periphery of the coal molecule, i.e on the functional groups, one has to rely on analytical

methods. It is evident that the quantitative analysis of functional groups in coal is very difficult because of long reaction period and need to grinding (32). It is believed that only a minor part of oxygen in coal is present as non-reactive groups.

Hydroxyl Groups :

The hydroxyl groups in coals are predominantly phenolic or at least acidic in character, there is little evidence for the presence of alcoholic or very weakly acidic hydroxyl groups in coal. Brown coals may contain up to about 9 % hydroxyl oxygen.

Carboxyl Groups :

All investigations have shown that whereas carboxyl groups are absent in true coals, these groups occur in brown coals and lignites. At the carbon percentages of 72 % and 80, carboxyl oxygen contents of 5.1 and 0.3 % respectively. At 83 % carbon all- carboxyl groups have disappeared.

Methoxyl Groups :

True coals don't contain methoxyl groups. STENGEL found 2.8 % methoxyl oxygen in brown coal. At the carbon percentages of 65.72 and 80. Blom found methoxyl oxygen contents of 1.1, 0.4 and 0.2 % respectively.

Carbonyl Groups:

Carbonyl oxygen is found in all products of the coalification series. At 83 % carbon and higher, BLOM found carbonyl oxygen content is less profound than of the hydroxyl oxygen (15).

Nitrogen :

Nitrogen occurs almost completely in cyclic structures. The

proportion of the original nitrogen retained on oxidation in the insoluble residue is usually appreciably higher than the proportion of the carbon retained.

Sulphur :

Little is known with certainty about the distribution of organic sulphur groups in coal.

Organic sulphur in coal existed in four forms and that between 100°C and 300°C there is an evaluation of volatile sulphur compounds. Gusev stated that coal contains thioether and bio-thioether groups. Evidence for the existence of the thioether group rested on the reaction of the coal with methyl iodide, that for the bio-thioether groups on the reaction with cuprous-chloride.

3.5 Coal Impurities

3.5.1 Mineral Matter

All coals contain noncombustible mineral matter. The residue from these minerals after coal has been burned is called ash. Several empirical formulas have been developed for calculation of mineral matter from the ash content. The Parr formula is as follows :

$$1.1 \text{ ash} + 0.1 \text{ sulphur}$$

$$\text{Mineral Matter} = \dots\dots\dots \text{Eqn (1)}$$

The calculation of mineral matter from chemical composition of coal is as follows

$$\text{Mineral Matter} = 100 - (C+H+N+O+S) \dots\dots\dots \text{Eqn (2)}$$

Table 3.4 shows the mineral that may occur in bituminous coals. The species are listed in approximate order of the amount present with in each group.

3.5.2 Ash

As previously stated, ash is the residue derived from the mineral matter during complete incineration of coal. It differs in chemical composition and is usually less than the mineral matter originally present in the coal. The amount of ash normally is determined by burning a 1-2 gr sample of coal in a muffle furnace to 750-800°C (12,16).

3.6 Utilization of Coal

As far as utilization is concerned, coals can be divided into three broad categories :

1. Carbonization of coking coals for metallurgical purposes
2. Thermal energy or steam coals
3. Conversion coals

Currently, about 25 % of the world's coal production is used for coke making and almost all the remainder is thermal coal.

Fig 3.3 shows how rank and utilization are related and this chart also includes conversion coals, which currently represent a futuristic area for utilization; and special

Table 3.4 Principal Minerals In Coal Seams (22)

Minerals occurring in mudstone, 'shale', siltstone and sandstone	Quartz, SiO_2 Minerals of the clay group, principally: Kaolinite, $\text{Al}_2\text{Si}_2\text{O}_5(\text{OH})_2$ Montmorillonite, $\text{Al}_2\text{Si}_4\text{O}_{10}(\text{OH})_2 \cdot n\text{H}_2\text{O}$ with some Mg and Na Minerals of the mica group, principally: Muscovite, $\text{KAl}_2(\text{Si}_3\text{Al})\text{O}_{10}(\text{OH})_2$ Illite, $\text{K}_x\text{Al}_4(\text{Si}_{8-x}\text{Al}_x)\text{O}_{20}(\text{OH})_4$ Minerals of the chlorite group, general formula $(\text{Mg,Fe})_3(\text{OH})_6 \cdot (\text{Fe,Mg,Al})_3(\text{Si}_3\text{Al})\text{O}_{10}(\text{OH})_2$
Sulphide minerals	Pyrite ('iron pyrites'), FeS_2 Marcasite, FeS_2 Arsenopyrite (mispickel), FeAsS
Carbonate minerals	Siderite (chalybite), FeCO_3 Ankerite, $(\text{Ca, Mg, Fe})\text{CO}_3$ with some Mn Calcite, CaCO_3 Dolomite, $\text{CaMg}(\text{CO}_3)_2$
Chloride minerals	Halite, NaCl Sylvite, KCl
Other minerals	Fluorapatite, $\text{Ca}_{10}\text{F}_2(\text{PO}_4)_6$ Minerals of the feldspar group, including: Orthoclase, KAlSi_3O_8 Albite, $\text{NaAlSi}_3\text{O}_8$ Titanium minerals, including: Sphene, CaTiSiO_5 Rutile TiO_2 Ilmenite, FeTiO_3 Tourmaline, $(\text{H, Na, K, Fe, Mg})_3\text{Al}_3(\text{B,OH})_2\text{Si}_4\text{O}_{19}$ Barytes, BaSO_4 Minerals formed through weathering, including: Limonite, $2\text{Fe}_2\text{O}_3 \cdot 3\text{H}_2\text{O}$ Gypsum, $\text{CaSO}_4 \cdot 2\text{H}_2\text{O}$ Melanterite, $\text{FeSO}_4 \cdot 7\text{H}_2\text{O}$

coals, which represent only a minute portion of the total (22).

For power generation, almost any type of coal can be used, including the very low-rank lignites which are usually cheap to mine. The higher the rank of the coal, the higher its calorific value and also, the higher its mining cost.

The rarity of coking coals and the suitability of any type of coal, including non-coking coals, for power stations has led to a situation where international coal trade is dominated by the former. At present, little over 10 % of world coal demand is traded internationally, as compared with 75 % for oil, and 70 % of this coal trade is coking coal imported for use in the steel mills of the Far East, mainly Japan, Korea, and Europe.

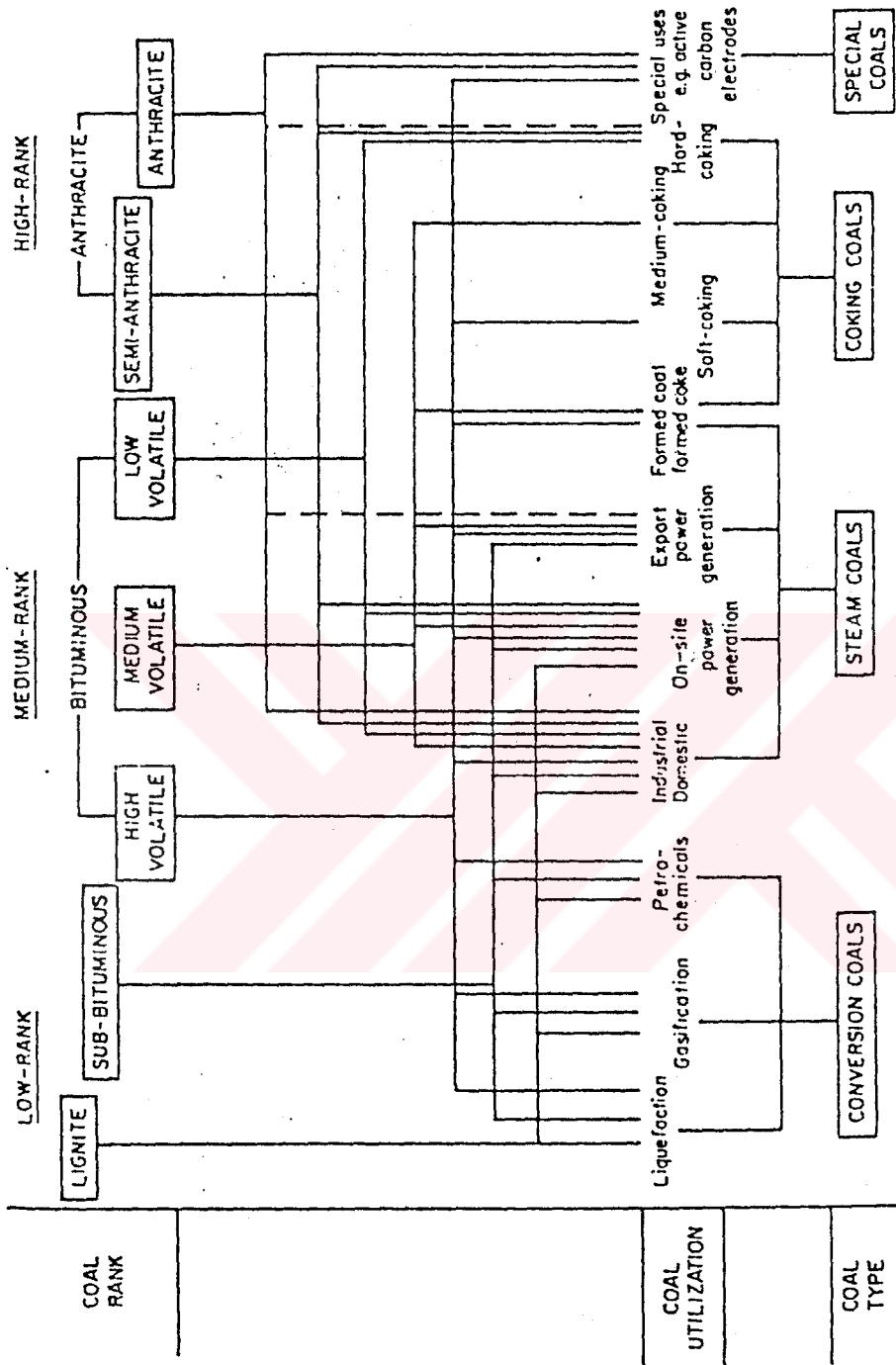


Fig 3.3 Chart showing relationship between coal rank and utilization (22)

4. COAL PREPARATION

The term "coal preparation" should be defined as including all work activities that are performed on all components of a run-of-mine coal from the time it leaves a surface reserve or an underground portal, until all components are either ready for shipment or are stowed on. This definition excludes the work that is performed to improve coal quality or handleability inside mine or pit. This work is commonly called "face" or "pit" preparation (14).

Gravimetric separators are most widely used in coal preparation. The function of all gravimetric separators is to segregate each particle in the feed into either a float or sink stream, and since the velocity with which each particle moves depends on its mass, particle mass affects the floatability of coal. Fig 3.1 shows washer efficiency as a function of mean particle size.

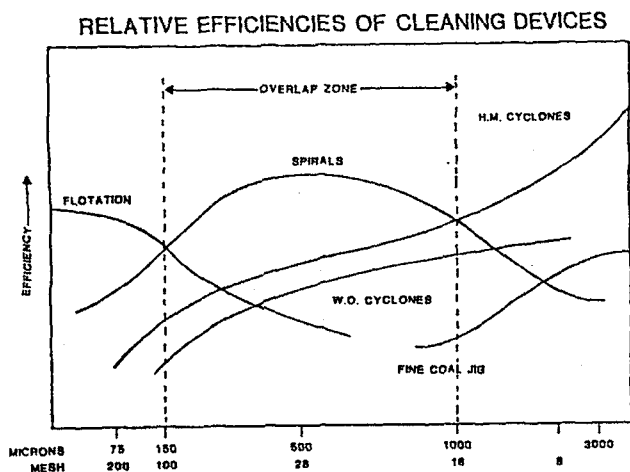


Fig 4.1- Graph showing efficiency as a function of particle size (14).

Float-sink tests provide washability data (see fig 4.2) vital for assessing the probable performance of separation equipment, which depends upon density differences; and froth flotation tests simulating the separation characteristics of the density separator provide valuable performance data for fine coal treatment.

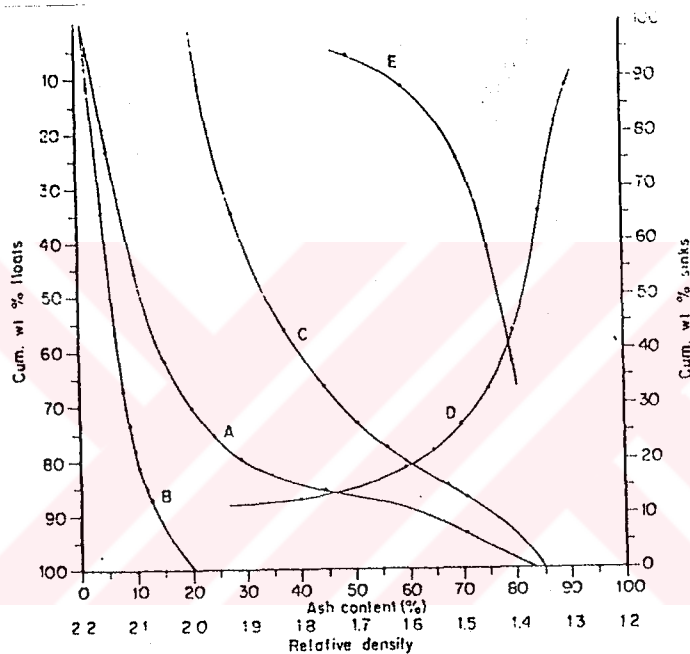


Fig 4.2-Washability curves:A, primary curve ;B, clean-coal curve; C,discard curve ; D,relative density curve E; 0.1 distribution curve.

All separation equipment has well-established empirically based performance characteristics which allow its performance to be determined once the washability data have been obtained.

Coal preparation processes can be classified into two groups depending on particle size (13)

1. Coarse coal cleaning

Wet separators : jig, dense medium, launder

Dry separators : jig

2. fine coal cleaning

Wet separators : jig, dense medium, launder, tables, hydrocyclone, flotation, agglomeration

Dry separators : jig, tables

4.1 Coarse Coal Cleaning

4.1.1 Dense Medium Separation

Commercial application of the dense medium process is a practical extension of the familiar laboratory float-sink tests, which is used as a standard gravimetric separation. Theoretically, any size particle can be treated by dense medium processes, practically the sizes treated range from about 28 mesh (0.5) to about 6 inches (150 mm).

Dense medium processes offer the following advantages over other coal cleaning processes:

1. Ability to make sharp separations at any specific gravity within the range normally required even in the presence of high percentages of the feed range.
2. Ability to maintain a separating gravity that can be controlled
3. Ability to handle a wide range of sizes up to 14 inches.

4. Relatively low capital and operating costs
5. Ability to change gravity of separation to meet varying market requirements.

Fig 4.3 shows a typical plant employing dense medium for coarse coal cleaning.

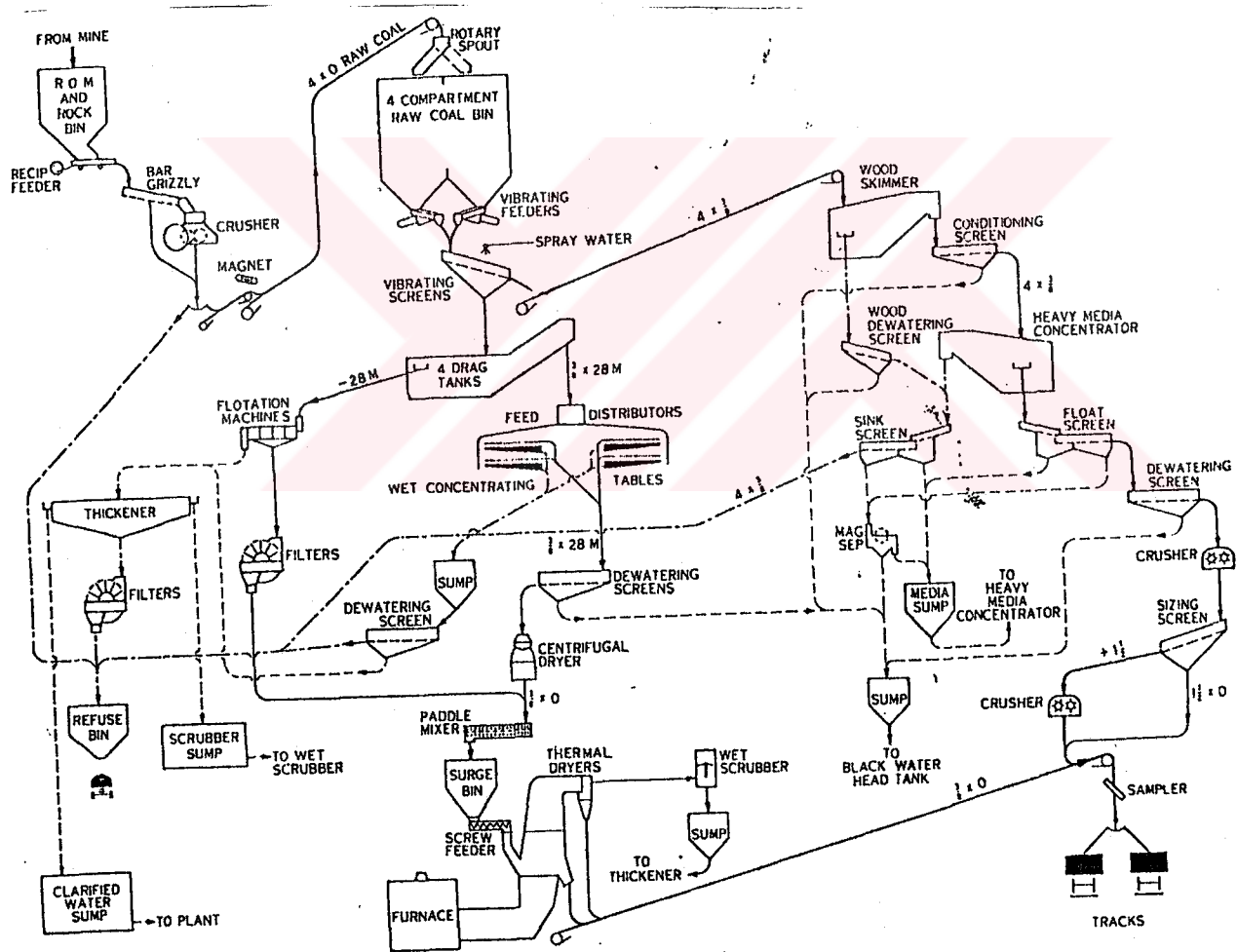


Fig.4.3 Typical three-process plant employing dense medium for coarse coal cleaning (22)

4.1.2 Hydraulic Separation Jigs

The most widely used separating device used to clean bituminous coal is the coal jig. Jigs used early in this century were usually of the plunger, diaphragm or vibrating basket which with new developments such as the Batac and Feldspar jigs, have increased the capacity and efficiency of the jiggling process for fine coals.

Baum jigs are usually limited to the treatment of raw coal within the size range of 150 mm to zero with an upper size of not less than 25 mm. Normally the Baum jig wouldn't be expected to upgrade raw coal finer in size than 0.5 mm.

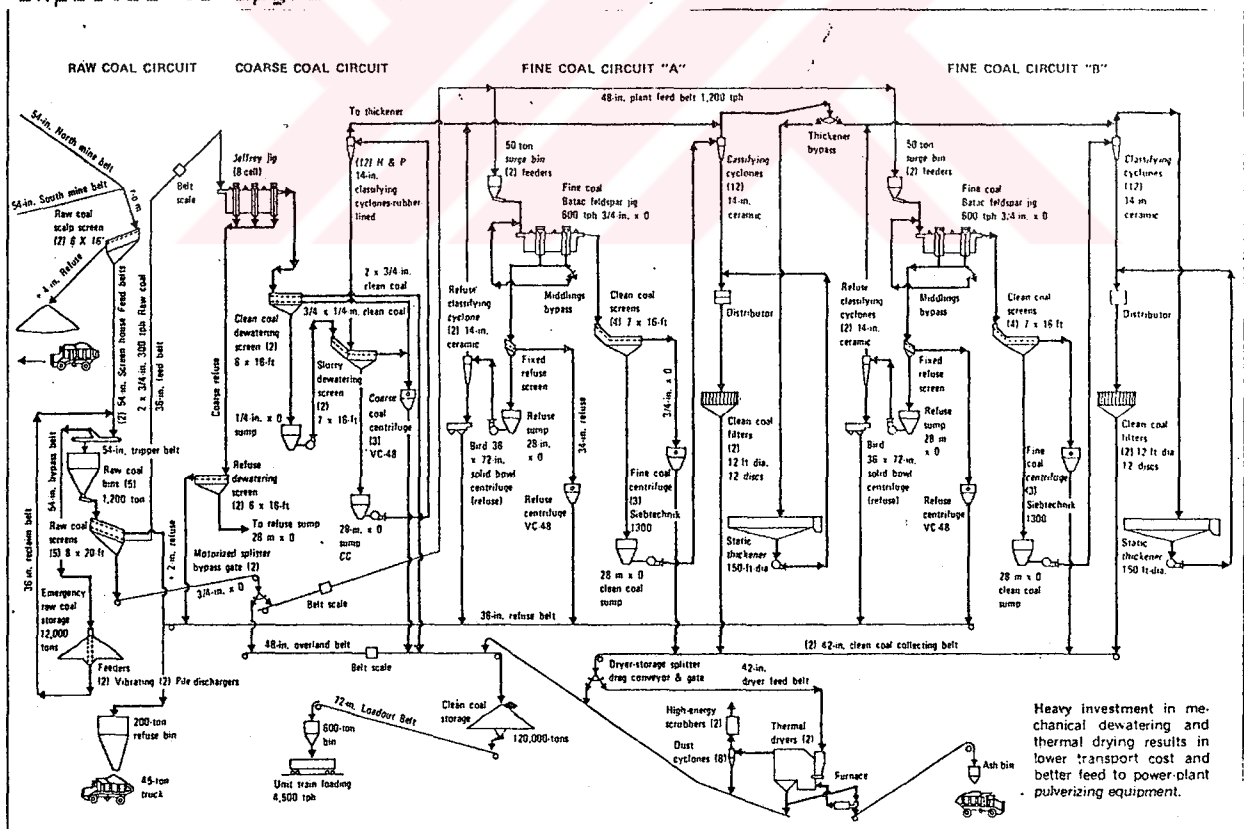


Fig 4.4 Schematic flowsheet for the Batac jig at the Greenwich Collieries (22).

4.2 Fine Coal Cleaning

Cleaning coal fines, which in the past was at best marginally profitable, is now an essential part of coal preparation strategy for a number of reasons.(14)

- 1- Raw coals now cost more to produce and hence there is incentive to maximize the recovery of salable coal.
- 2- Plant rejects, especially those that are water borne, now cost more to dispose of and hence there is incentive to minimize the loss of coal values to refuse.
- 3- The percentage of size finer than 28 mesh in the plant feed has increased dramatically as a result of intensive mechanization and by mixing, blending and storage and reclaim operations. More intensive coal processing also has increased the production of fines through degradation and decrepitation.
- 4- The quality of raw coal particles smaller than 28 Mesh has decreased significantly through the wide spread application of full seam, non selective mining systems.
- 5- Retreatment of middlings through crushing and recleaning has increased the final percentage of fines cleaned.
- 6- Coarse particle sizes no longer carry a premium price, as advances in combustion and carbonization technology now favor burning finer coals.

The techniques for cleaning 28 Mesh x 0 fine coal progressed from almost non existence to advanced technology in about 50 years . Fig 4.5 shows that proportion of saleable frothed

coal, in the U.K. has increased from the level of 0.4 % in 1950 to nearly 8 % in 1983.

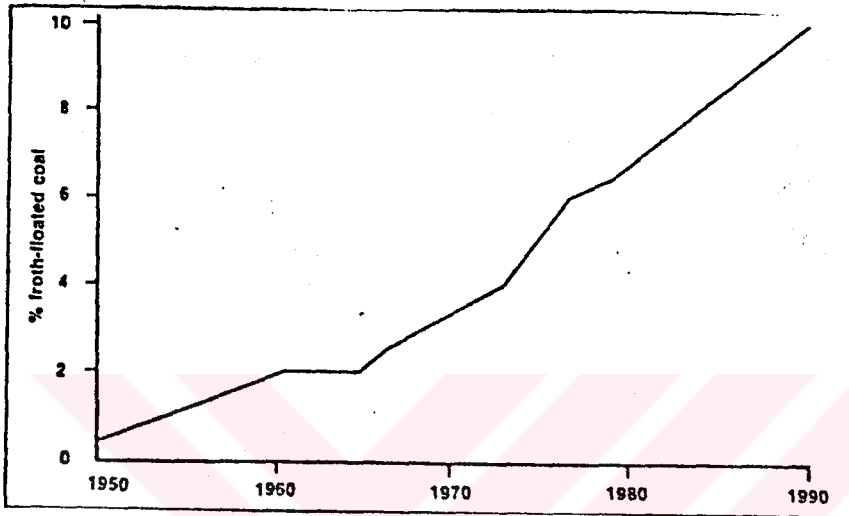


Fig 4.5- Proportion of froth floated saleable coal (31).

Modern technology applicable to fine coal- beneficiation except for froth flotation, is gravimetric by nature.

4.2.1 Dense Medium Cyclone

Cleaning fine coal in dense medium cyclones is becoming increasingly attractive for the following reasons :

1. Sharpness of cut compared to other gravity methods and ability to separate oxidized or weathered fine coal.
2. Need to eliminate maximum amount of pyritic sulphur

3. Improved performance of mechanical coal dewatering equipment.

4.2.2 Water Only Cyclones

The water only cyclone has, for many years, been widely applied in the treatment of fine coal sized below 0.5 mm. They are essentially designed with included angles of greater than 45° and long vortex finders and are used to clean the finer fraction of coal. The principal attractions have been two fold.

1. Reduction of pyrite from high-sulphur raw coal
2. Effective cleaning of fine coal with below 10% near density material.

Cleaning efficiency decreases sharply with increasing fineness in size and for this reason the cyclone is rarely effective below 0.1 mm in grain size.

4.2.3 Shaking Table

Under normal loading conditions a single table is able to treat between 9 and 12 t/h of 10 mm top-size coal feed. Shaking tables are most commonly supplied in multiple-deck form and units comprising two, three and four tables are available with differing combinations of drive mechanism.

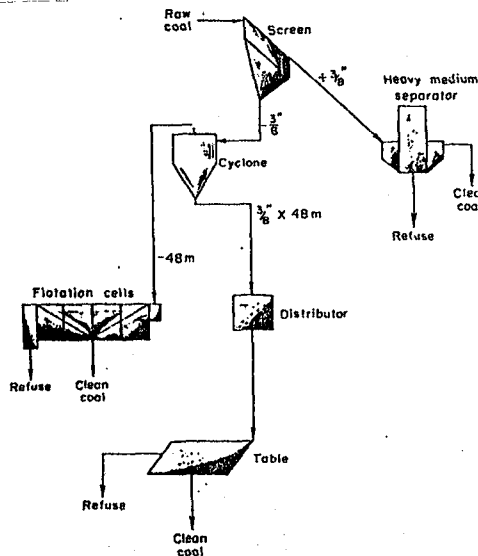


Fig 4.6- Typical modern flowscheme (6,22)

4.2.4 Feldspar Jig

The feldspar jig is a fine-coal washer used for cleaning (12 mm x 0) coal. This jig is actually an air-pulsated Baum type jig utilizing a bed of feldspar or some other hard rock of good cleavage characteristics.

The feldspar jig has a capacity of about 0.8-1.2 ton per in of with depending on the size and washability characteristics of feed.

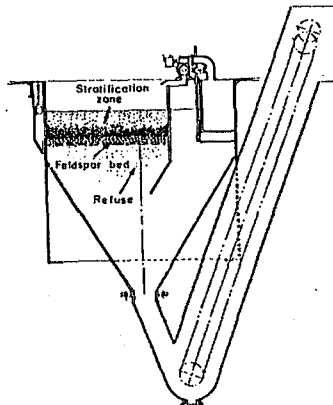


Fig 4.7- Cross- section through fine coal feldspar jig (22)

4.2.6 Flotation

The essential differences between ore and coal flotation is that for ores the entire tonnage is ground to flotation size, whereas for coal only that fine fraction not processed by gravity concentration is treated by flotation (9)

There are two reasons for this recent rapid growth rate :

1. Flotation is an excellent means for mitigating the preparation plant black water problem.
2. With substantial increases in the value of coal and its cost of production, the fines previously wasted become a very valuable product.

Since coal flotation is used for only a very small percentage of the total coal cleaned, it would be ridiculous to discuss capital and operating costs.

Besides, the environmental problems and the fact that was a cheap commodity, there are several reasons why flotation has not been more widely practiced (9).

1. Marketing problems with fines
2. Cost of dewatering
3. Indifferent test results in removing sulphur
4. Inability to make clean separations with finer sizes
5. Inability to clean slurries containing a high percentage of clay

4.2.6 Oil Agglomeration

Another method for cleaning the ultrafines is by the use of oil agglomeration. This method has been receiving wide attention in Canada, The United States, and else where.

Oil agglomeration is another name for bulk flotation. The amount of oil used will vary with the type of coal and its size consists; usually the quantity will vary from 3-25%.

The oil can be crude oil, fuel oil, or kerosene or some oil derivate coal tar oils are also effective. As in froth flotation, the separation of coal from shale is based upon the affinity of oil to coat coal surface and to be hydrophobic towards shale, clay and other like impurities.

In addition to the remarkable reduction in ash and sulfur the formation of the agglomerate aids in moisture reduction as the water drains off rapidly (22).

The agglomerate can be pelletized in a slowly revolving drum and moisture in these pellets has reached as low as 8-10% . This gives this process a potential advantage of possibly eliminating vacuum filtration for dewatering concentrates (22).

5. FLOTATION

5.1 Natural Floatability of Coal

5.1.1 Contact Angle

For flotation to occur, an air bubble must become attached to a solid surface of a particle immersed in water. The extent to which this interaction occurs is characterized by contact angle (22,24,32). Bubble particle contact therefore stabilizes to an equilibrium contact between air bubble and solid. The contact angle, θ , is the angle formed between the liquid-solid and liquid-gas interfaces measured through the liquid as shown in fig 5.1.

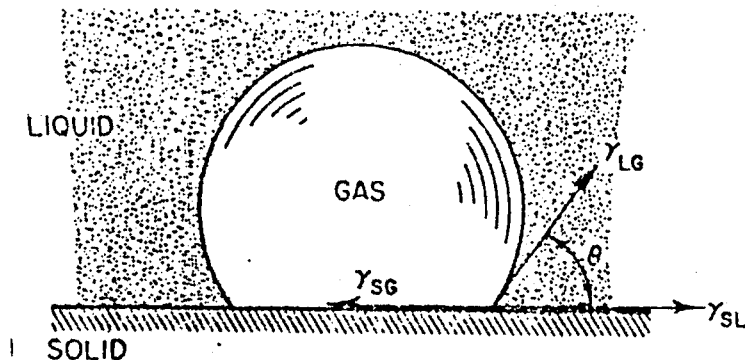


Fig 5.1- Equilibrium contact angle (22).

The general thermodynamic condition for this phase contact is defined by Young's equation to be

$$\gamma_{sg} = \gamma_{sl} + \gamma_{lg} \cos \theta \quad \dots\dots \text{Eqn (3)}$$

Where γ_{sg} , γ_{sl} and γ_{lg} are the tensions of each of the solid-gas, solid-liquid and liquid-gas interfaces. Contact angle can be measured by experimental procedures.

Brady and Gauger evaluated the natural floatability of coal by using the contact angle. Brown plotted also a curve for the variation of contact angle with carbon content of coal. The maximum contact angle was observed at 89% C. Up to this value, there is a loss of polar character by the decrease of hydroxyl and acid groups so that the coal becomes more hydrophobic and floatable with a maximum at 89% C, above 89% C the hydrophobicity decreases slightly as the carbon atoms form structures with increasing order in three dimensions. Fig 5.3 shows the dependency of contact angle on rank (9,22,24).

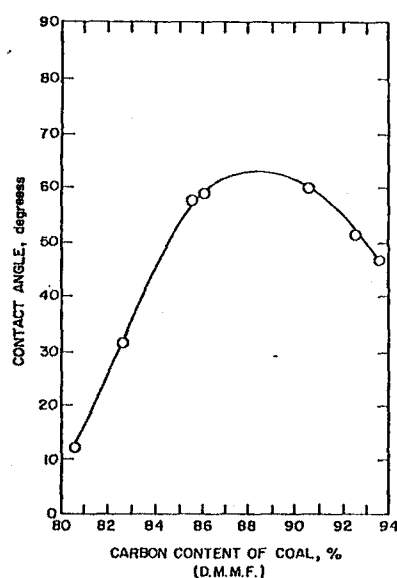


Fig 5.2- Variation of contact angle with carbon content of coal

The contact angle and floatability are also functions of the ash content of the coal mineral and decreases in order to vitrain, clarain, durain and fusain (9). Table 5.1 shows the contact angles of a Turkish bituminous coal seam and its lithotypes (23,24).

Table 5.1 Natural contact angles at various coal surfaces (23)

Sample	Contact Angles (Degrees)		
	Min	Max	Average
Vitrain	57	63	61
Durain	24	47	42
Fusain	--	--	--
Cay channel sample	26	53	49
Acılık channel s	28	50	47
Piric channel s	28	59	55

5.2.2 Hydrophobicity of Coal Particles

Contact angles have long been used to assess the wetting behavior of coals. But because of the heterogeneity of coal, conventional contact angle measurements may yield questionable information about the hydrophobicity of coals (20). From a more practical point of view, assessment of hydrophobicity/hydrophilicity through study of the wetting characteristics of a mass of particles may provide a better measure of the average wetting behavior and, perhaps, provide a more realistic correlation with the performance of various technological processes. One such technique involves the rate

at which a powder transfers from a gaseous phase into a liquid. In an investigation of the action of surface-active agents in coal dust abatement, Walker et al. studied wettability by determining the surface tension of the solution into which individual particles would be imbibed immediately when dropped from a 1 cm height. Fuerstenau et al. (1986) correlated immersion time measurements with salt flotation response and the extent of surface oxidation of various coals.

Williams and Fuerstenau (1986) developed a method which is a film flotation process, a monolayer of closely sized particles is placed on the surface of a liquid in a shallow vessel of 8 cm diameter and 2 or 3 cm depth. In general, the solids either remain on the liquid surface or are immediately imbibed. Thus, for closely-graded, monosized coal powder fractions, times of immersion need not be measured but rather after film flotation of the solids, the solid material is split into its hydrophobic and hydrophilic fractions which are then dried to constant weight.

Fig 5.4 illustrates the film flotation response of 100x150 mesh particles of Dave Johnston coal. Three parameters from the kinds of results given in Fig 5.4

1. $\gamma_{c \min}$: the surface tension of the solution that wets all particles.
2. $\bar{\gamma}$: the mean surface tension of particles in the distribution.

3. $\gamma_{c \max}$: the surface tension of the solution in which none of the particles are wet.

Using data from 5.4, the mass fraction of the particles which are hydrophilic is plotted in Fig 5.5 as a function of the surface tension of the imbibing solution. The average surface tension parameter for each distribution. $\bar{\gamma}$ can be calculated from the histogram using the equation

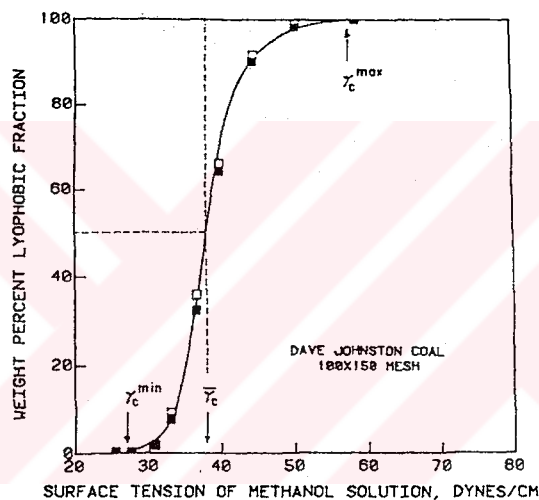


Fig 5.3- Partition curve for the film flotation 100x150 mesh particles of Dave Johnston subbituminous coal in aqueous methanol solutions (38).

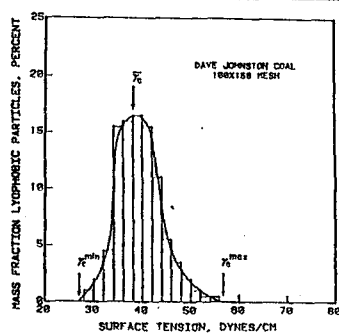


Fig 5.4- Frequency histogram for the lyophobicity of Dave

Johnston coal obtained with the film flotation cell (38).

$$\gamma_c = \gamma_L f(L) d_L / d_L \dots\dots \text{Eqn (4)}$$

Where γ_L is the surface tension of the given liquid and $f(L)$ is the frequency distribution for the solid.

In summary, film flotation technique where in about a monolayer of particles is placed on the surface a series of solutions having different surface tensions, and the fraction of particles that sink is then determined. The fraction that sinks has only a small dependence on particle size. From these data a cumulative distribution of particles versus their hydrophobicity can be made, analogous to the washability curve used to assess gravity separations in coal cleaning. The mean surface tension of wetting appears to correlate well with the maximum flotation response of the coal (20).

5.3 Theories For the Flotation of Coal

Three theories have been suggested for the explanation of different floatabilities of various coal.

5.3.1 Carbon-Hydrogen Ratio Theory

Taggart et al. explained the difference in floatability between bituminous and anthracite coal by the difference in C/H ratio. This is not applicable to the relative floatability of other coals and carbons (24,12).

5.3.2 Carbon Content Theory

Wilkins reported that the floatability of coals increased with increase of the carbon content or rank. However, bituminous coals that possess moderate carbon contents are actually more floatable than anthracite coals that have higher carbon contents.

5.3.3 Surface-Components Theory

Sun developed a surface component theory which correlates bulk chemical analyses with the surface hydrophobicity. The substance of this theory is that these heterogeneous materials consist of both floatable and non floatable chemical constituents and that their floatabilities are governed by the balance between these two groups of components. The floatable constituents are considered oil-avid, whereas the non-floatable constituents are oil-repellent.

This hypothesis takes into account of hydrocarbon molecules, excess carbon atoms, water molecules and the relative number sulphur, oxygen and nitrogen atoms. Mathematically the hypothesis is expressed as follows (12,24) :

$$F_c = x (H/2.0796) + y (C/12 - H/2.0796) + 0.4 (S/32.06 - z M/18) - 3.4 (O/16) - (N/14) \dots\dots\dots \text{Eqn (5)}$$

In which F_c is calculated floatability representing the balance between the floatable and the non-floatable surface

components. The symbol H,C,S,N,O and M are respectively the weight percent of hydrogen, carbon, sulphur, nitrogen and moisture. The assumed numerical values of factors x, y, z are given in table 5.2 (4,29).

Table 5.2 The assumed numerical values of factors x,y,z for Sun's surface component theory (24).

Ash	Remnant Carbon			Moisture	
Ash% as received	x	Hydrogen%	y	Moisture% as received	z
0 to 8.9	3.5	0.08 - 0.28	1.2	0 to 14.1	4
9 to 13.0	3.0	0.29 - 1.0	0.8	14.1	3
13.0	2.5	1.0	0.6		

Sun's surface component theory has two defects:

1. Complete chemical analyses are rarely available
2. Since the analyses are based on the bulk sample, surface oxidation can completely destroy the estimate because surface oxidation leads to poor flotation.

5.3 The Flotation System

The successful industrial practice of froth flotation involves knowledge of how the three major components (chemistry, equipment and operation mode) of the flotation "system" interact with each other in a given plant situation. The terminology "system" implies that there are many factors

involved that are highly inter-related to each other so that it is difficult to study the influence of a given factor by it self. Fig 5.6 has been used extensively to present this system's concept (29,14).

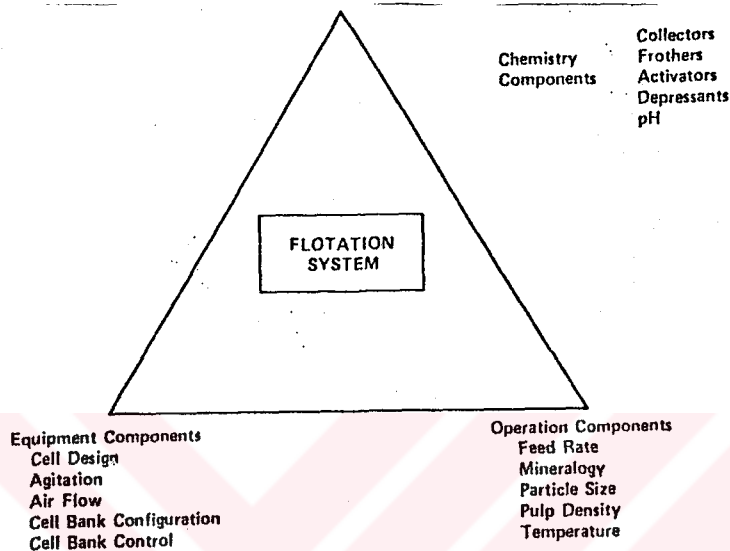


Fig 5.5- The process of flotation illustrated as a three-cornered interactive system.

Fig 5.5 shows is that there are three major categories of components that are involved :

1. Equipment
2. Chemistry
3. Operation

Each component has a large number of factors that can be set. A crude approximation to assume that chemical component factors such as frother, collector, depressant, etc. are independent each other (8). Klimpel concluded that chemistry component is of much greater significance than often realized (29).

5.4 Factors Affecting Coal Flotation

5.4.1 Affects of Chemical Components

Since coal is often naturally floatable, the material need only a "nudge" to float readily. There are mainly four types of reagents used in coal flotation; nonpolar oils, heteropolar reagents, inorganic electrolytes and depressants.

5.4.1.1 Non-polar oils (Collectors)

The most effective collectors that can be employed in coal flotation are those oils with low solubility in water derived from petroleum, wood and coals.

Typical nonpolar reagents often termed oils are the kerosenes, fuel oil, crude petroleum and certain distillates. Nonpolar reagents give no reaction with water dipoles and have pronounced water-repellent properties due to the presence of weak Van der Waals forces between the hydrocarbon molecules and covalent no solidophil groups which are links between the radicals of the reagent and the mineral surface in their structure, they don't form oriented adsorption layers on the mineral surfaces. Their attachment is based on adhesion. Klassen describes the mechanism of oil attachment to coal surface as film formation. During film formation the reagents don't react chemically with the mineral surface and their chemical composition doesn't change.

High rank bituminous coals frequently requires only a frother while intermediate to low rank bituminous coals may require from 1 to 3 lb/t of oil for good recovery, and lignite requires either large amounts of oil or may be only poorly collected. The average quantity of oily collector is an about 1.7 lb/t (31).

Kerosene is one of the most widely used oily collectors in the world. The composition of kerosene depends to a large extent on the source of the naphta from which it is derived. Light oils, those of low molecular weight, spread easily over the surface of the coal, but tend to be absorbed too quickly into its porous surface. Heavily oils, require considerable agitation or conditioning to spread the droplets over the surface but they are more effective as they stay on the surface, showing less tendency to be absorbed.

Another widely used oily collector which is not derived from petroleum is creosote. Industrially used creosote is a crude phenolic mixture which is usually of coal-tar origin (24). For oxidized or weathered coals, a long chain amine has been found to be an effective collector (9).

While addition of collector improves the flotation of recovery, it is seen to increase the amount of ash content in the froth (3,31).

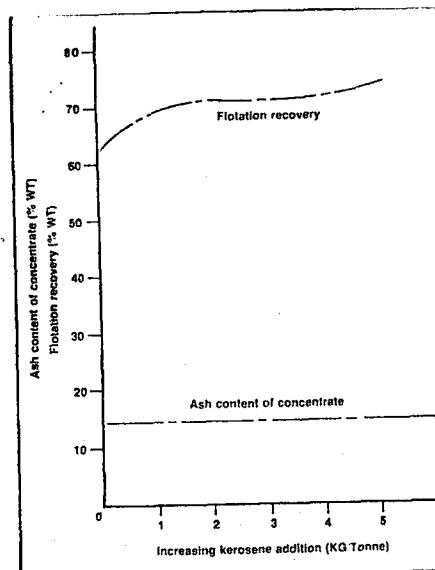


Fig 5.6 Graph showing recovery/reagent addition (31).

(frother addition constant at 0.1 kg/t)

5.4.1.1.1 Effect Of Various Hydrocarbon Oils As Collectors

Viscosity is an important factor which influence the dispersion of oil in the flotation system. For that reason, viscosity can be used as a criterion to relate the oils to each other. Table 5.3 shows viscosity values of some oils (31).

Table 5.3 Viscosity values of some oils (31)

Commercial name	Abbreviated name	Viscosity (cps)
Kerosene	Kerosene	3.0
Distillate	Diesel	4.6
Light spindle oil	LSO	48.2
Medium spindle oil	MSO	117.0
Heavy medium spindle	HMSO	152.0
Heavy spindle oil	HSO	268.0
Very heavy spindle oil	VHSO	627.0

Townsend (1985) determined that there is a relationship between percentage recovery and viscosity (fig 5.7) (22,31).

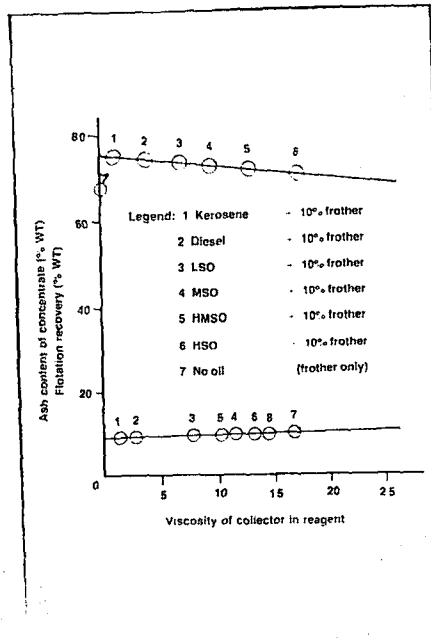


Fig 5.7 Graph showing viscosity of collector in reagent/recovery (31).

As it is seen from this graph, diesel oil having a higher viscosity than kerosene takes a longer time to disperse in the flotation system. Nevertheless, all the hydrocarbon oils investigated have been noted to have produced higher recovery values than the condition when no oil, only frother is used (31).

5.4.1.1.2 Effect Of Collector Dosage

As it is seen from fig 5.8 b, rate constant K , increases with increase in the dosage of collector added up to certain level and then decreases showing a maximum.

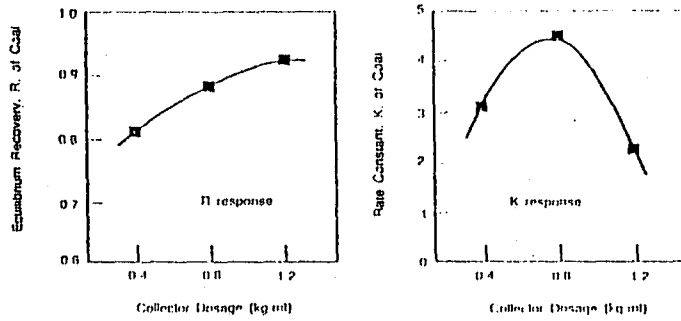


Fig 5.8 Influence of changing collector dosage (14).

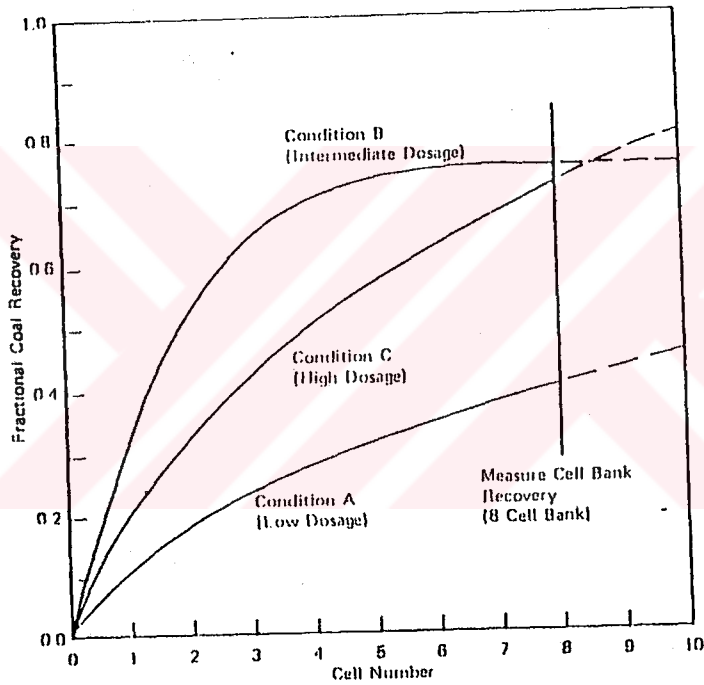


Fig 5.9 Illustration of the R/K trade-off as a function of collector dosage holding all other variables constant (14).

Fig 5.9 shows that the high collector dosage has lowered the rate K sufficiently.

5.4.1.2 Frothers (Heteropolar Reagents)

In coal flotation, the most widely used heteropolar reagents are frothers. The main purpose of frothing agents is to facilitate the production of stable froths. Only substances which can change the surface tension of water can be frothers (12). They have structural formulas that are characterized by the presence of two opposing constituents, one part of the molecule being nonpolar (repels water) and the other polar (hydrolyzes). They are therefore heteropolar. Heteropolar reagents make the coal surface more hydrophobic if they are added at low concentration (24).

In the flotation of coal, monohydric aliphatic alcohols and monocarboxylic acids are good collectors. When the hydrocarbon radical contain between five to ten carbon atoms, with an optimum of eight (24,31).

Table 5.4 Alcohols used in coal flotation and their carbon chain length (31).

Alcohol	Carbon chain length
Butyl	4
Amyl	5
Hexyl	6
Heptyl	7
Octyl	8
MBIC	6
2 ethyl hexanol	8
Phenol	6
d- Cresol	6
B Napthol	10
x Terpeneol	10
Cyclomexanol	6
Diacetone alcohol	6
Ethers suitable for use as frothers	
Ether	Number available
Polyoxpropylene Glycols	At least 4
Polyoxethylene Glycols	At least 3

We can see from fig 5.10 that optimum recovery can be effected by matching the carbon chain length of the alcohol used to each coal surface.

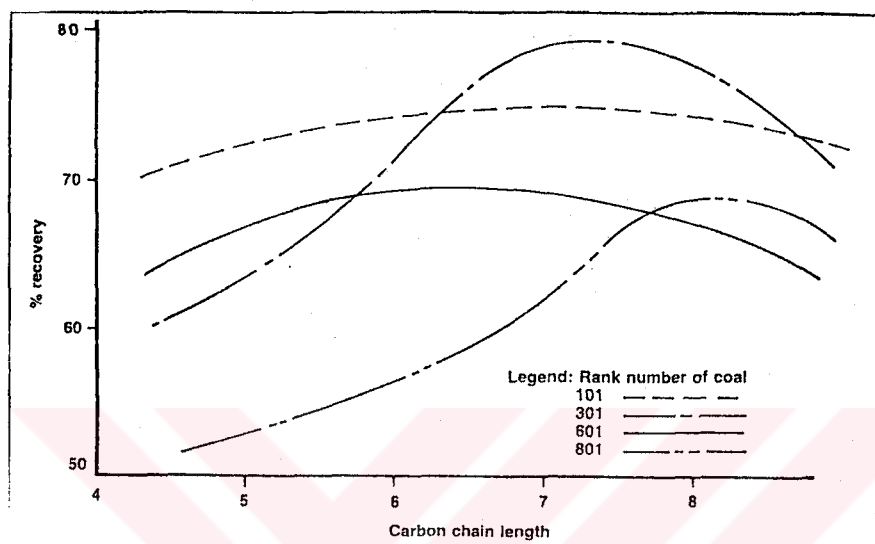


Fig 5.10 Graph showing effect of carbon chain lengths of frother material on differing coals in terms of recovery (3)

Frequently 0.1 to 0.5 lb/t frother is used (9).

As it is shown in fig 5.10, amyl, hexyl, heptyl and octyl alcohols are examples of frothers. Methyl isobutyl carbinol (MIBC) is also an aliphatic alcohol in common use. It has both frothing and collecting properties. Klimpel and Hansen found that MIBC was more effective for slightly finer sized particles (27). Terpeneol, pine oil, cresylic acid are especially attractive.

The application of ionic collectors such as xanthates, carboxylic acids, sulfates and sulfonates in coal flotation is not more than of a scientific interest. sun found certain cationic collectors, laurylamine, rosin amine-D acetate and iso-amyl amine to be effective in the flotation of oxidized coals. This may be due to the fact that chemical reaction and ionic attraction can takes place between the positively charged polar groups of amines and the negatively charged ions, particularly carboxyl groups of the oxidized surfaces of the coal (24).

Figures 5.11, 5.12 and 5.13 also show is that all the different frothers listed also demonstrate R/K trade-off phenomena as a function of changing frother dosage.

It can be seen from these figures that the alcohol frothers (e.g MIBC) tend to be much more effective and selective fine particle frothers but poor coarse particle frothers and conversely, the polyglycol methyl ether frothers are much more effective for coarse particle recovery but give unselective fine particle recovery. Related to this observation is the tendency of change in collector dosage (e.g soltrol hydrocarbon) in the presence of alcohol frothers to demonstrate a much bigger R/K trade off than the same collector changes with glycol type frothers (e.g compare the MIBC) calculated rate numbers indicated above each column to that of the two major glycol frother used in industry (DF 400 with molecular weight 400 and DF 1012 with molecular weight 400). This phenomena has also been documented at the plant

Figure 5.11

The Performance Of Various Engines At A 0.2 minute Turn On The Recovery Of 1.35 U.S. Barrel Coal/Min Engines As A Function Of Fractional Recovery Of Coal Through Various Engines (As Specified In Table 5.1)



Figure 5.13

The Performance Of Various Engines At A 0.2 minute Turn On The Recovery Of 1.35 U.S. Barrel Coal/Min Engines As A Function Of Fractional Recovery Of Coal Through Various Engines (As Specified In Table 5.1)

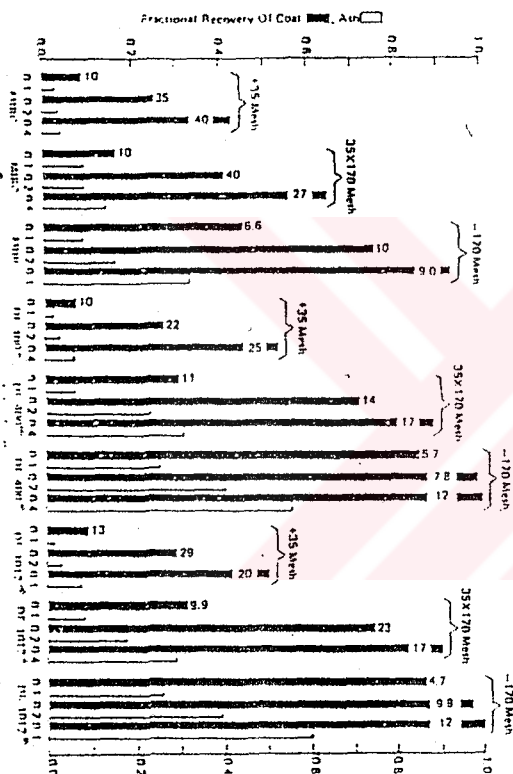


Figure 5.12

The Performance Of Various Engines At A 0.2 minute Turn On The Recovery Of 1.35 U.S. Barrel Coal/Min Engines As A Function Of Fractional Recovery Of Coal Through Various Engines (As Specified In Table 5.1)

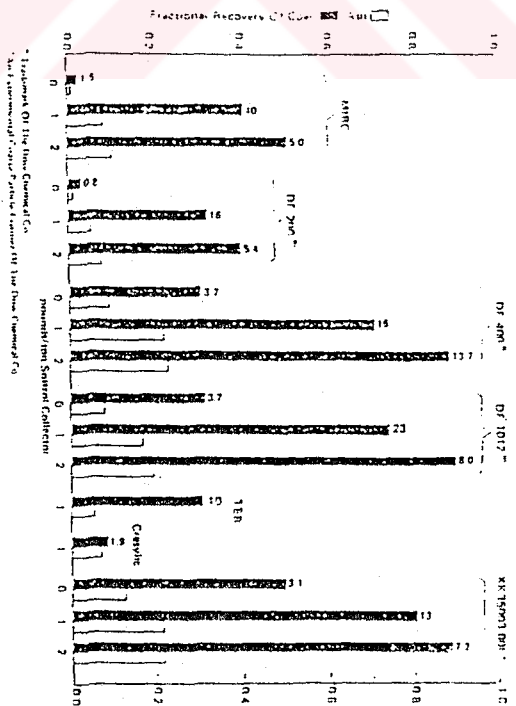


Figure 5.14

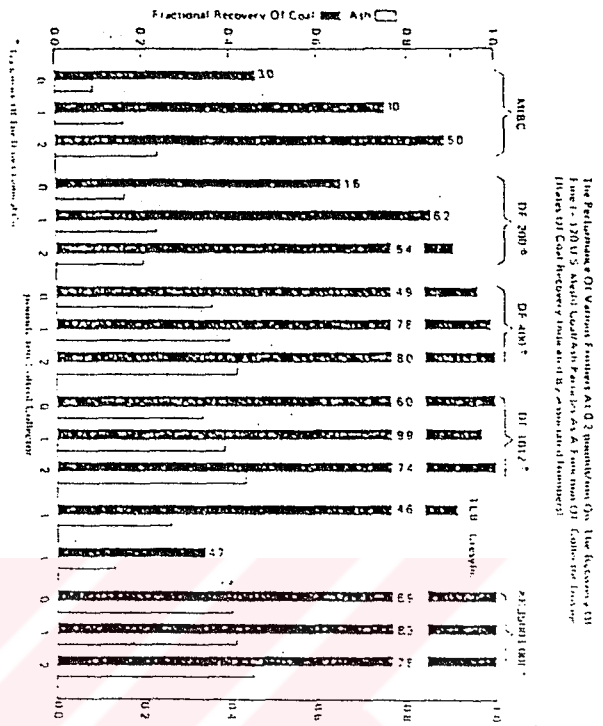


Figure 5.15

Recovery Of Coarse (-20 U.S. Mesh) And Total Coal Ash As A Function Of Frother Type (Amounts In (1) Indicate Selectivity Values Of Coal Over Ash)

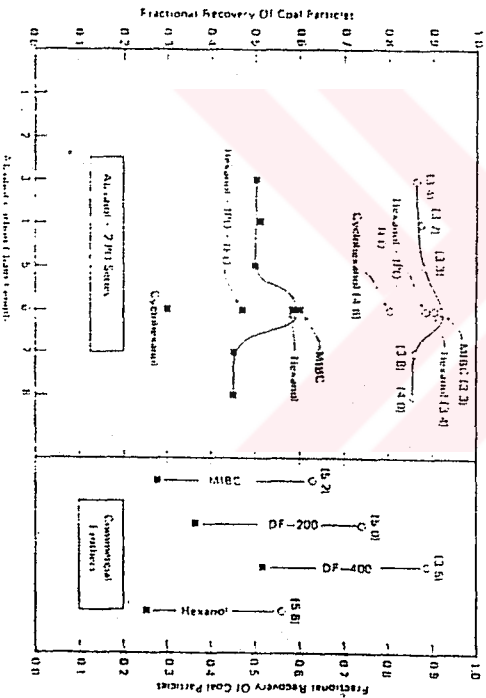
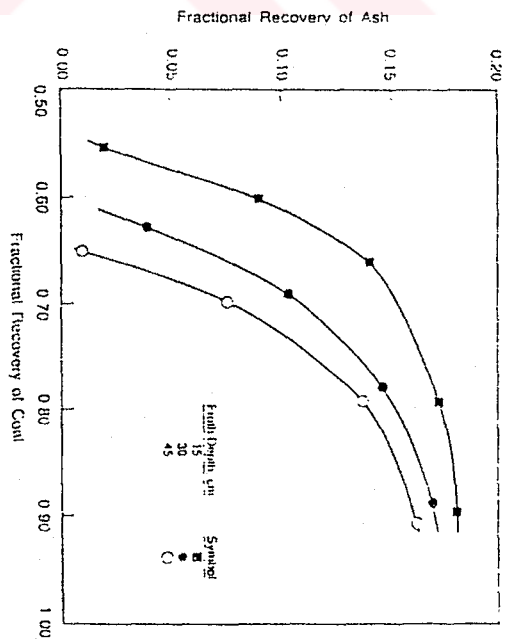


Figure 5.16

The Influence of Froth Depth on Coal Recovery and Selectivity Using DOW 1012 Frother and by Manipulating Appropriate Frother Parameters Such As Feed Rate, Pulp Level, and Dosage to Create Froth Depths Indicated.



level lowering the molecular weight of the glycol frothers from 400 to 200 makes the glycol frother look more like the alcohol in both its size range of effectiveness as well as the R/K trade-off sensitivity (14).

Another important practical observation is documented in Fig 5.14 as used to measure the data of fig 5.11, 12, 13. In this figure, the collector dosage is held constant and the frother dosage is varied. This commonly observed plant observation that if an operator attempts to use frother dosage as a means to drive an under collected coal flotation rather than to optimize the proper collector dosage along with the frother dosage; the result is often disastrous on selectivity.

Fig 5.15 shows a series of frother products made with alcohol 2PD adducts are improved coarse particle floaters over typical commercial products with the max occurring at a six carbon alcohol (14,29).

5.4.1.2.1 The Effect Of Froth Depth On The Quality Of Coal Floates

This is an important variable as is shown in fig 5.16. Froth depth and rate of froth removal are parameters often used by operators as a means of visual control to increase selectivity and, as such, are probably some of the more effective means of controlling selectivity as long as other factors like changing feed rate, changing collector dosage.

5.4.1.3 Inorganic Electrolytes

It is possible to float many naturally hydrophobic minerals in solutions of inorganic salts without additions of any other reagents. At low concentrations, the electrolyte makes the mineral particles hydrophobic by decreasing surface potential and thus decreasing their hydration. At higher concentrations, the electrolyte causes froth formation (24).

Coal has a considerable capacity for adsorption of electrolytes. Electrolytes change the electrostatic charge at the coal surface more or less hydrophobic by the consequent reduction or increases of charge.

Only nonpolar minerals, and above all the coals, can be floated in 1-3 % solutions of calcium chloride, sodium chloride and similar salts. Waste from soda works has great value for this purpose (26).

Klassen et al. showed that additions of sodium chloride improved the attachment of coal particles to air bubbles. The use of electrolytes in coal flotation known as "salt flotation" was not widely developed because of their corrosive properties but today shortages of fresh water or the poor qualities of some called fresh waters have forced several mills to conduct flotation in sea water (24).

It is interesting to note that the use of sea-water yields a drier froth in the concentrate with a water content of 30 to 35 percent while flotation with nonpolar reagents, the water content is around 50 to 60 percent.

Combined addition of inorganic electrolytes and nonpolar reagents are especially important in coal flotation practice. The addition of a very small amount of kerosene sharply increased both the rate of flotation and the recovery.

5.4.1.4 Emulsifiers

Emulsifiers generally increase coal flotation yields by dispersing the collector oil into finely divided droplets throughout the flotation pulp. The increased number of oil droplets aids flotation kinetics. Emulsifiers also lower the oil/water interfacial tension which in turn lowers the energy required to spread the collector oil, across the coal surface, increasing its hydrophobicity and therefore the adhesion of the air bubble to the coal particle (21).

Emulsifiers are divided into two groups ; ionic and nonionic. Both non-ionic and anionic emulsifying agents were initially investigated and the flotation results compared with those obtained using the conventional hydrocarbon collector/frother reagents together with creosote, cresylic acid systems. fig 5.17 and 5.18 show results for both ash content and combustible recovery against reagent dosage.

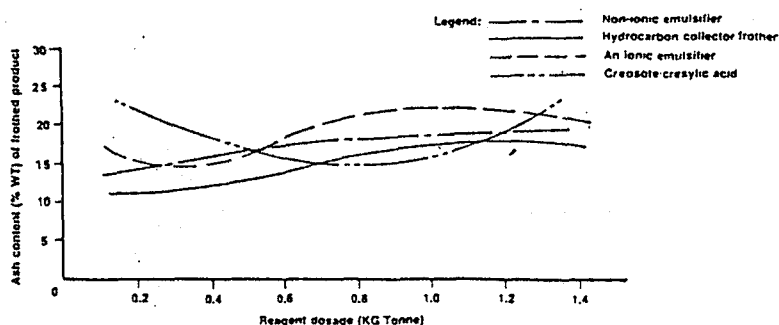


Fig 5.17 Graph showing effects of additions of emulsifying reagents into on ash content of frothed product (31).

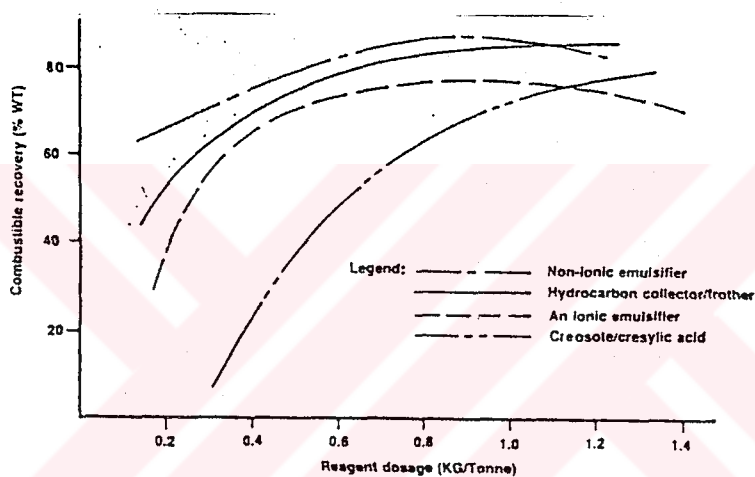


Fig 5.18 Graph showing effects of additions of emulsifying agents into reagents in terms of recovery (31).

It can be seen that for the non-ionic emulsifier, significantly combustible reagent dosage were required to produce the same combustible recovery as that for the conventional system. Anionic emulsification produced inferior results than both of the above, while the creosote/cresylic acid system, although offering the cheapest reagent unit cost, required very much higher dosage levels (31).

In fig 5. 18, at dosages in excess of 1.0 kg/t, very little difference in combustible recovery is apparent. at these levels, anionic emulsification indicated reduced recoveries.

5.4.1.4.1 Non-ionic Emulsifiers

A large proportion of commercially available emulsifiers are non-ionic and consist of an alkyl chain bonded to a polyethoxly chain of variable length. These molecules attach at the oil/ water interface with the ethoxly chain directed towards the aqueous phase, which at monolayer coverage, forms a hydrophilic surface. Accordingly, the addition of excess emulsifier to the flotation system could retard the flotation process between the air bubble and this binary surface (21).

Moxon (1986) did some tests on the run-of-mine sample of Great Northern seam coal using by polyethylene nonyl phenol $\{ C_9 H_{19} (C_6 H_4) - (OCH_2 CH_2)_5 - OH \}$ as surfactant dodecane as collector (1 kg/t) and MIBC as frother (10 gr/t). The results of these tests plotted in fig 5.19

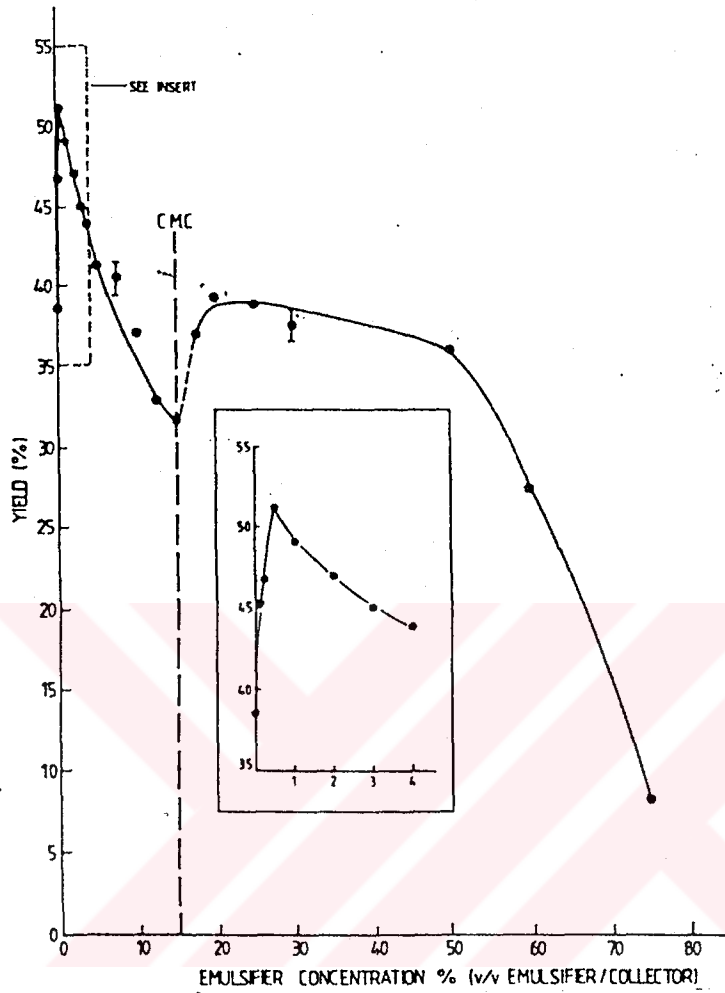


Fig 5.19 Variation of yield with emulsifier concentration (21).

Fig 5.19 indicate a maximum improvement in yield at a surfactant addition of approximately 0.5 % (v/v emulsifier/collector). This corresponded to a yield increase of approximately 12.7% at only a slight increase in ash (i.e 1%)

5.4.1.5 Depressants

In coal flotation, it is possible to depress either the coal or the associated gangue minerals. For coal depression starches, especially dextrin and the natural hydrocolloids have been found to be effective. Inorganic salts such as stannous chloride, hypophosphorous acid and potassium permanganate are also depressants. Any reagent which will oxidize the surface of the coal, thus destroying its natural hydrophobicity, may be considered as a potential coal depressant. pH may also be considered for coal depression, since recovery drops off at low and high pH values (9).

Since high slurries are normally clays and silicates, especially quartz standard depressants for this class of minerals such as sodium silicate, phosphates especially hexametaphosphate, lignin and other polymeric sulfonates, quebracho and tannis may be considered. Sodium cyanide, lime and the oxidation products of pyrite, ferrous and ferric sulfate are used to depress the pyrite (4,31).

Arnold and Aplan (1985) showed that Calgon, Masperse and sodium silicate depress the flotation of coal (3).

5.4.7 Oxidation

Oxidation of coals negatively effect their floatability. In general, freshly-mined coal floats better than coal which has been exposed to the atmosphere.

It has been reported that coal oxidation seems to proceed in three stages (5). In the first stage coal forms complexes with oxygen due to superficial oxidation. In the second stage, substances like humic acid or hydrocarboxyl acids are formed and in the third stage water oxidation of coal results in the formation of ether linkages and strongly acidic carboxylic or phenolic groups. These kinds of oxygen containing groups have also been reported for activated carbons.

$$\text{The hydrophilicity index} = \frac{(-\text{COOH}) + 2(-\text{OH})}{(\text{R-H}) + (\text{Ar-H})} \dots\dots\dots \text{Eqn (6)}$$

Where $(-\text{COOH})$, $(-\text{OH})$, (R-H) , (Ar-H) are the values of the absorption intensity (5).

Fuerstenau et al. (1986) carried out vacuum flotation experiments in salt solutions (0.5 NaCl) on 48x65 mesh Cambria coal. Sample was thermally oxidized in air in a natural convection oven by heating the samples at 150, 200 and 244 C for periods of 19 hours. Results are shown in fig 5.20 that oxidation makes the coal more hydrophilic and reduces the flotation yields markedly (38).

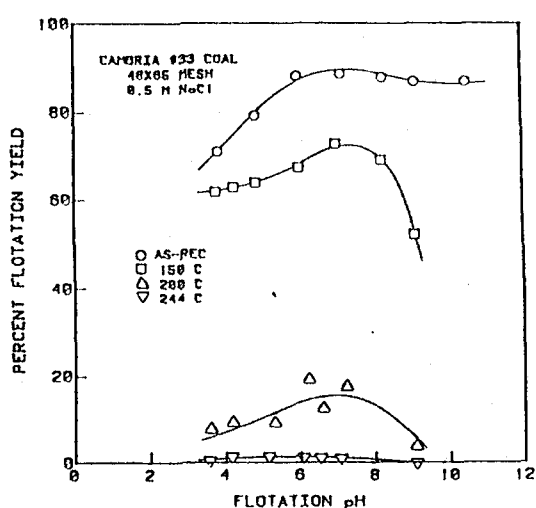


Fig 5.20 Vacuum flotation recovery of 48x 65 mesh particles of Cambria coal after various oxidation treatments. Flotation was carried out for 1 min in 0.5 M NaCl solutions (38).

The resistivity of coals to oxidation increases with their rank. The lower rank bituminous coals are not affected by oxidation than are the higher rank bituminous coals (9). The degree of oxidation has been shown to be very dependent upon temperature and the time of exposure and these two factors most influence froth flotation. Anthracite and lignite coal are little affected by surface oxidation.

Resistivity of petrographic components to oxidation increases in order of vitrain, clarain, durain and fusain. Oxidized vitrinite forms a large volumes of fine-grained slurry (1).

Of more value are reagents suitable for promoting the flotation of oxidized coal. It can be shown that there is a similarity in the surface characteristics of coal and oxidize minerals, therefore like oxidized minerals, oxidized coal flotation can be enhanced by the use of surfactants (31).

For example, for oxidized coal a cationic collector such as amine can be used. Alternatively, flotation may be restored by dissolving the oxidized layer in 1% caustic soda or by reduction with benzoyl alcohol. Scrubbing might be a more practical alternative if amine are not used (9,12). Kerosene-pine oil systems are not so strongly influenced by surface oxidation until the extent of oxidation is very great.

5.4.1.6 Zeta Potential And pH

All the coal samples and their lithotypes have negative zeta potentials in distilled water (24). It is appropriate to discuss the surface negativity of coal in respect to its nature. Coal is a heterogeneous substance and its surface is anisotropic. the anisotropic surface consists of two parts, one which is formed by the rupture of Van der Waals bonds and is hydrophobic and the other which is formed by the rupture of ionic or covalent bonds and is hydrophilic.

The variation of zeta-potential of a Turkish bituminous coal seam (Cay) and its lithotypes with pH is shown in fig 5.21 Fig 5.21 shows that when the hydroxyl ions are added into the pulp, the coal surface becomes more negative due to the adsorption of hydroxyl ions. Fig 5. 21 also shows that IEP's of vitrain (isoelectric point), durain and fusain are 4.10, 4.50 and 3.60. The IEP's of their parent coal occurs at pH 4.05. Generally speaking IEP's of all coals occurs at lower pH as the degree of weathering increases.

The effect of temperature on electrokinetic potential was examined by Kovachev. He found that a rise in temperature increased the electrokinetic potential of the coal surface. Temperature affected the stability of the hydrated layers, solvating the mineral grains by changing their surface charge as well as their structure and mobility.

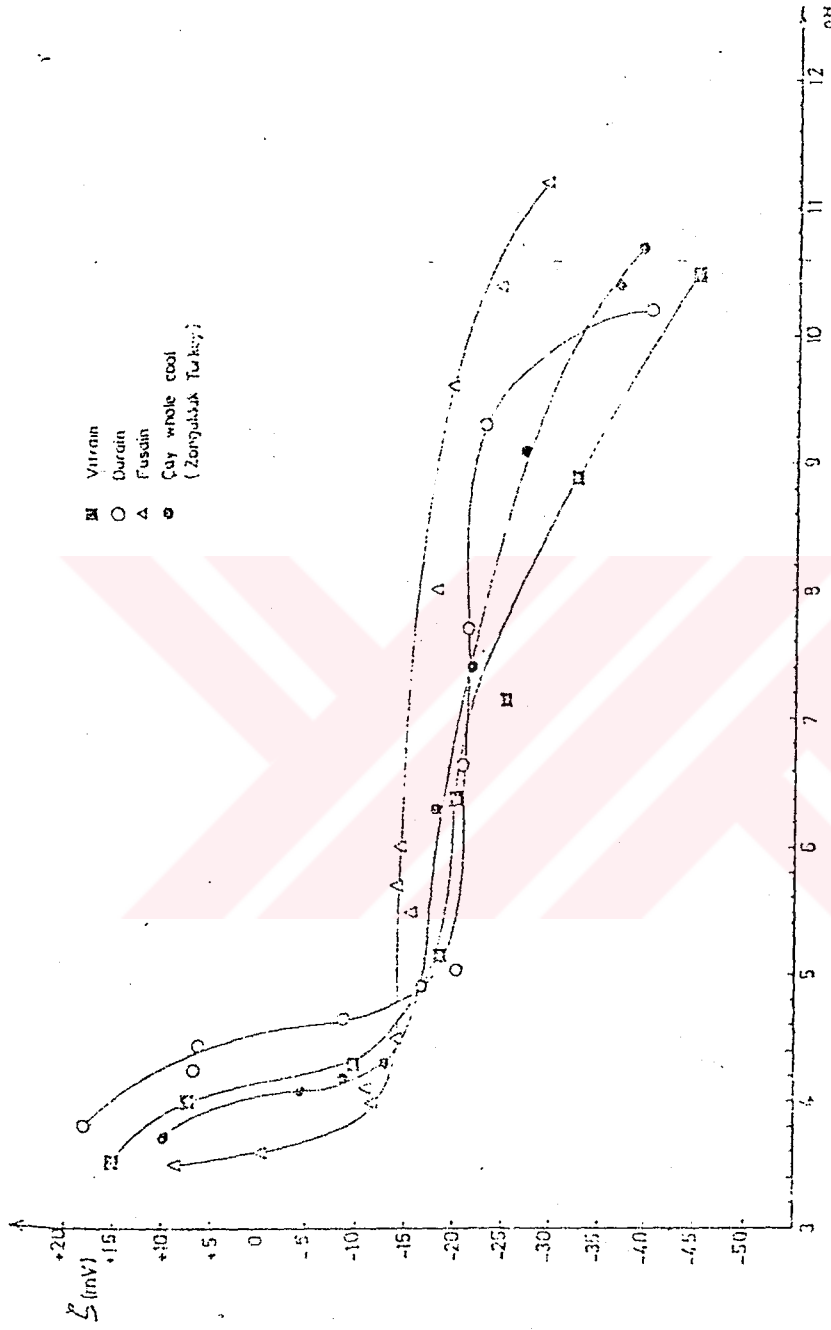


Fig 5.21 Variation of electrokinetic potential of lithotypes with pH (24)

The effect of concentration of various alcohols such as butanol, MIBC, amyl alcohol and heptanol on the electrokinetic potentials of bituminous coal was studied. It was founded that heptanol with seven carbon chain length lowers the electrokinetic potential continuously with the increase of concentration. As the change in magnitude of the zeta potential was not accompanied by the change of its sign, the attachment of alcohols concerns only the diffused portion of the electrical double layer.

Although coal can float well over a wide range of pH, the optimum pH is usually near pH 7 (24,31,9,16). Sun found the optimum pH value for unoxidized bituminous coals in the presence of oily collector to be 7.5 fig 5.22 shows the effect of pH on the floatability of coal (9,12,24).

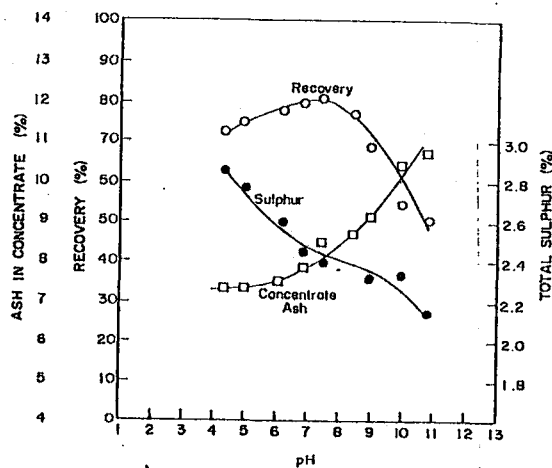


Fig 5.22 Effect of pH on the floatability of coal (9,24).

As seen, recovery drops off at lower or higher values. Low pH value can be controlled by additions of lime. High pH helps to inhibit the floating of pyrite.

A generalized diagram showing the zeta potential versus pH curves for several different coals is depicted in fig 5.23 (17).

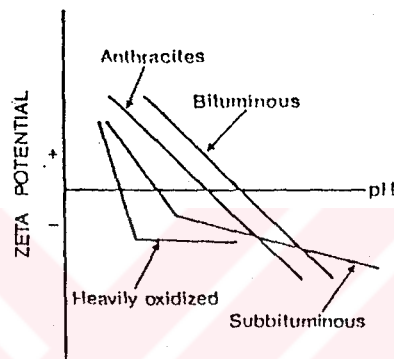


Fig 5.23 zeta potential vs pH curves for various coals (17).

It is seen that, lower rank as well as oxidized coal samples are characterized by negative zeta potential values over the broad pH range from slightly acidic to alkaline.

5.4.2 Affects Of Operational Components

5.4.2.1 Particle Size

The size of coal particles to be floated by froth flotation is very important due not only the mechanics of the process but also the economics.

It is generally excepted that the upper particle size limit for recovery by froth flotation is 28 mesh (0.5 mm) (12,20,24,31,34). For the flotation of larger sizes, several bubbles are attached to a floated particle and that the froth doesn't have enough suspending power to elevate these coarsest particles to the top of the froth column. The flotation of coarse particles will be a function of the rank of the coal, amount and kind of frother/collector, pulp density, the hydrodynamic condition in the flotation cell, and the time of flotation. In general coarse (14x28 mesh) coal flotation is favored by relatively large amounts of oily collector, a higher percentage of solids, higher aeration rate and long flotation times.

Losses of coal occur mainly either in the larger particle size range or in the finer fraction. An attempt has been made to understand the effect of fines content in the feed on the overall flotation performance and on the behavior of individual size fraction in the feed during flotation. From fig 5.24, it is noted that when the fines (-0.075 mm) content in the feed is kept constant, the recovery of non-ash material increases with increase in flotation period. At any particle size, the recovery decreases with increase in fines content of the feed.

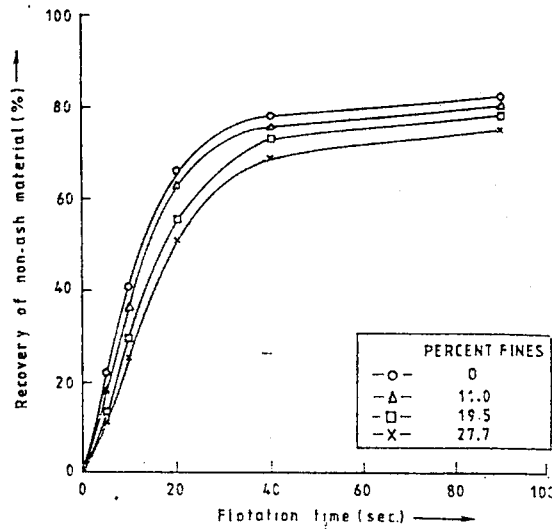


Fig 5.24 Variation of recovery of non- ash material with flotation time (12).

In order to maintain high quality of coal concentrate, it must be determined an optimum size range. Fig 5.25 shows the optimum sizes for froth flotation.

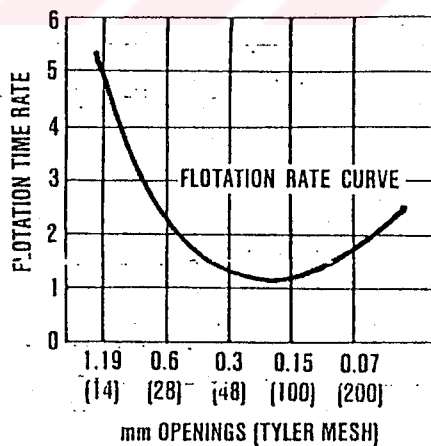


Fig 5.25 Floatability as a function of particle size (9).

5.4.2.2 Slimes

With increased mechanization in coal mining, raw coal contains an increased amount of the ash-forming minerals, such as quartz, calcite, siderite, illite, montmorillonite, kaolinite, accidental incorporation of material from the floor, roof and partings. The problem will be especially serious in those cases where the clay minerals and other fine refuse particles accompany the fine coal sent to the flotation circuit. Slime material complicates the process flowsheet and increases the volume of flotation feed. The consumption of reagents also exceeds the froth formation and disturbs plant operation. In addition, clay minerals cause depression by adsorption at the coal surface acting as hydrophilic colloids and in some instances may completely suppress flotation. Particles of the less floatable higher ash coal components will be more difficult to float in the presence of slimes, if not impossible. The higher the content of slimes in flotation feed, the lower the rate and selectivity of that reasons slime materials are objectionable in flotation of coal.

5.4.2.2.1 The effect of amount and kind of clay

Arnold and Aplan (1985) did some experiments on Pittsburgh coal seam containing 8.62% ash to determine the effect of slimes on coal flotation (19). For this purpose, kaolinite, illite and bentonite were added to coal in varying

amounts. MIBC was used as a frother, fuel oil as a collector. results are shown in fig 5.26 that with increasing kaolinite and illite, the overall yield continually decreases. The clay in the feed increases.

Bentonite acts much differently than kaolinite or illite, serious depression occurring above a 2% bentonite addition with the yield dropping below the theoretical value. For 20% bentonite, almost complete depression of the coal occurs.

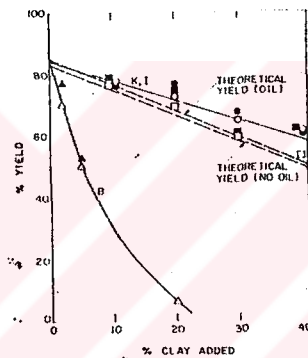


Fig 5.26- The effect of kind and amount of clay on yield in Pittsburgh coal flotation. Circles=k=kaolinite Squares=Illite triangles=B=Bentonite. Open symbols=0.1 kg /t MIBC; closed symbols=0.05 kg/ton MIBC and 0.5 kg/t Fuel oil.(3)

5.4.2.2.2 Water Carry Over (The Null Hypothesis):

Fine clay particles can enter the froth either as

- 1- A carry-over with the water accompanying the froth product
- 2- A specifically collected entity
- 3- Slime coatings on the coal
- 4- Particles entrapped in the froth.

A "null hypothesis" is that clay is carried into froth in proportion to the amount of water transferred to the froth.

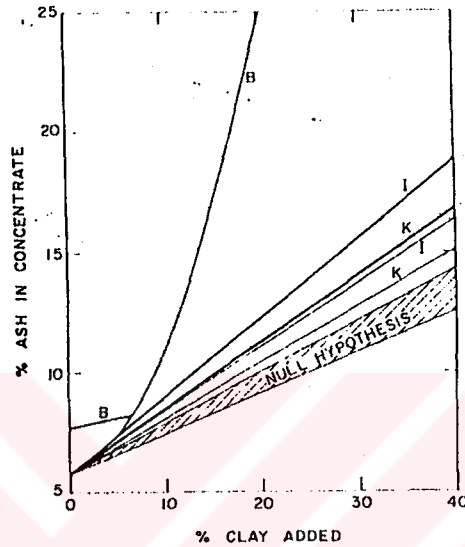


Fig 5.27 - The effect of kind and amount of clay on % ash in the froth in Pittsburgh coal flotation K=Kaolinite, I=Illite, B= Bentonite. Thin lines=0.1 kg/t MIBC, heavy lines= 0.05 kg/t MIBC and 0.5 kg/t fuel oil (3)

Fig 5.27 shows the null hypothesis region in terms of ash in the froth product. While most of the ash reporting to the froth can be attributed to froth carry over when little clay is present. For higher clay concentrations other factors must be responsible for the clay in the froth.

5.4.2.2.3 Removing Slimes From Flotation Circuit

It may be possible to remove slimes by mechanical or chemical methods. Mechanical methods such as attrition and desliming can only produce partial removal of slimes which would suggest that the mode of adsorption of slimes at the coal surface is due to chemical forces. This makes the study of the electrokinetic properties of these clay minerals important. It is possible that the positively charged edges of the clay minerals may attract to the negatively charged coal surface and therefore adsorption takes place.

Perhaps; more importantly the various ions in the water used in the plant, i.e flocculants also influence the degree of slime coating and the ease of removal from coal surface. Use of phosphate electrolyte and starch has improved the flotation of coal with slime problems on a commercial scale but addition of these reagents can produce insurmountable problems in handling the tailings. For example, polyvalent phosphate ions create the negative charge density at the mineral surface resulting in their depression. In such cases intensive investigations of the surface chemistry and settling behavior of particles are required prior to selection of an appropriate polymer to flocculate the tailings in the thickener (31).

In addition to these methods, dilution of the pulp and recleaning of the coal froths would be suggested to obtain cleaner concentrates and removing slimes.

5.4.2.3 Conditioning Time

The conditioning time of coal slurry with reagent would play an important role in achieving optimum flotation recovery and the grade of the froth concentrate. The conditioning time in coal flotation depends especially on the types of reagents used and the rank of coal. A short mixing is needed when an alcoholic frother is the only reagent used, long conditioning periods perform no useful purpose. However, when an oily collector is used, the conditioning period is essential (9). Fig 5.28 show a sharp rise in recovery value between the conditioning time of 2 and 3 minutes (31).

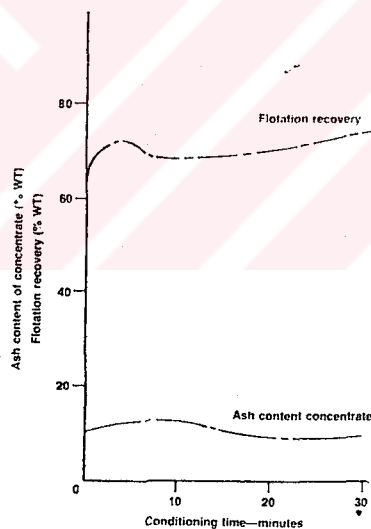


Fig 5.28- Graph showing conditioning time/recovery (reagent used 1kg/t collector, 0.1 kg/t frother)

Longer conditioning time is generally required for the lower rank and oxidized coals (12).

Plants are in operation where this conditioning takes place in the first flotation cell, but it is generally more satisfactory to have a separate conditioner tank.

5.4.2.4 Flotation Time

It has been shown by many researches that recovery of coal over time from a batch flotation cell flows approximately a first-order disappearance law. Stated on a cumulative recovery, r , versus time, t , basis the classical first-order model is given by :

$$r = R (1 - \exp(-Kt)) \dots\dots\dots \text{Eqn (7)}.$$

Where the R is the equilibrium recovery at long flotation times and K is the rate constant (min^{-1}). The parameters R and K are numerically determined with appropriate statistical regions.

Fig 5.29 shows the effect of flotation time on cumulative mass recovery and ash content of the concentrate. It is seen that concentrate ash is independent of flotation time.

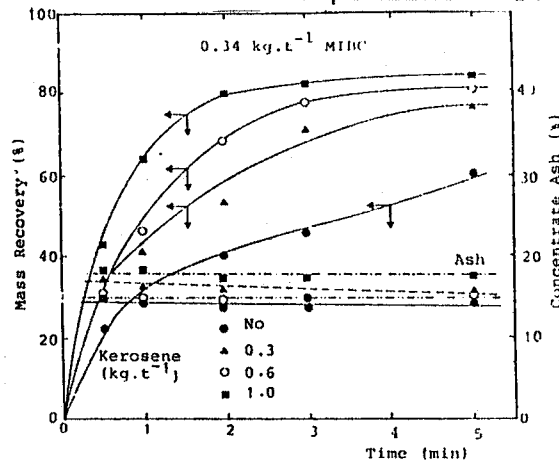


Fig 5.29- Effect of time on flotation of ultrafine coal (11).

5.4.2.5 Pulp Density

Pulp density or the percent solids in the coal-water slurry, affects froth flotation results and must be regulated to counteract particle size. Pulp density varies in practice from 6 to 25 percent, depending on the rank, type, ash content and size distribution of the coal feed.

Crafword reported in 1938 that the best conditions for coal flotation was at 12 % solids by weight. The optimum is near 10% solids by volume (9,22). Zimmerman has found a feed density of 10 to 14 percent as being desirable, raising it to 14-18 percent with an increase in coarse sizes and lowering it to perhaps 5 percent in treating an all slime feed (12,24)

Everson showed that the optimum percent solids is somewhat dependent upon the concentration of frother used, higher solids content requiring higher frother levels (9).

It is used 3.5 to 4.5 solid for ultrafine coals (27).

5.4.2.6 Temperature

Generally, a change in temperature affects the reagent viscosity and interfacial tensions as well as their collecting and frothing properties. In coal flotation recoveries are markedly affected by temperature variations when certain reagents particularly simple aliphatic alcohols

are used. for other reagents particularly oily types flotation recoveries are independent of variations in pulp temperature (24).

Burden found that pulp temperatures has little influence on the rate of coal flotation over the range of 3 to 50 C (9).

5.4.3 Affects Of Equipment Components

5.4.3.1 Aeration and Agitation

Aeration and agitation are essential for flotation. The relation-ship between the flotation rate, k , and the air rate, Q , in cfs is

$$k(\text{sec}^{-1}) = K \times Q^{0.79} \dots\dots\dots \text{Eqn (8)}$$

Thus, from equation (7) the rate of flotation increases with increasing air rate within practical limits. However, there is some air rate beyond which the flotation cell cannot effectively diffuse the air and the relationships breakdown. While coal recovery increases with increased aeration rates, the effective utilization of the air decreases (14).

Agitation has a much smaller effect on coal flotation (31). Too low an impeller speed gives insufficient mixing, insufficient oily collector dispersion and poor bubble-particle contact. Excessively high speeds can decrease effective air dispersion and destroy fruitfull bubble-

particle contacts. Coal yield may be correlated by the air flow number :

$$N_Q = Q / ND^3 \quad \text{..... Eqn (9)}$$

Where Q is the air flow rate in cfm, N is the impeller speed in revolutions per minute and D is the impeller diameter in feed. the optimum occurs near $NQ = 3 \times 10^{-2}$ value.

5.5 MECHANICAL FLOTATION MACHINES

All flotation systems require the generation of a swarm of gas bubbles to levitate the solid particles. These gas bubbles can be formed by three distinctive methods (35).

- 1- By mechanical (dispersed flotation)
- 2- By nucleation of gas from solution (vacuum, dissolved air and microflotation)
- 3- By electrolysis the aqueous phase (electro flotation)

All three methods have been applied to coal separation but dispersed flotation is the most widely used.

While the mechanical flotation designs differ in detail between manufacturers, the internal hydrodynamic characteristics fall one of the two general flow categories shown in fig 5.30

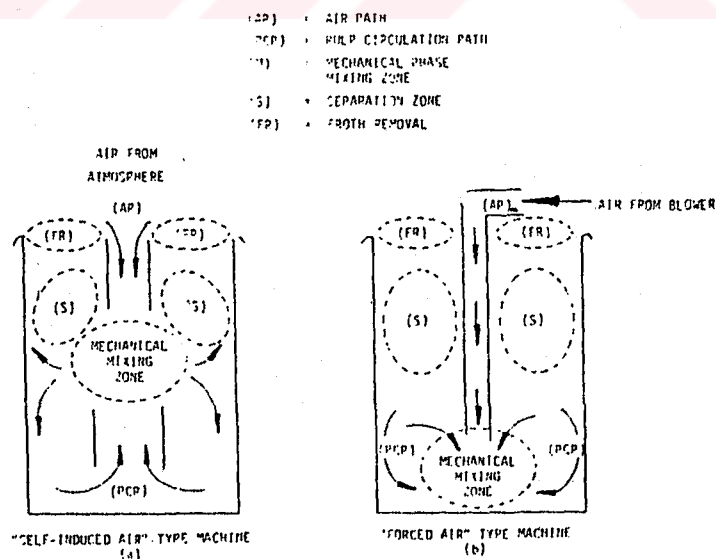


Fig 5.30 Mechanical flotation machine hydrodynamics (35)

Fig 5.30 indicates the air transfer and pulp circulation paths, as well as specific separation or beneficiation zone and the froth removal region in the cell (35).

Dispersed machines can be subdivided into four classes

- 1- Mechanical (cell to cell and aeration)
- 2- Pneumatic
- 3- Froth separators
- 4- Column tank

5.5.1 Mechanical Machines

The original mechanical coal flotation machines were applied to freely floating coals and tended to be of simple, open-through design. Generally, cell volumes of 2-3 m³ were the most popular in an open-through arrangement of five to seven cells.

The cell to cell arrangement became employed when coal of a more difficult nature was encountered, and it was realized that better control in selectivity and higher potential recovery can be obtained as a trade-off for slightly higher operational costs. As a result cell to cell configuration has now become widely employed.

The Denver sub-it coal flotation machine serving as a good example of a cell to cell design. Fig 4.31 comprises; an adjustable weir with a feed pipe , which transports slurry from the weir to pump impeller, a self aerating impeller and large capacity froth paddles, to accommodate the large concentrate yields characteristics of coal flotation.

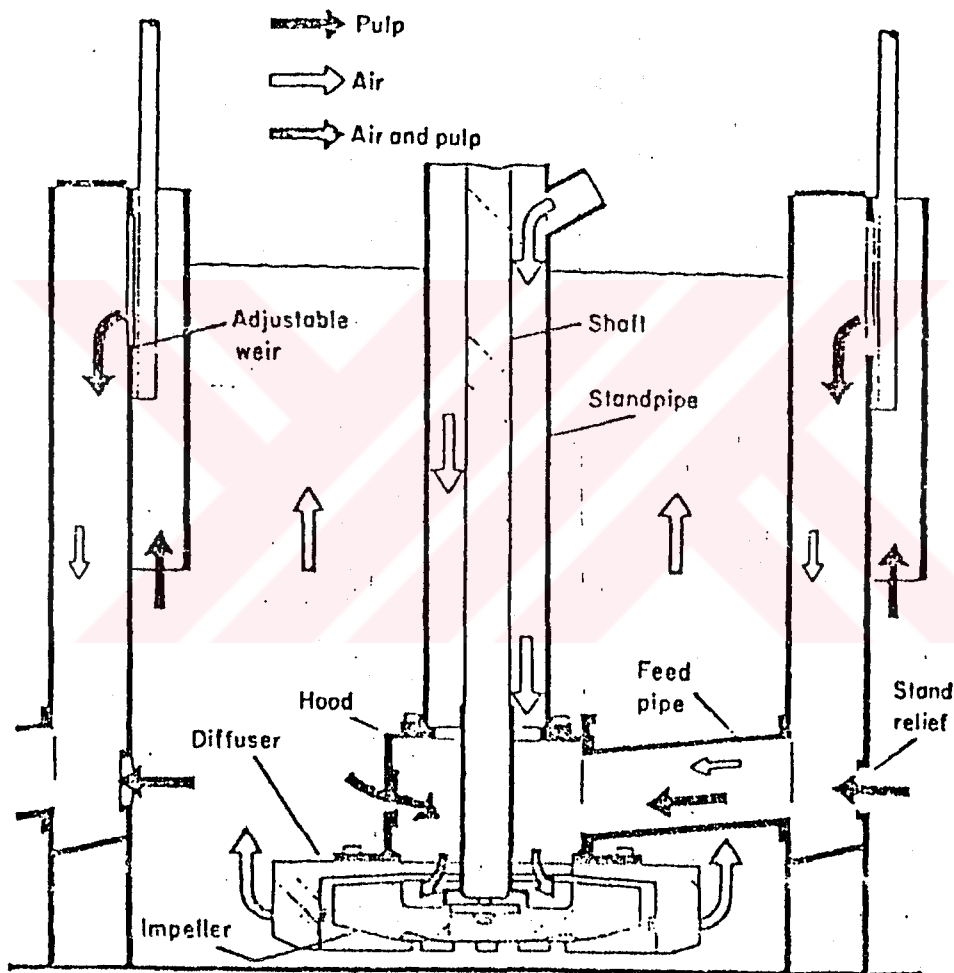


Fig 5.31 A cross-section of a Denver Sub-A, typicall cell to cell machine. (35)

Aeration machines that have air pumped to them by a blower; sub-aeration machines use the impeller design to induce larger quantities of air.

4.5.2 Pneumatic machines

Very few pneumatic machines are currently used for coal flotation. One recent development from Germany, the Schaunberg machine, has been effectively used for coal flotation.

Feed-solids concentration of 30-50 g/lt is a common range for coal, but this can be exceeded to 60 g/lt.

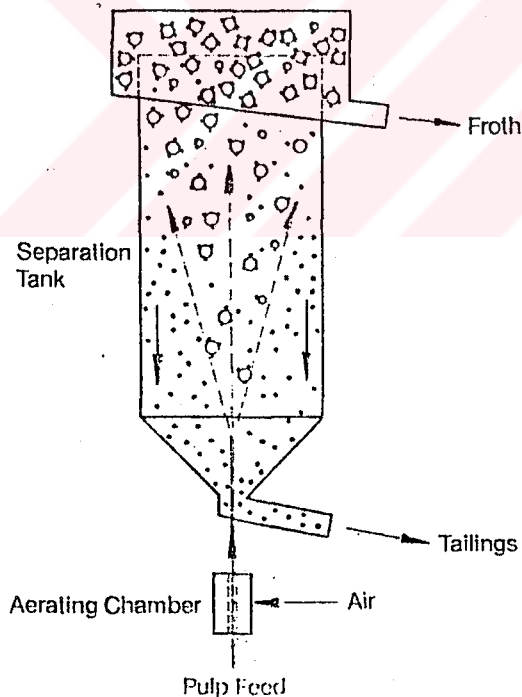


Fig 5.32 Pneumatic flotation principle

4.5.2.1 The Jet-Cyclo Cell

A different type of machine called Cyclo-cell has been used in the U.S.A. This is a new-type flotation machine that incorporates jetting and a centrifugal force field as its working principle to pulp aeration and bubble mineralisation.(10)

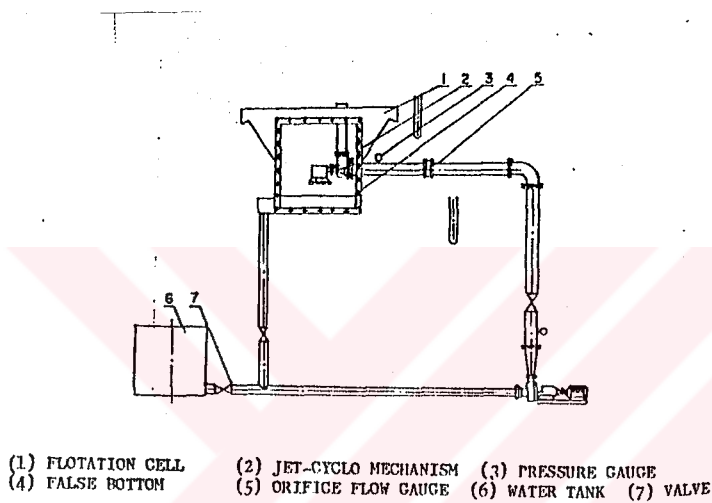


Fig 5.33 The jet cyclo flotation cell.(36)

The experimental results showed that :

- 1- The coarser coal grains are better separated in the jet cyclo flotation cell than in the other machines.
- 2- Since the of the jet-cyclo flotation cell produces an excellent emulsifying effect on the flotation reagents, the amount of collector required with similar dosage of frother, may be 20-30 % lower than other machines.
- 3- The jet-cyclo flotation cell has the further advantages of being simple in design, light in the weight, and easy to manufacture and repair.

5.5.3 Froth Separators

Froth separators are still of major interest in ore treatment, but some coal use has been reported in the U.S.S.R. Potential for application lies in the treatment of very coarse coal slurries (from 1 mm to 0.050 mm), possibly those containing finely divided clays. The benefits in terms of coal treatment are that larger particles can be captured and high treatment rates are possible.

5.4.4 Column Flotation

Coal flotation columns tend to rely on the formation of a froth bed on the surface of the column and a washing phenomenon caused by interaction of the rising bubble mass with newly introduced slurry insurges through the pulp.

Leeds column was invented and designed by Dell to provide effective counter-current froth cleaning (14,22). The Leeds flotation column consists of a rougher/scavenger bank of flotation cells, upon which cleaner and recleaner stages are mounted directly on top. Froth from the rougher cells at the diffuses through barriers, which are so designed that a continuous thin of froth is maintained beneath each barrier.

Clean water is passed down from the top stage counter-current to upwardly rising froth (14).

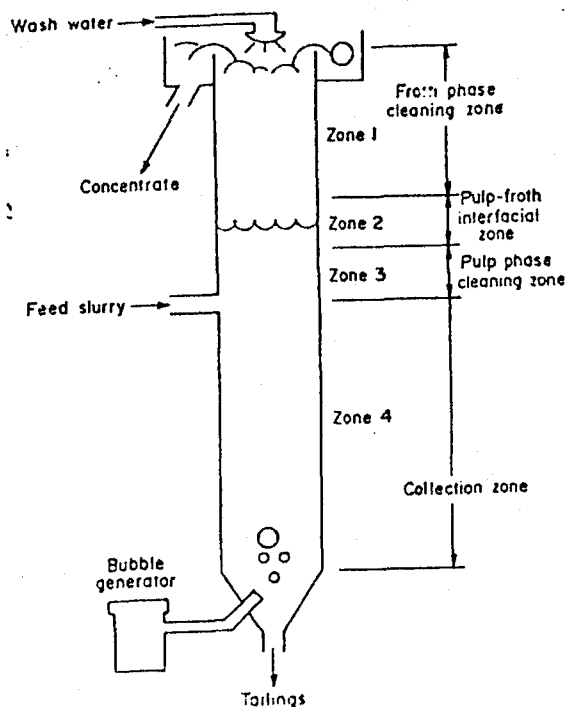


Fig 5.34 Flotation column diagram showing beneficiation zones (14).

5.6 FLOTATION CIRCUITS

Most froth flotation circuits in American coal preparation plants are relatively simple. They produce primary or rough concentrate and primary tailings with no cleaning of either product. This simple straight line flotation circuit is illustrated in fig 5.35 (29).

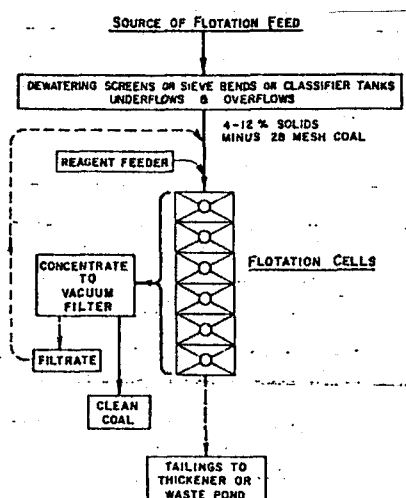


Fig 5.35 Simple coal froth flotation circuit (35).

Some circuits, of course, are more complicated, and may utilize special circuits for slimes and coarse sizes. One approach, becoming increasingly common in the treatment of metallurgical coking coal, is the use of two cell-banks for treating a preclassified feed, i.e. coarse and fine coal feeds (see Fig 5.36).

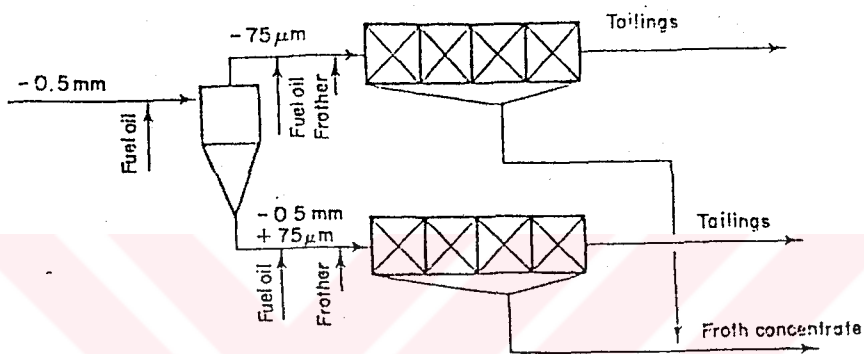


Fig 5.36 Twin single-stage flotation treatment of a coarse and fine coal, following preclassification using a hydrocyclone.

Other flotation circuits employ hydrocyclones which serve to provide a thickened feed to the flotation circuit and also effective desliming of clay-mineral content of the feed, as shown in fig 5.37

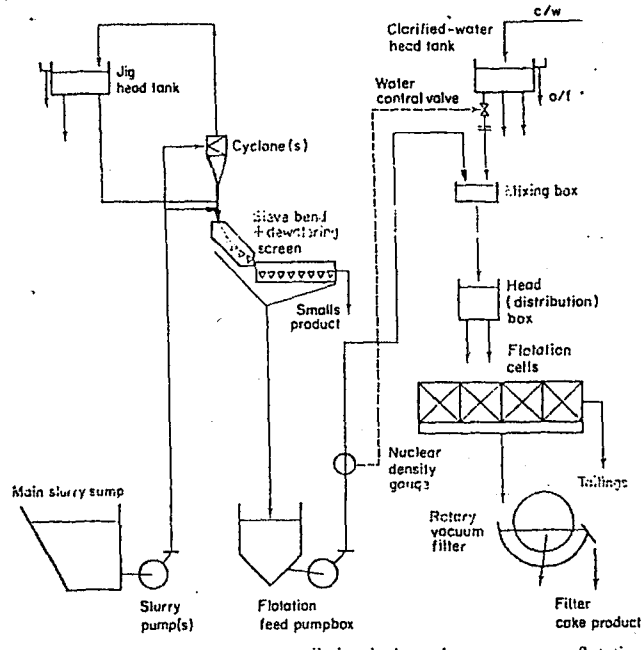


Fig 5.37 Flotation circuit employing desliming hydrocyclone to prepare flotation cell feed (35).

In some cases, there may be benefit in using a two-stage circuit employing a second cell bank for recleaning the concentrate obtained from the first stage. Fig 5.38 shows a typical two-stage flotation circuit and fig 5.39 shows an alternative employing a desliming cyclone for preclassifying in the secondary stage.

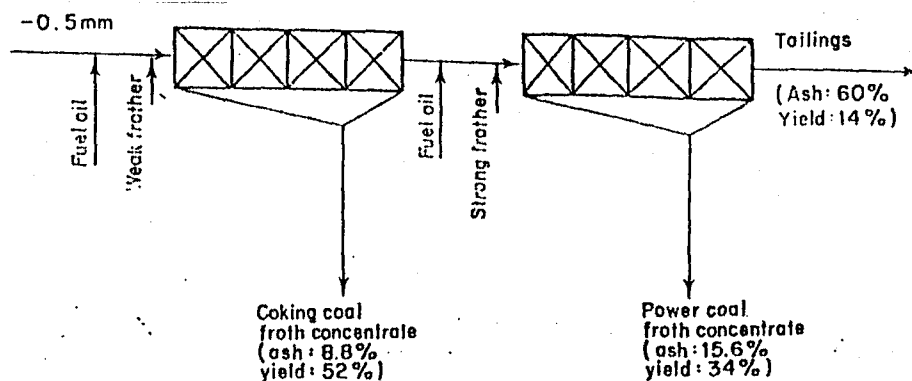


Fig 5.38 Two-stage flotation circuit

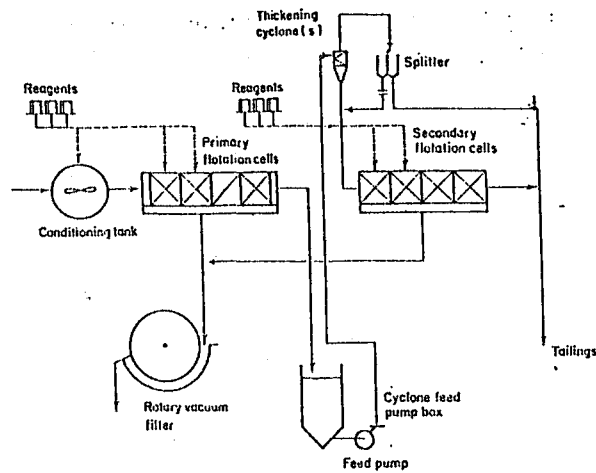


Fig 5.39 Two-stage flotation circuit with preclassification of primary tailings (35).

Three-product separation involving a two-stage flotation circuit has also been operated effectively in Australia. The primary cell bank was used to obtain a low-ash content cooking coal by using "starvation" reagent dosages. A thermal coal product of 15-16% ash content was then obtained by reconditioning the primary tailings with additional reagents and floating off a second product from the secondary cell-bank.

6. THE ENRICHMENT OF ZONGULDAK FINE COAL BY FLOTATION

Today, the mechanical production methods which are applied in coal mining cause to the increase of fine coal fraction. This fraction which can not be enriched with gravity methods can only be utilized by flotation. In Turkey, 14-18 % of the bituminous coal which in the size of -0.5 mm is concentrated by flotation.

This chapter includes explanations about Zonguldak hard coal region, flotation experiments applied to flotation feeding of Zonguldak Central Washery, findings and results obtained from these experiments.

6.1 Regional Description

The Zonguldak coal region is about 130 km long and 30-50 km wide and lies the southwest coast of the Black Sea. The basin is the the main bituminous coal producing region of Turkiye. (fig. 6.1)

6.1.1 Geology

6.1.1.1 Regional Stratigraphy

The Carboniferous northwest of Anotolia has a total thickness of 2000 meter and contains almost every stage found in the standart European sequence. The lower Carboniferous consists of marine sediments, wehereas the upper carboniferous is dominantly terrestrial. Fig 6.2 shows the generalized stratigraphic section for the carboniferous in the Zonguldak region.

The carboniferous of the Zonguldak region falls into natural divisions. SeriesA, the Alacaagzi is within the Namurian stage. the Alacaagzi series contanining 15 coal seams, the average thickness of a good coal seam is 75 cm. The unit mainly consists of lacustine sandstones and shales. Series B is Westphalian A in age and contains Kilic (300-600 m) and Kozlu (710-730m) formations. The sediments are largely lacustine clastics. This series is best represented at Zonguldak, Kilimli, Catalagzi and between Kozlu and Gelik.

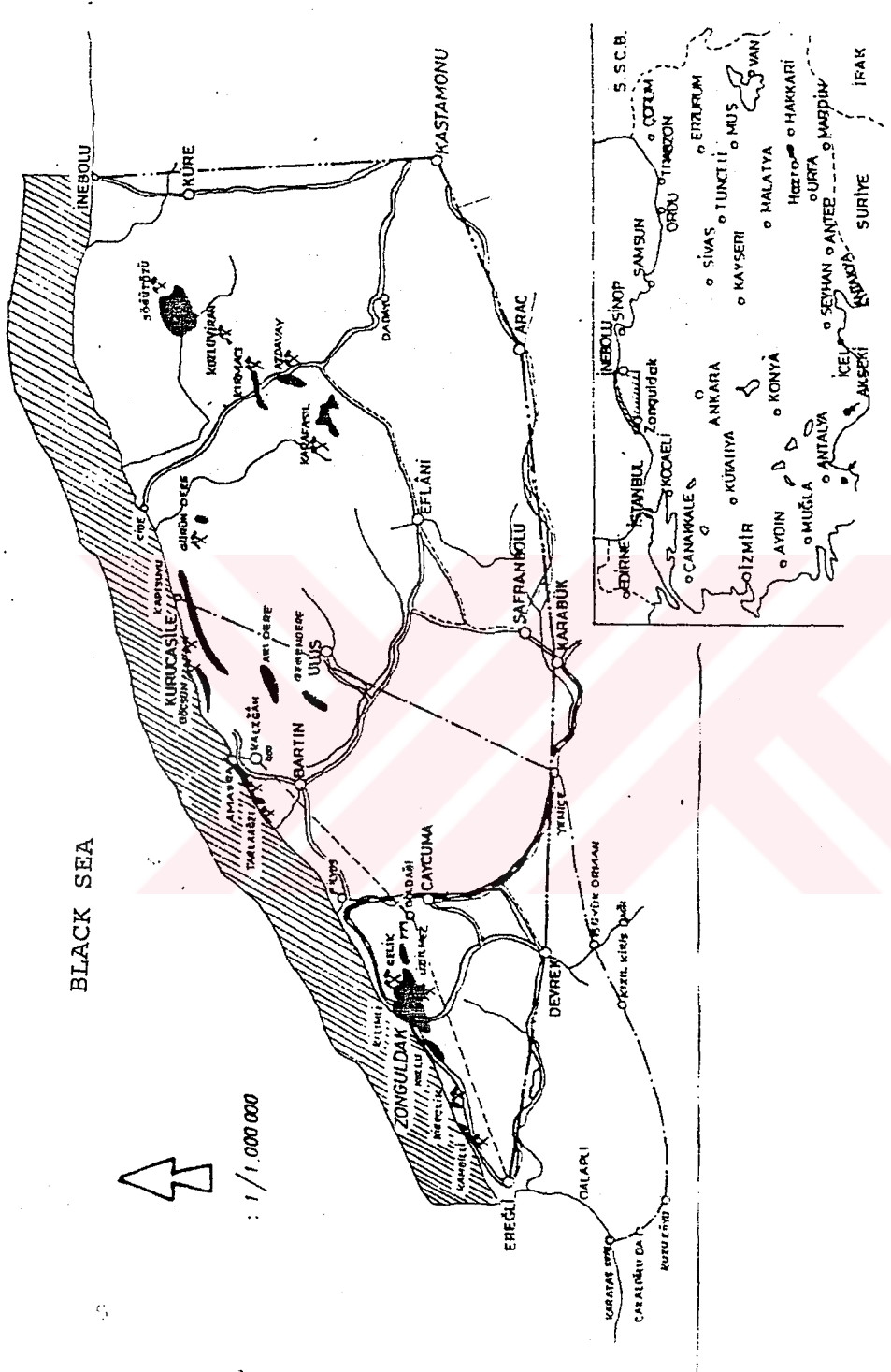


Fig 6.1 Regions of Turkiye bituminous Coal

The Karadon series, to the south of Kozlu and north of Gelik, has a thickness of 350-500 meters and contains eight seams of poor coal.

6.1.2 Structure

The structure of the Zonguldak region is extremely complicated. The region is heavily faulted and folded. Early hercynian movements are seen between the namurian and westphalian sequences.

Two sets of faults, one with an east-west orientation and the second with a northsouth orientation occur, in the area. The east-west set has been related to the hercynian orogen while the north-south set is thought to be Alpine. The faults are predonimantly normal and have been interpreted as growth or epigenetic in nature.

6.1.3 Mining

Estimated bituminous coal reserves in Turkiye are approximately 1.3 billion metric tonnes (Table 6.1)

The bituminous coal is mined by the state owned Turkish Hard coal Authority. About 5 million tonnes of bituminous coal is produced each year.

The main regions of production are Kozlu, Amasra, Kirecli,

Table 6.1 Estimated reserves of Zonguldak coal Region

COAL REGIONS	SECTION	MINEABLE	PROVEN	PROBABLE	POSSIBLE	TOTAL
ARMUTÇUK	Kandıllı	535.000	5.381.422	14.458.324	21.247.000	41.621.745
	Alacağzı	210.000	1.500.000	8.610.776	21.064.000	31.384.775
	Kireclik	—	27.934	2.839.211	16.264.300	19.131.445
TOTAL		745.000	6.909.356	25.908.311	58.575.300	92.137.967
KOZLU	İhsaniye	7.909.337	10.054.266	—	—	—
	İncirharmanı	—	—	21.825.000	240.000.000	—
TOTAL		7.909.337	10.054.266	21.825.000	240.000.000	279.788.603
ÜZÜLMEZ	Asma-Dilaver	11.904.741	42.188.094	73.122.000	31.680.000	158.894.835
	Çaydamar	1.818.510	9.824.325	28.632.000	47.620.000	87.894.835
	TOTAL	13.723.251	52.012.419	101.754.000	79.300.000	246.789.670
KARADON	Gelik	5.224.500	21.051.950	38.268.000	68.783.945	133.328.395
	Karaden	5.270.000	11.165.741	21.922.000	—	38.357.741
	Kilimli	5.204.500	2.018.392	19.537.000	275.787.057	302.546.949
TOTAL		15.699.000	34.236.083	79.727.000	344.571.002	474.233.085
AMASRA	Amasra	950.197	29.960.876	183.527.272	60.665.446	275.103.750
	Azdavay	14.500	1.822.001	1.943.000	1.297.000	5.076.500
TOTAL		964.697	31.782.877	185.470.272	61.962.446	280.180.250
BASIN TOTAL		39041.285	134.995.001	414.684.583	784.408.748	1.373.129.590

Karadon, Uzulmez, Kilimli and Gelik. Longwall and room and pillar methods are used for coal extraction. Due to the complex geological structure, which results in roof instability, roof support is very important and crucial in mining of coal. The mine roof is supported by wooden posts and steel arches. Hydraulic roof supports and cribs are used for especially unstable zones in the mines. Mine access is provided by vertical shafts and material handling by electrical trains, conveyor belts and horse-pulled cars.

Exploratory drilling is commonly used to evaluate the nature and extent of coal reserves in the region. The high accident and fatality rate in the mines are probably results of not observing safe mining practice and safety regulations many workers are killed yearly by methane gas explosions and roof falls.

6.1.4 Petrographic Description

6.1.4.1 Special Characteristics of Turkish Coals

Almost all of the Turkish bituminous coals exhibit poorly developed bonding with only a few thick bands of vitrain. They are mainly composed of a complex mixture of macerals. the coals have a high volatile matter content and in some cases enough mineral matter so that they can be called shaly coal.

The coal seams contain a considerable amount of inertite macerals. Inertites, especially fusinite are present in almost all coals. The distribution of inertites in Turkish bituminous coals is sometimes random and sometimes stratified with other macerals. Rarely some inertites show a rounded granular texture.

Exinites are abundant. Both macros and micropores are readily discernible. Some are preserved within a resinite globe and some may incorporate mineral matter.

6.1.4.2 Minerals

Clays are the most common minerals in the Turkish bituminous coals. Some have a refractive index around 1.608 and probably are illite or monmorillonite. Others have a lower refractive index ($n=1.52$) and are probably kaolinite. Clay minerals are found as cell fillings and cavities. Coal-clay mixtures where the clay is either layered with coal randomly distributed iron sulfides (pyrite and marcasite) are the second common minerals. There are three kinds of pyrite forms in the Turkish bituminous coals. 1- Frondoidal 2- Cell cavity fillings 3- Fracture fillings.

Other relatively common minerals in the coals are calcite, siderite and quartz.

6.1.5 Washeries

In order to prepare appropriate coal suitable to the demands of industry, there have been established Armutçuk, Zonguldak, Çatalağzı and Amasra washeries from the west to east whose total capacities are 2150 t/h.

Zonguldak washery, which clean the coal produced in Kozlu and Uzülmez Establishments, was instituted in 1957. In this washery, in the size of 0-100 mm raw coal is enriched. All the methods which are applied to raw coal in Zonguldak washery are shown in fig. 6.2 as a flowsheet.

The coals of Kozlu and Uzülmez are enriched by dense medium, flotation and filtration units in addition to baum and acco type jigs. Together with baum and acco type jigs, dense medium serves to clean coarse coal. In the dense medium unit together with 0-18 mm raw coal, middling of baum jig is cleaned. In the baun type and acco type jigs, 0-100 mm size ranges of raw coals are cleaned. In the coarse coal cleaning units, 0.5-6 mm, 10-15 mm, 18-50 mm and 50-100 mm size ranges of clean coal are produced as well the middling.

The size of 0-0.5 mm production is utilized in the coal cleaning units like flotation and filtration.

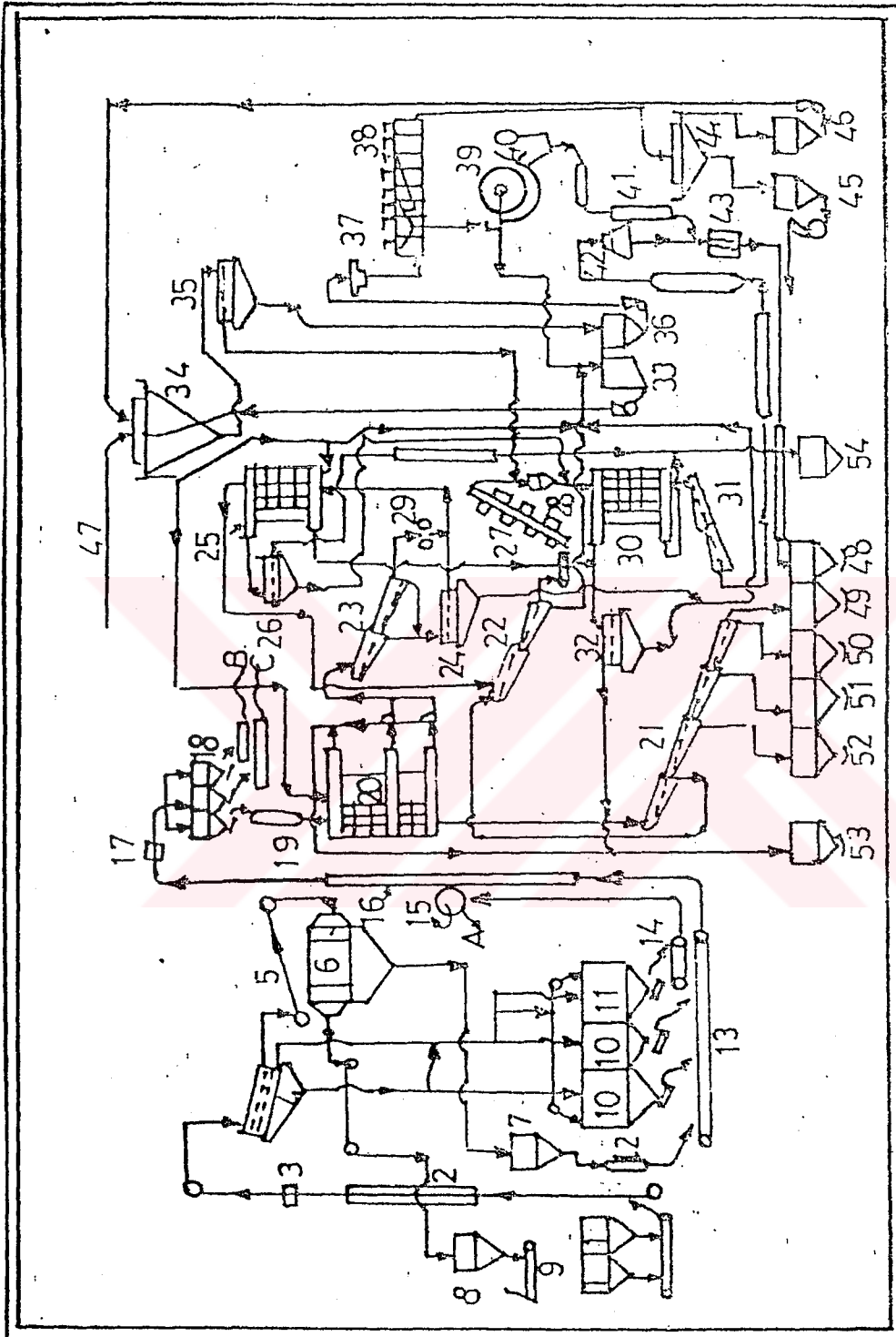


Fig 6.2 Flowsheet of Zonguldak Central Washery

Explanation of flowsheet of Zonguldak Central Washery

- 1- Unloading bunkers (3000 tonnes capacity)
- 2- R.O.M belt (3 total, 1000 tph)
- 3- Scales (3 total)
- 4- R.O.M screens (50 mm, 90 mm and 100 mm)
- 5- R.O.M belt (30")
- 6- Rotary breaker (9'Ø + 14' long)
- 7- Raw coal bunker (100 m³ x 0 mm)
- 8- Rock bunker (150 m³ capacity)
- 9- Relocated picking belt (36")
- 10- Raw coal bunkers (1000 t. capacity)
- 11- Raw coal bunker to feed dense medium unit (18 mmX0 mm)
- 12- Boney belt
- 13- Plant feed belt
- 14- Heavy medium plant belt (250 tph)
- 15- Heavy medium plant
- 16- Plant feed belt
- 17- Scales
- 18- Feeding bunkers (3-total)
- 19- Plant feeding belts (3-total, 250 tph)
- 20- Baum jig (3-total, total capacity is 750 t/h)
- 21- Sizing screens (100-50 mm, 50-18 mm, 18-10 mm, 10-6 mm)
- 22- Screens for 0-6 mm
- 23- Middling screens
- 24- Screens (3-total)
- 25- Acco jig

- 26- Screens (3 total)
- 27- 0.5-6 mm elevator (3 total)
- 28- Feeding
- 29- Middling crusher
- 30- Acco jig (0.5-6 mm)
- 31- 0.5 X 6 mm clean coal screens (3 total)
- 32- Screens (3 total)
- 33- Re-cycling pond
- 35- Clean coal sizing screens (12 total)
- 36- Slime pond
- 37- Flotation cells (9-total)
- 38- Flotation cells
- 39- Filters (Emco type)
- 40,41- Belts (6 total)
- 42- Centrifugal dryers (4 exists, 2 new)
- 43- Middlings mixers
- 44- Static thickener (60")
- 45- Tailing pond
- 46- H.M clean-up pump
- 48-0-10 mm clean coal bin
- 49- 50-100 mm clean coal bin
- 50- 18-50 mm clean coal bin
- 51- 10-18 mm clean coal bin
- 52- 6-10 mm clean coal bin
- 53- Refuse bin

6.2 Experimental Studies

6.2.1 Materials and Method

6.2.1.1 The samples used in experiments

Coal sample of central washery was taken from the part which is fed to flotation circuit. After the sample was dried in room temperature, representative samples were prepared for tests and flotation experiments. In order to prevent oxidation, the samples were kept under boiled water.

The results of the sample's dry analysis and screen analysis are shown in table 6.2, 6.3 and fig 6.3.

Table 6.2 The results of Central Washery sample analysis

ANALYSIS TYPE	DRY IN AIR	DRY
ULTIMATE ANALYSIS		
Moisture (%) :	2.41	--
Ash (%) :	51.60	52.87
Volatile Matter (%) :	15.59	15.97
Fixed Carbon (%) :	30.40	31.16
Total	100.00	100.00
HEATING VALUES		
(Kcal/kg)		
Net Heating Value :	3739	3831
Gros Heating Value :	3874	3969

Table-6.3 Size analysis of the test coal sample

Size (mm)	Weight (%)	Ash (%)	Cumulative upper size	Cumulative under size	Dist.	Average Ash (%)
-1+0.5	9.74	36.19	9.74	100	11.38	38.19
-0.5+0.315	34.72	39.11	44.46	90.26	32.06	38.91
-0.315+0.2	17.93	43.50	62.39	55.54	19.08	40.23
-0.2+0.1	19.58	78.65	81.97	37.61	17.38	49.57
-0.1+0.075	5.85	58.02	87.82	18.03	6.89	49.98
-0.075	12.18	60.02	100	12.18	13.80	51.20
TOTAL	100	51.20			100	

As it can be seen from the table 6.3, 9.74 % of the feeding is in the size of +0.5 mm, the majority of this fraction consists of coal particles. In the -0.2+0.1 mm size fraction, the ash percentage is very high and it increases the probability of bounded particles in this grain size. While +0.2 mm grain size of average ash value is 40.23%, average ash value of the sample is 51.20%.

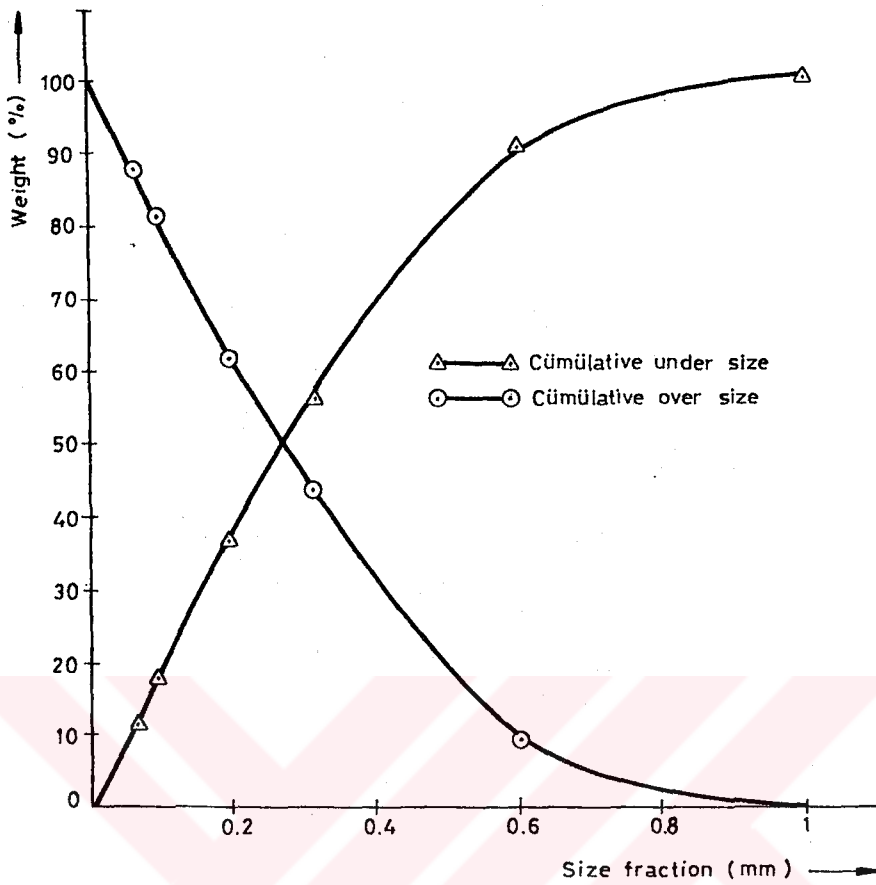


Fig 6.3- Graph showing curves for upper and under size of the sample

6.2.1.2 Reagent Used In Experiments

The reagents used in experiments are collectors, frothers, depressants and non-ionic emulsifier.

In general, of the collectors used in coal flotation, spindle oil, light neutral, bright stock, heavy neutral, kerosene, fuel oil, diesel and white spirit were used and these reagents were obtained from izmir Aliaga refinery.

As frother, strain chain aliphatic alcohols like methanol, athanol, buthanol, amyl alcohol, isooctanol, ether type DF 250

and DF 1012, aromatic type frother phenol and cyclic type pine oil were used.

In the coal flotation, locally produced sodium silicate and tri-poly phosphate as depressants, non-ionic type emulsifier were used.

6.2.1.3 Method

Flotation experiments were carried out in a Denver laboratory type flotation machine. The experiments were done gradually as a rough and single cleaning in one liter flotation cell. The experimental studies, the mentioned situations excluded, are as follows:

pH = nature pH

Impeller speed= 1300 rpm

Solid-liquid ratio= 10% by weight

Conditioning time= 3 min

Flotation time= 3 min

Experiment current scheme is as follows:

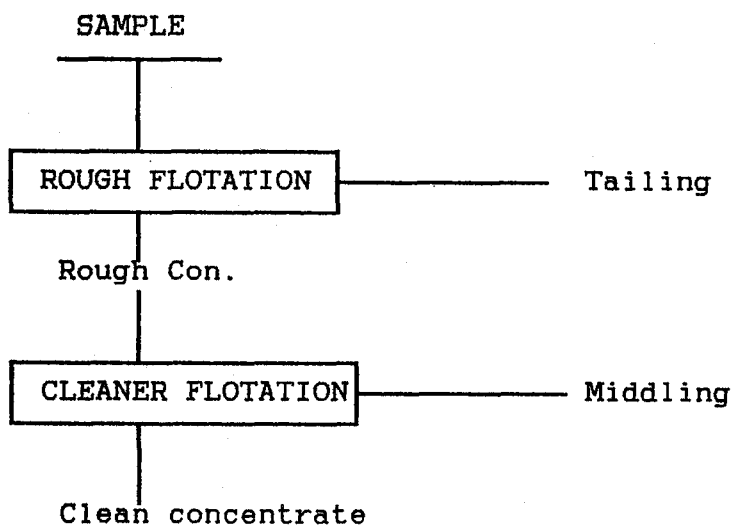


Fig 6.4 Experiment current scheme

6.2.2 Flotation experiments

In order to examine the flotation characteristics of fine coal and to find out the effects of various reagents on the coal flotation and to determine the most proper flotation conditions, various experiments have been done.

The aim of the experiments has been to obtain low-ash-content froth products from high ash levels (approximately 51.20 %). In other words, the aim has been the utilization of the tailings and attribution of an economic value to them.

The flotation experiments have been carried out in two steps. In the first step, the optimum flotation conditions have been determined. In this step, the examined parameters are as follows :

- 1- Optimum particle size
- 2- Collector type
- 3- Frother type
- 4- Conditioning time
- 5- Solid- liquid ratio
- 6- Depressant type
- 7- Effect of emulsifier

In the second step, under the optimum conditions obtained from the first step, it has been examined what effects of the collector amount, frother amount and flotation time on the coal flotation. From the obtained evidences, bound to

Efficiency index, in the desired flotation time, the most proper reagent amounts have been determined by utilizing third-degree regression equation and using computer programme.

6.2.2.1 Optimum size fraction

It is generally accepted that the upper particle size limit for recovery by froth flotation is -0.5 mm.

For examining the effect of size fraction to flotation and to determine the optimum size range, flotation experiments were carried out for the size fractions of -0.5 mm, -0.315 mm, $-0.5+0.075$ mm and original sample. (table A-6.1)

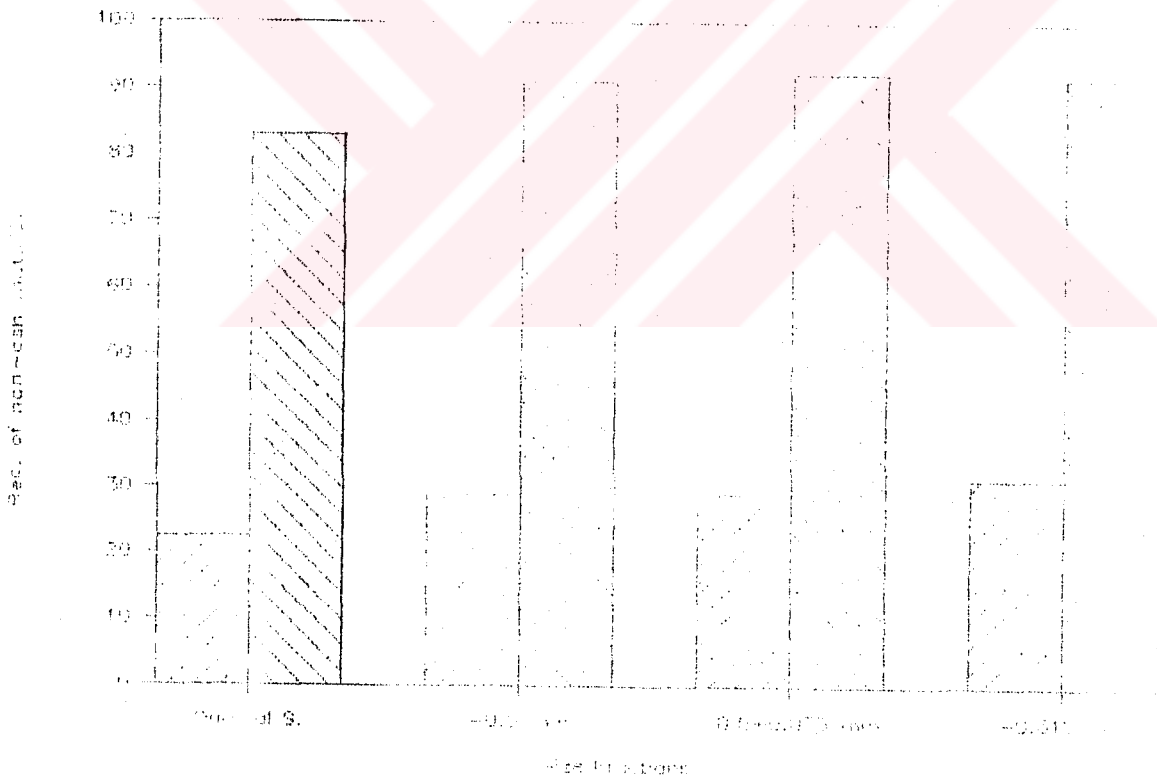


Fig 6.5 Effect of size fraction

(Kerosene= 1.8 kg/t, mixture of alcohol=200 gr/t)

As it seen from fig 6.5, the recovery of non-ash material increases when the particle size is reduced but ash content of the concentrate is about 29%. As a result, grinding is not necessary for this type of coals and original sample has been used for the other steps.

6.2.2.2 Collector Type

To determine the effects of collector on the flotation yield of the high ash content of the coal sample, eight collectors of different viscosity mentioned in part 6.2.1.2 were tested. Experiments were carried out in two different stages using with pine oil and DF 250 as frothers. The results of these tests are given in Table A-6.2,6.3 and plotted in fig 6.6.

The best results in terms of combustible yield and ash content of concentrate were obtained in the usage of diesel, kerosene and light neutral of collector types. Although it has been obtained better values of yield using with higher carbon content of collectors, ash contents of the froth concentrate were too high. Besides the highest of yield was obtained using with spindle oil having a lower viscosity, ash content of concentrate was not appropriate. Taking into consideration due to ash content, the most proper collectors are kerosene, diesel or light neutral. A series of experiments were done to determine the effect of mixture of collectors chosen above (Table A-6.4)

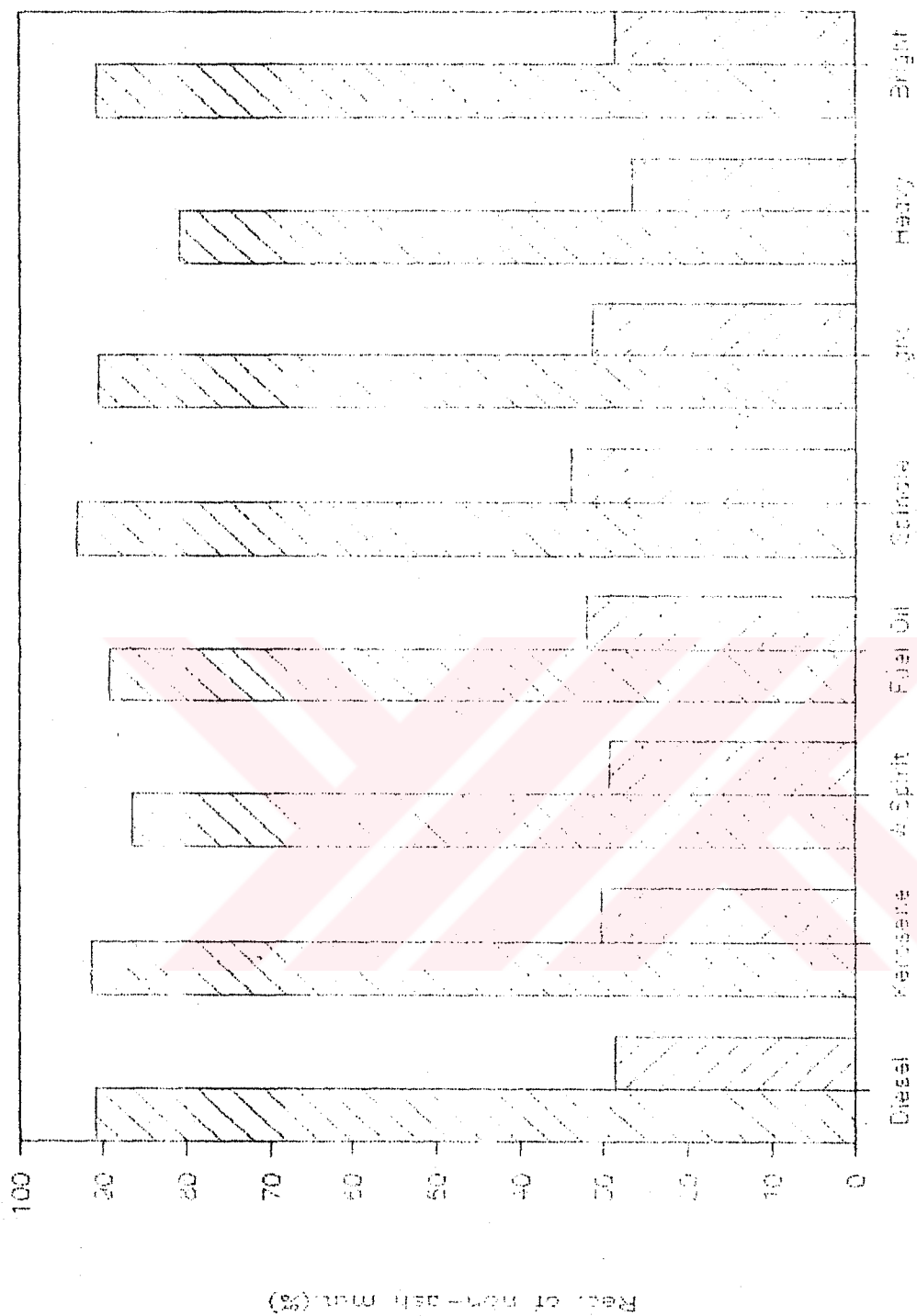


Figure 1. Effect of Collector Type

DF 250

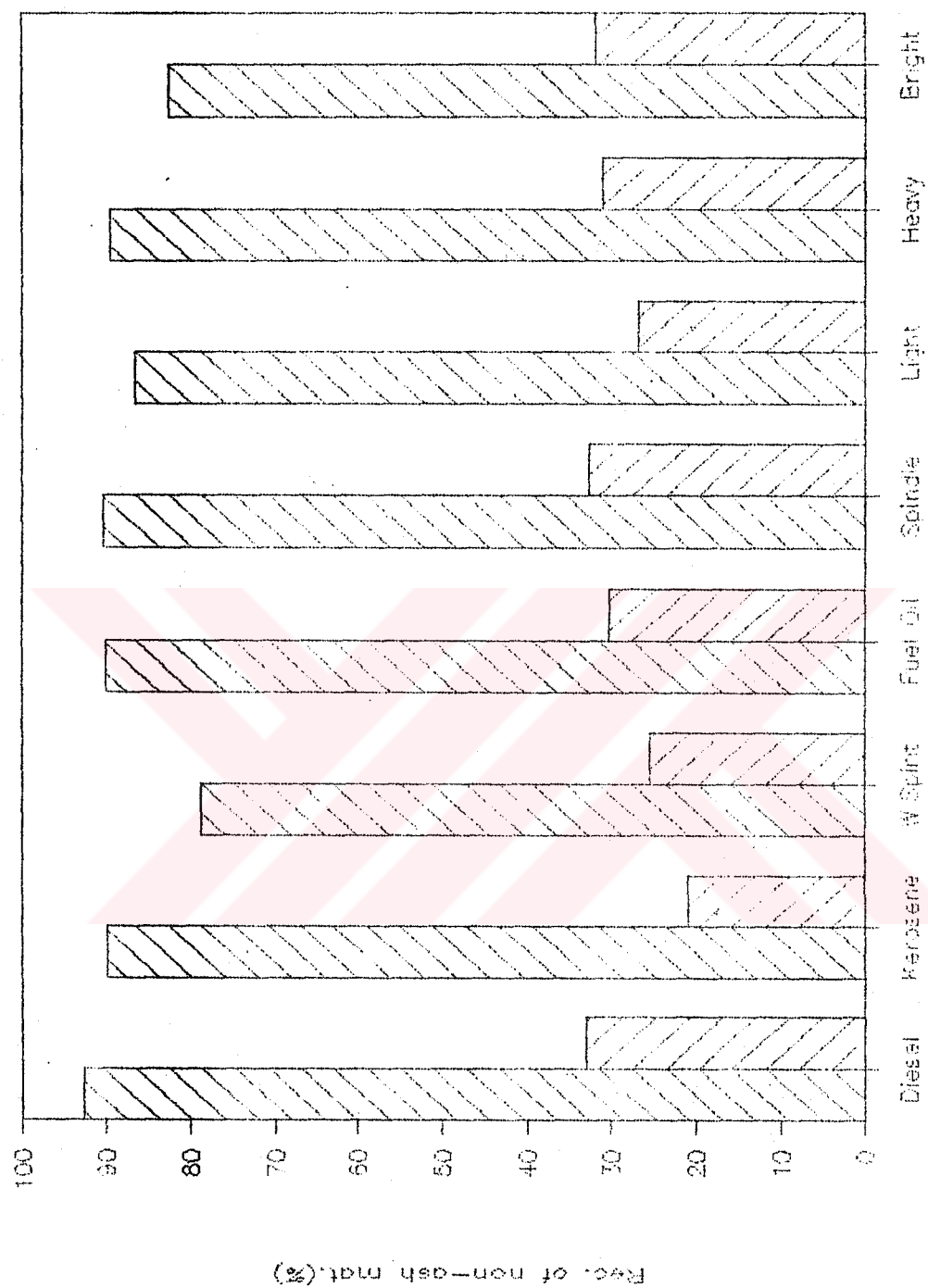


Figure-0 Effect of Collector Type

L:Light , K:Kerosene , D:Diesel

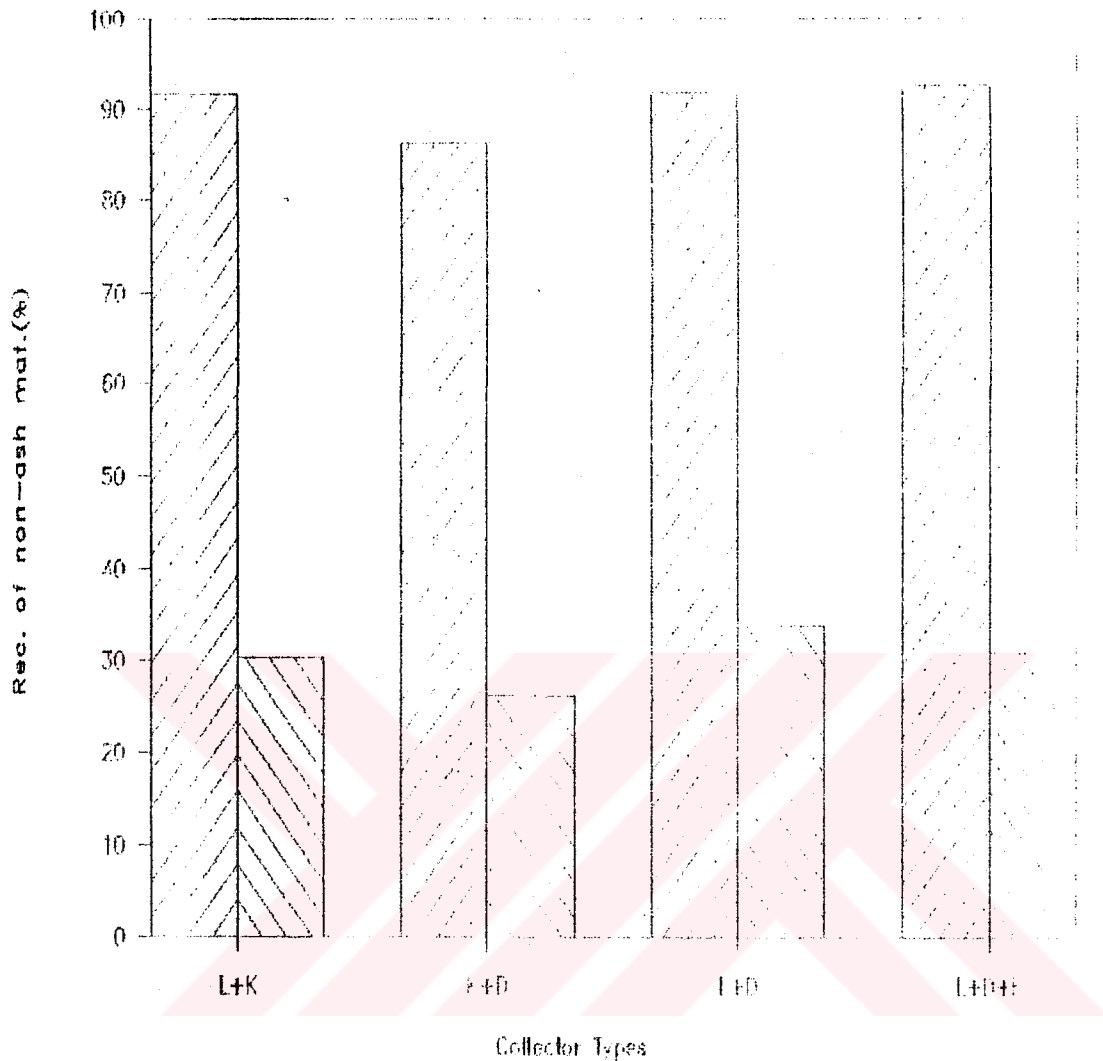


Fig 6.7 Effect of mixture of collectors

(mixture of collector=2.7 kg/t,pine oil=300 gr/t)

As it is seen from fig 6.7, there is not any differences between the use of mixture of collectors and the use of these collectors by them selves. For this reason, kerosene has been used as a collector for the latter experiments.

6.2.2.1.1 Effect of Collector Addition

Results shown in fig 6.8 suggest that while increases in the addition of kerosene improve both the flotation of recovery and ash content of the concentrate. When kerosene addition is 4 kg/t, yield of 95% remains at the same level and ash content is approximately 30%.

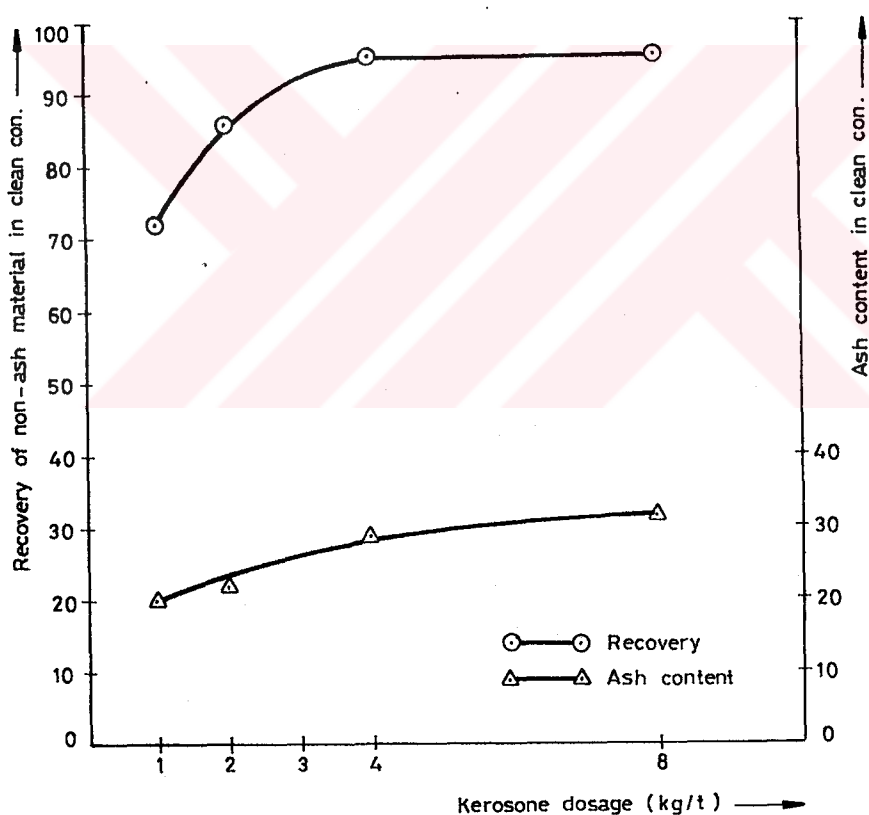


Fig 6.8 Effect of collector dosage(frother= 200 gr/t)

Low yield and ash content of the concentrate were obtained using with only a frother. These values are equal to the results in the use of kerosene addition of 1 kg/t. To maintain the selective action of this collector on coal for the study of the effect of other variables, it was decided that the collector (kerosene) addition was to be maintained at 1.8 kg/t.

6.2.2.2.2 Effect of Flocculant

It was suggested that the use of flocculant could reduce ash content of concentrate. Results of experiments are tabulated in table A-6.6 and plotted in Fig 6.9

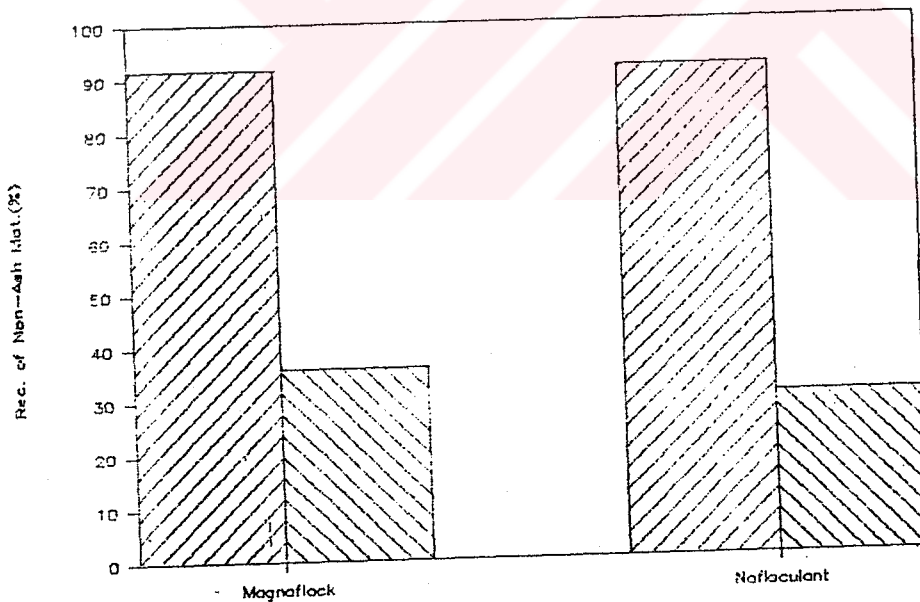


Fig 6.9 Graph showing effect of flocculant

(Kerosene=1.8 kg/t, octanol=200 gr/t, magna flock=1.5 kg/t)

As it is seen from fig 6.9, although ash content of the concentrate has increased dramatically (5%), yield has not showed any noticable value. This shows that clay particles carry into the froth when the use of magna flock. For that reason, kerosene has been used alone as collector for the latter studies.

6.2.2.3 Effect of Frother

The frother types mentioned in chapter 6.2.1.2 have been used in experiments. The experimental results are shown in Table A-6.7 and in fig 6.10

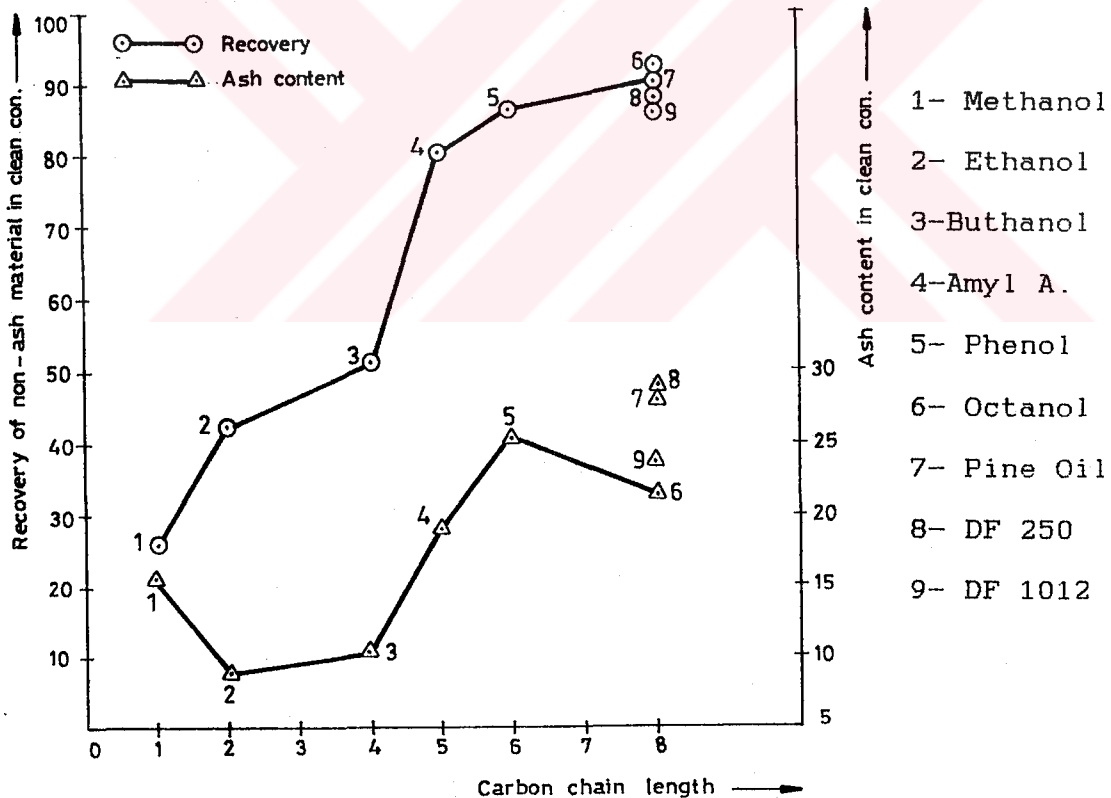


Fig 6.10 Graph showing effect of carbon chain lengths of frother material on fine coal in terms of recovery and ash content of concentrate frother material on fine coal in term (frother=200 gr/t, kerosene=1.8 kg/t)

In these series of experiments, in which the effect of strain chain aliphatic alcohol type methanol, ethanol, butanol, amyl alcohol, isooctanol, ether type DF250 and DF 1012, aromatic type phenol and cyclic type pine oil, with regard to combustible carbon yield and ash content of froth concentrate, different outcomes have been obtained. When ethanol and butanol were used, in contrast ash contents of the concentrates have been the lowest value (10%), combustible yield has hardly been around 50%. As the carbon chain increases, ash content of the concentrate increases and the combustible yield an over 80% value were obtained with the use of isooctanol, approximately 22.67% ash content was obtained with a 90.2% yield.

Low carbon chain length (2,4 C) alcohols, in an high clay contamination, it has been seen that they provide coals with selective dehidriation and so that, ash content decreases. But because of poor frother characteristics, the desired outcomes in term of yield couldn't be obtained. Although with regard to yield, amyl alcohol and isooctanol have produced the desired results with regard to ash content above desired results were obtained. This situation has come about through coal clay content's being very high, through coating the coal particles with slime and carrying the fine clay particles (water-carry over) with water.

With the ether and brached type frothers, the ash content was not decreased below 28%. Although with regard to yield it is high, it is not suitable for this type of coal.

As a result, if a very low ash content wanted, use of

ethanol, higher ash content desired especially isooctanol and isoamyl alcohol use are appropriate. Strain chain alcohols make up the most proper frother types in the high clay content of fine coal flotation.

In the end of examination of frother type effects, when it is adviced to use of ethanol for acquirement of low ash concentration the examination of the results of the ethanol dosages are shown in table A-6.8 and figure 6.11

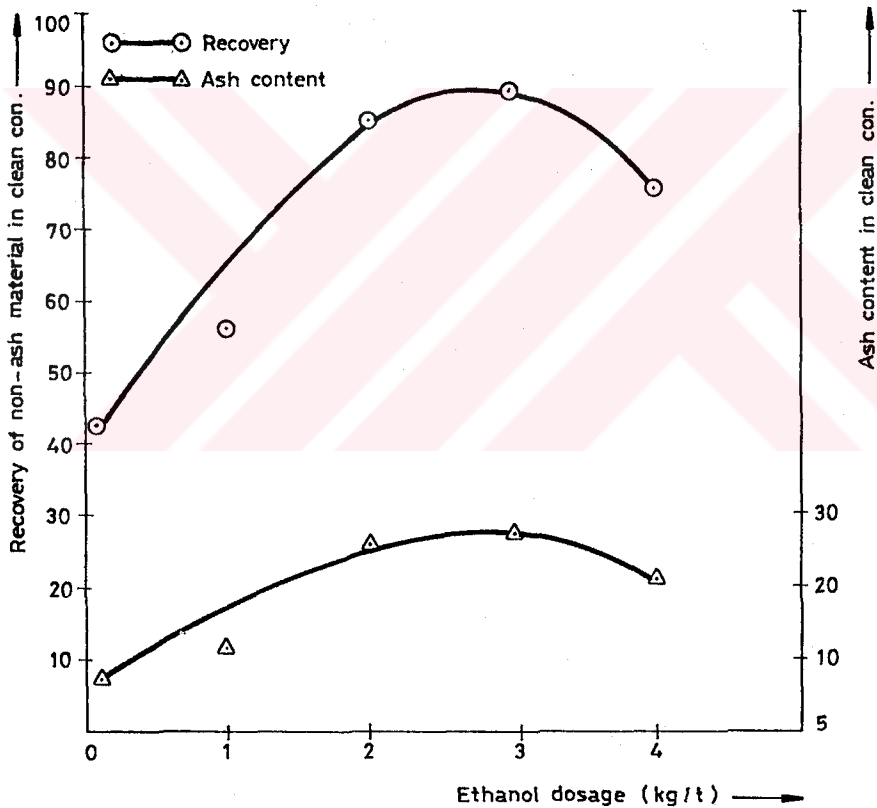


Fig 6.11 The effect of ethanol dosage (kerosene =1.8 kg/t)

As it is seen from the figure, when the amount of ethanol is increased, the yield raises but the ash content of froth

concentrate increases as well. And this shows that ethanol causes the clay material to be carried over to the froth. As a result the most proper value is about 200 gr/t.

If isooctanol is chosen as a frother type, the results of experiments done to determine isooctanol dosage are shown in table A-6.9 and figure 6.12

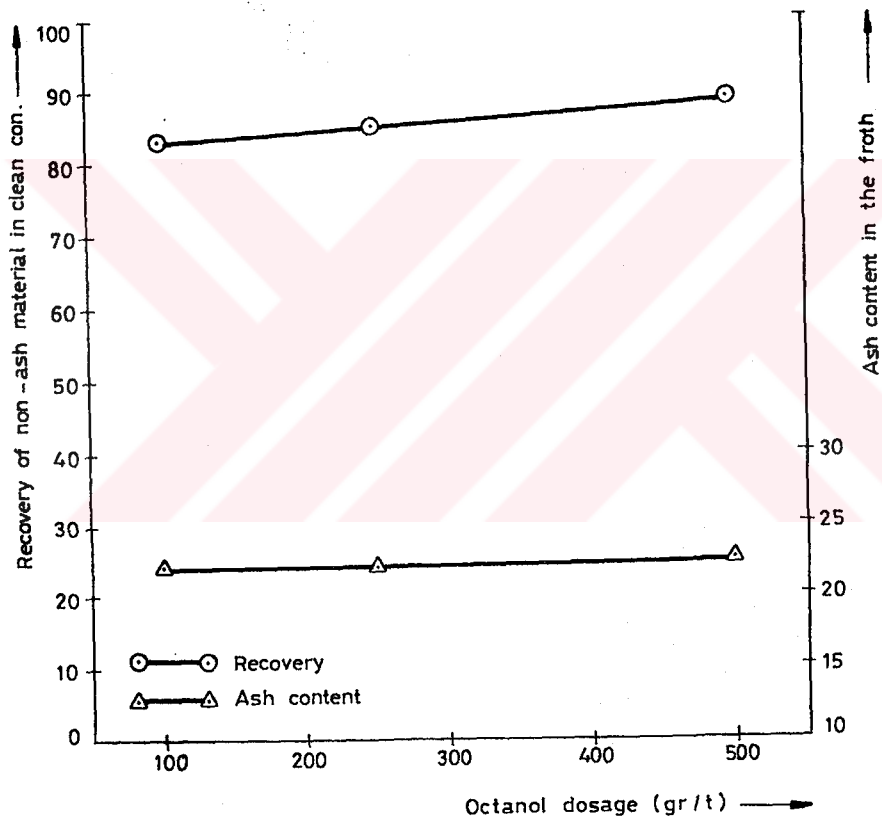


Figure 6.12 Effect of octanol dosage (kerosene=1.8 kg/t)

With the increase of octanol amount, as the yield show a partial increase, ash content simultaneously increases.

Because of this a 200 gr/t dosage is enough.

As a result the use of ethyl alcohol with very low-ash content concentrate and use of isooctanol approximately 22.67% ash content of concentrate with the use of both's mixture it is possible to obtain more appropriate ash content and high yield. The results of experiments are shown in table A-6.10 and figure 6.13

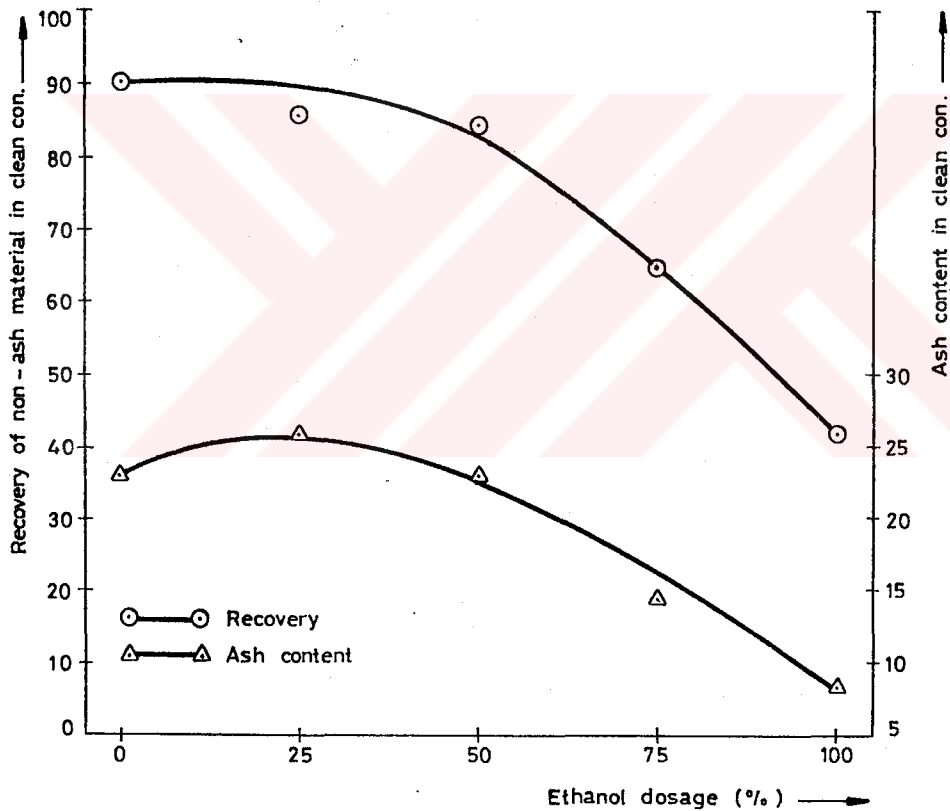


Fig 6.13 Effect of mixture of ethanol-octanol

As v/v 0, 25, 50, 75, 100 prepared mixture are used respectively in experiments, higher values of ash contents are obtained than received concentrate.

In mixture, as the yield increases by using frother, it is observed that ash is increasing as well. Ethanol has not show any selectivity effect on the coal particles because of the more powerfull effect of octanol.

In order to decrease concentrate ash content through the feeding octanol at different stages, the results are shown in fig 6.14 (Table A-6.11)

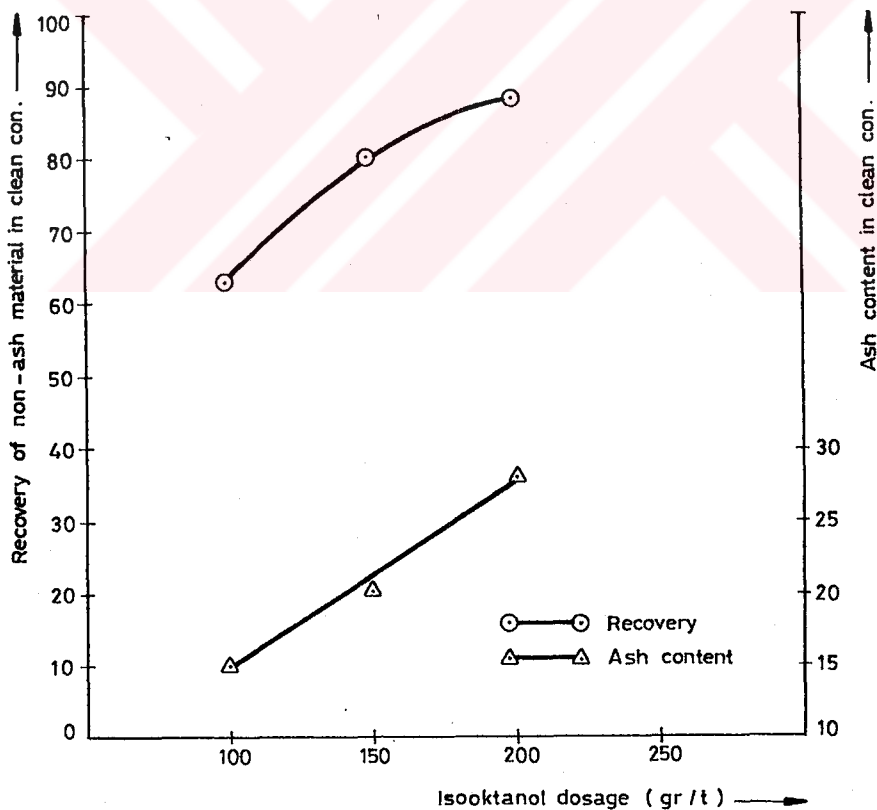


Fig 6.14 effect of octanol feeding at different stages

In the first step, low-ash contents obtained above 60% productivity. Nevertheless, clay particles are entrapped to the froth as the amount of frother increases and this causes increasing of ash content in the froth product.

In the end, there is no difference between gradual feeding and direct feeding of frother. But feeding at different stages can be applied when it is desired to obtain different ash contents.

In order to increase coal floatability, previously after a pretreatment was applied with ethanol, the experimental results which are acquired with the usage of pine oil and isooctanol tabulated in table A-6.12 and in fig 6.15

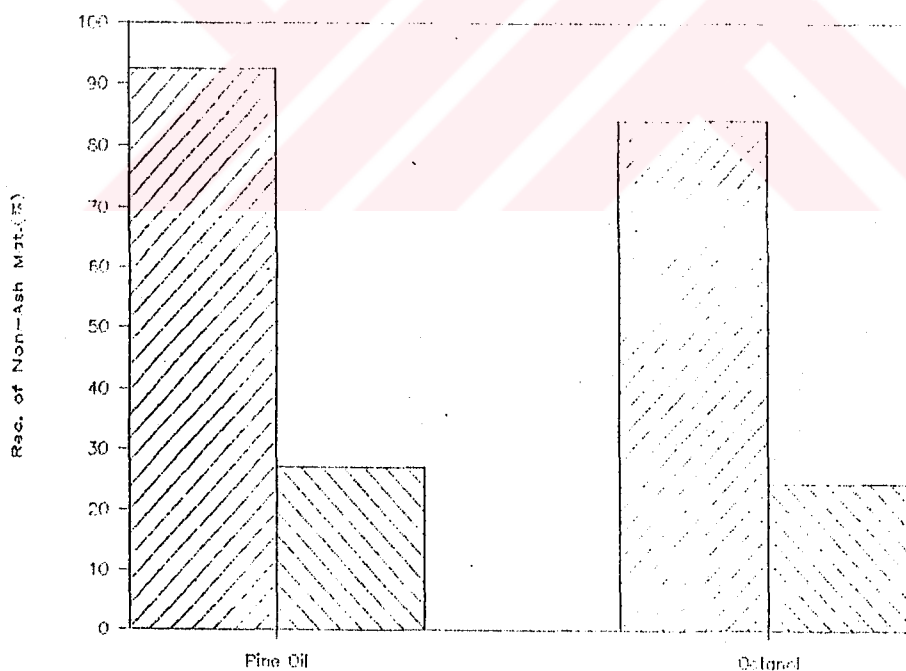


Figure 6.15 The usage of ethanol as a pretreatment.

In this case, through the usage of octanol 84.17 % yield and 24.52 ash content of concentrate have been obtained. If pine oil is used, yield is 92.43% but ash content rise to 27.15%. This does not decrease the ash content to the desired level. Because fine clay particles enter the froth as a carry-over with the water accompanying the froth product and slime coatings on the coal owing to increasing of active frother amount. For these kinds of coals, pretreatment is not suitable.

6.2.2.3.1 Gradual Usage of Ethanol and Octanol

As the use of ethanol produce froth product having low ash content and low yield, two-stage flotation circuit has been planned. (figure 6.24). The experiments done for this purpose are shown in table A-6.14.

In the first stage it is possible using ethanol to obtain 48.94 % yield and 9.61 %ash content of froth product. From the tailing of this stage, it is possible using octanol to obtain a high clay content of a product. The total yield and ash values of this operation are 89.95% and 20.15%. In this experiment, concentrates including different ash contents can be obtained.

The series of experiments have been done to determine the relation of the increase of the ash content to the compound particles (Table A-6.14)

In order to inquire the effect of grain size, sample was grinded to -0.315 mm. According to experiments, it is

possible to obtain 14.55% ash content and 62.60% yield. Through the decrease in grain size, 20% an increase in yield an increase 5% in ash content have been obtained.

To determine the bounded grain effect, the pre-concentrate in the first stage was taken with ethanol, than remnant was grinded to minus 0.315 mm. There have not been found any difference between the experiments done without grinding and after grinding. in this case it is foud that the role of bounded particles is not important.

6.2.2.4 Conditonig Time

Condioting time of slurry with reagent would play an important role in achieving optimum flotation recovery and the grade of the froth concentrate. To increase adsorption of collector (kerosene) at the coal surface, various condioting times were examined. Results are shown in table A-6.15 and fig 6.16

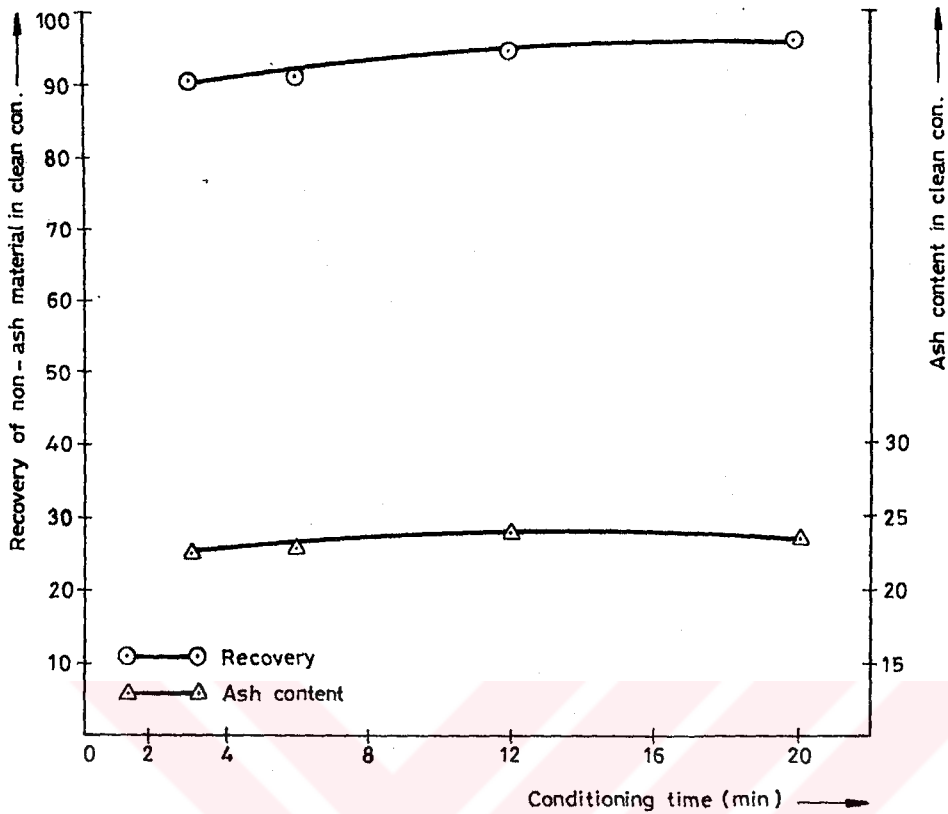


Fig 6.16 Effect of conditioning time

It is noted that when the conditioning time increases, yield and ash content of concentrate also show a slight rise. For that reason, conditioning time of 3-5 minutes is enough. Long conditioning periods perform no useful purpose.

6.2.2.5 Solid-Liquid Ratio

Coal flotation is ordinarily conducted at 5-20% solids. Results of experiments done to determine the optimum solid ratio are shown in Table A-6 16 and fig 6.17

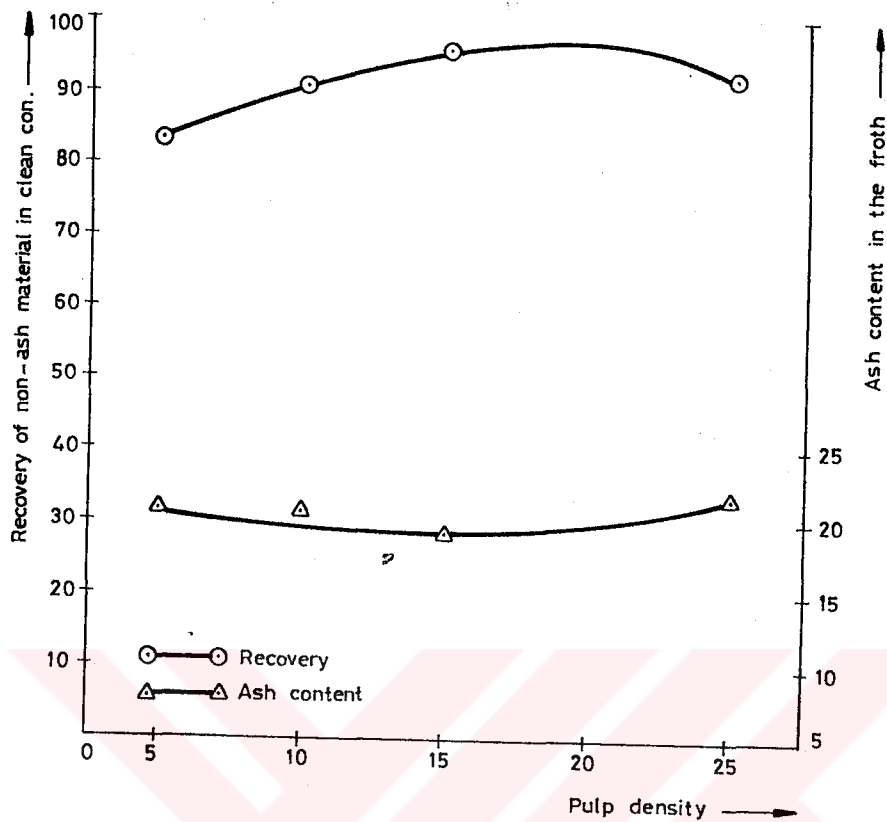


Fig 6.17 Effect of solid-liquid ratio

(kerosene=1.8 kg/t, octanol=200 gr/t)

Ash content of concentrate increases at low and high solid ratio. from the fig 6.17, optimum solid ratio is about 15%. At this point, yield increases its maximum value (94%) and ash content is near 19%.

6.2.2.6 Effect of Depressant Type

In the coal flotation including high slime contamination, in order to disperse the particles inot the pulp and depress the gangue, sodium silicate and tri-poly phosphate have been used

as depressants. In the experiments, first of all preconcentrate was taken with the use of ethanol and kerosene. Then, depressant of 1.5 kg/t was added and conditioned for ten minutes. After addition of octanol of 100 gr/t, froth was collected for 3 min. Experimental results are shown in table A-6.17 and fig 6.18

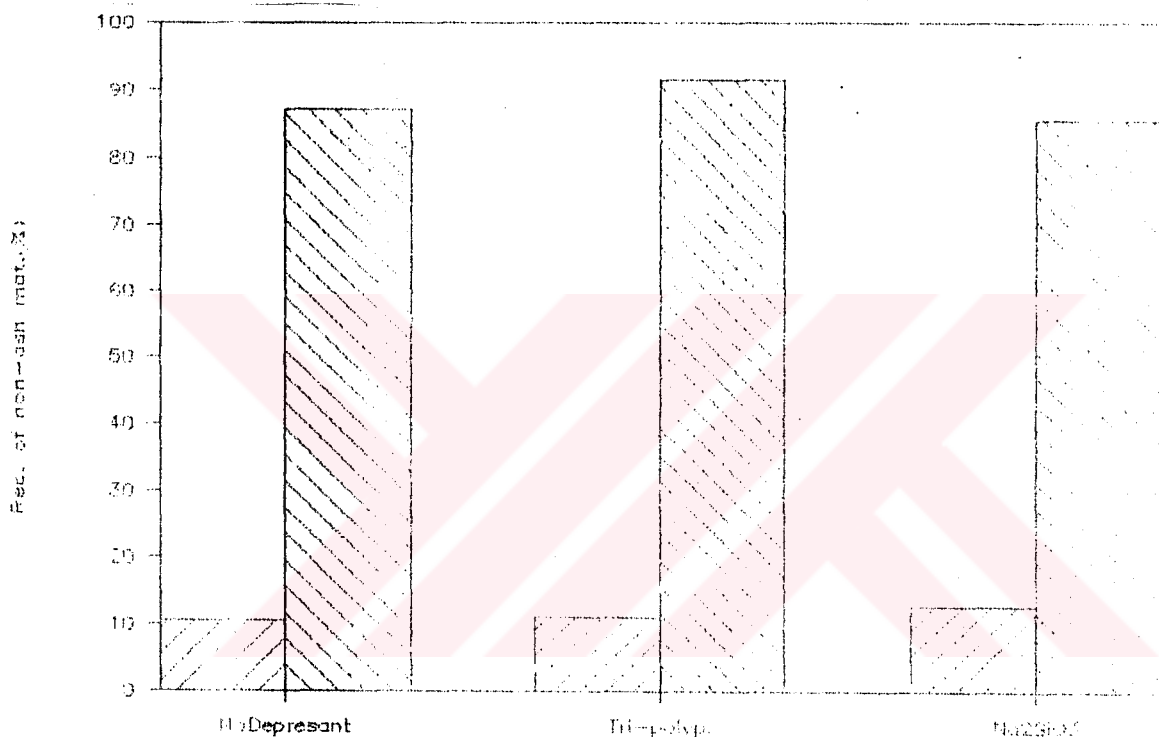


Fig 6.18 Effect of Depressant type

As it is seen from table a-6.17 and fig 6.18, when the use of sodium silicate, yield has shwed a slight decrease (2%). In contrast, use of tripolyphosphate has increased yield slightly (2%). In terms of yield and ash content of the concentrate, use of tri-polyphosphate has been suggested as a depressant

Results showed that sodium silicate depressed the flotation of coal.

6.2.2.6.1 Effect Of Amount Of Depressant

In the experiments, varies amounts of tri-polyphosphate were added and conditioned for ten minutes to determine the effect of depressant dosage. (Table A-6.18)

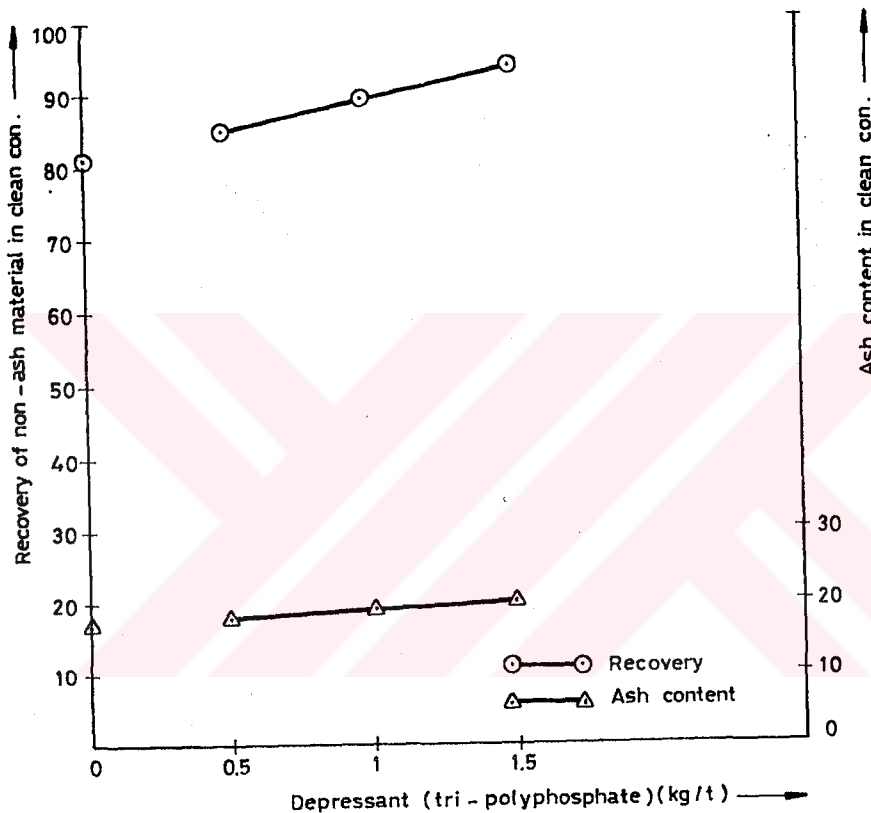


Fig. 6.19 Effect of tri-poly phosphate dosage

(Kerosene= 1.8 kg/t, octanol= 200 gr/t)

When the amount of deperssant increases, yield increases from 88.92% to 93.06% value but ash content of concentrate also shows a slight rise from 17.44% to 19.76%. It can be said that use of tri-polyphosphate has improved the flotation of coal because phosphate ions create the negative charge

density at the mineral surface resulting in their depression.
1.5 kg/t of dosage is appropriate for good recovery.

6.2.2.7 Effect Of Emulsifier.

Increases in flotation yield of coarse coal can be achieved using emulsifiers which reduce the oil/water interfacial tension and lower the energy required to spread the collector oil across the coal surface.

To determine the effect of emulsification on the flotation yield of a high clay content of feed coal, a series of flotation experiments were conducted at different levels of emulsification. The results of these tests, tabulated in table A-6.19 and plotted in fig 6.20.

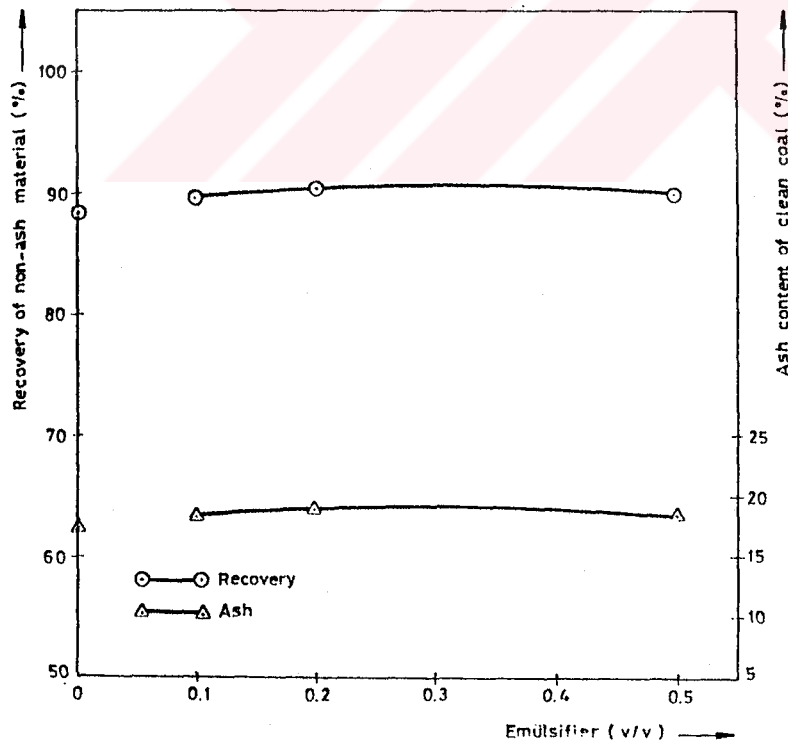


Fig 6.20 Effect of non-ionic emulsifier

(Kerosene=1.8 kg/t, octanol= 200 gr/t)

These results showed that, non-ionic emulsifier has not showed any increase in recovery relative to the unemulsified case. But it was observed that there had been a slight decrease in flotation time.

Although emulsifiers generally increase coal flotation yields, non-ionic emulsifier has not increase the probability of coal/collector contact in the pulp relative to high clay contamination.

6.2.2.8 Effect Of Some Operating Variables On The Efficiency Index Of A Coal Flotation Process.

Coal cleaning performance can be evaluated by a simple index called the efficiency index. This index is a useful parameter to compare the relative performances of the cleaning operation under various operating conditions. It is defined as follows:

$$E = \frac{\text{Recovery of non-ash material} \times \text{Tailings ash}}{\text{Product ash}} \quad \dots \text{Eqn (10)}$$

The efficiency index (E) is dependent on recoverable non-ash material and hence the performance on feed of different ash contents can be compared on the basis of this index.

In this part, an attempt has been made to study the effect of some operating variables in a coal flotation process on the efficiency index. The operating variables are

- 1- The collector dosage (Kerosene)
- 2- The frother dosage (isooctanol)
- 3- The time of flotation

Experiments have been conducted for four levels of collector and frother dosages. In the tests the following parameters were held constant at the levels indicated :

- 1- Impeller speed= 1300 rpm
- 2- Conditioning time= 3 min.
- 3- pulp density= 10 % solids by weight.

Collector dosage (kg/t):

$C_1=0.00$; $C_2=0.50$; $C_3=1.00$; $C_4= 2.00$

Frother dosage (kg/t) :

$F_1=0.050$; $F_2=0.10$; $F_3=0.20$; $F_4=0.30$

For each experiment, four concentrates were collected at time intervals of 30 seconds from the beginning of 90 th seconds. The efficiency index for each time interval was calculated using the equation 10 (Table A-6.20-23)

6.2.2.8.1 Effect of Reagent Dosage

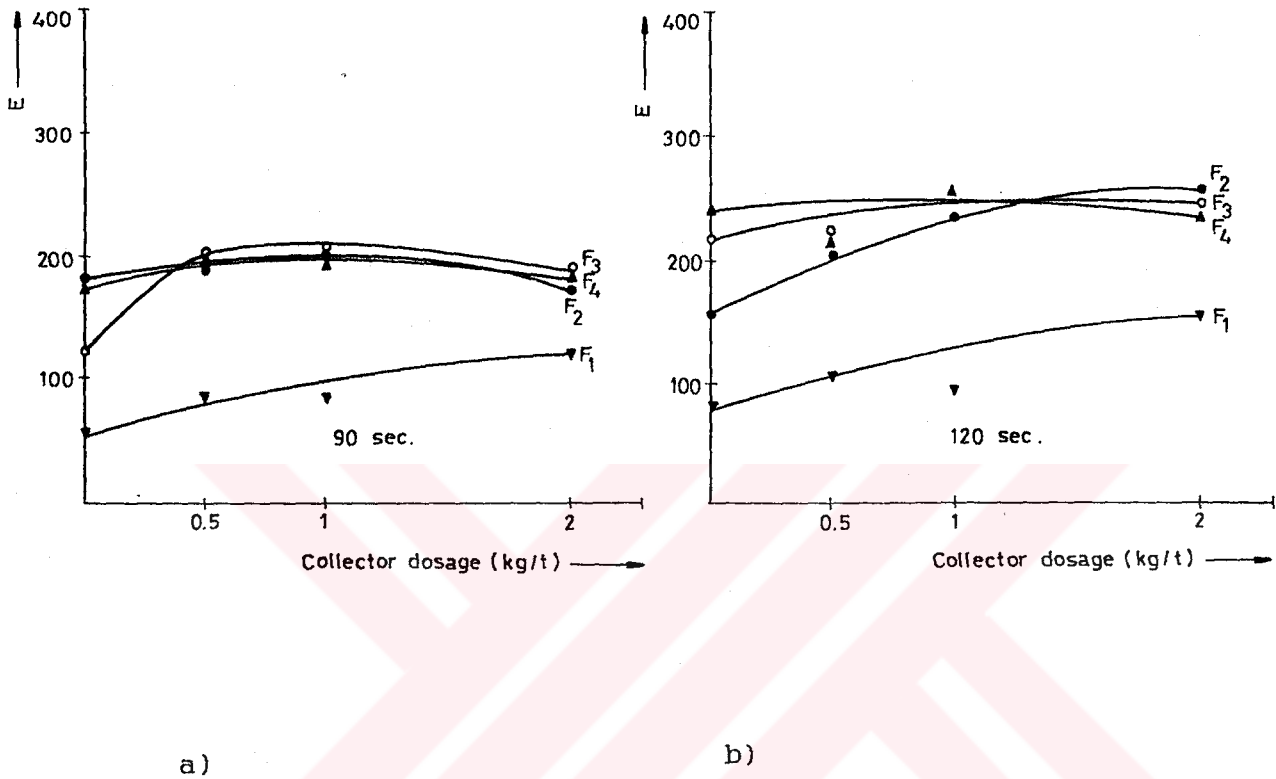


Fig 6.21 Variation of efficiency index with collector dosage

The efficiency index increases with increase in the dosage of collector added up to certain level and then decreases slightly showing a maximum as in fig 6.21a,b,c,d. At low flotation times (90 sec and 120 sec) values of efficiency index are not high.

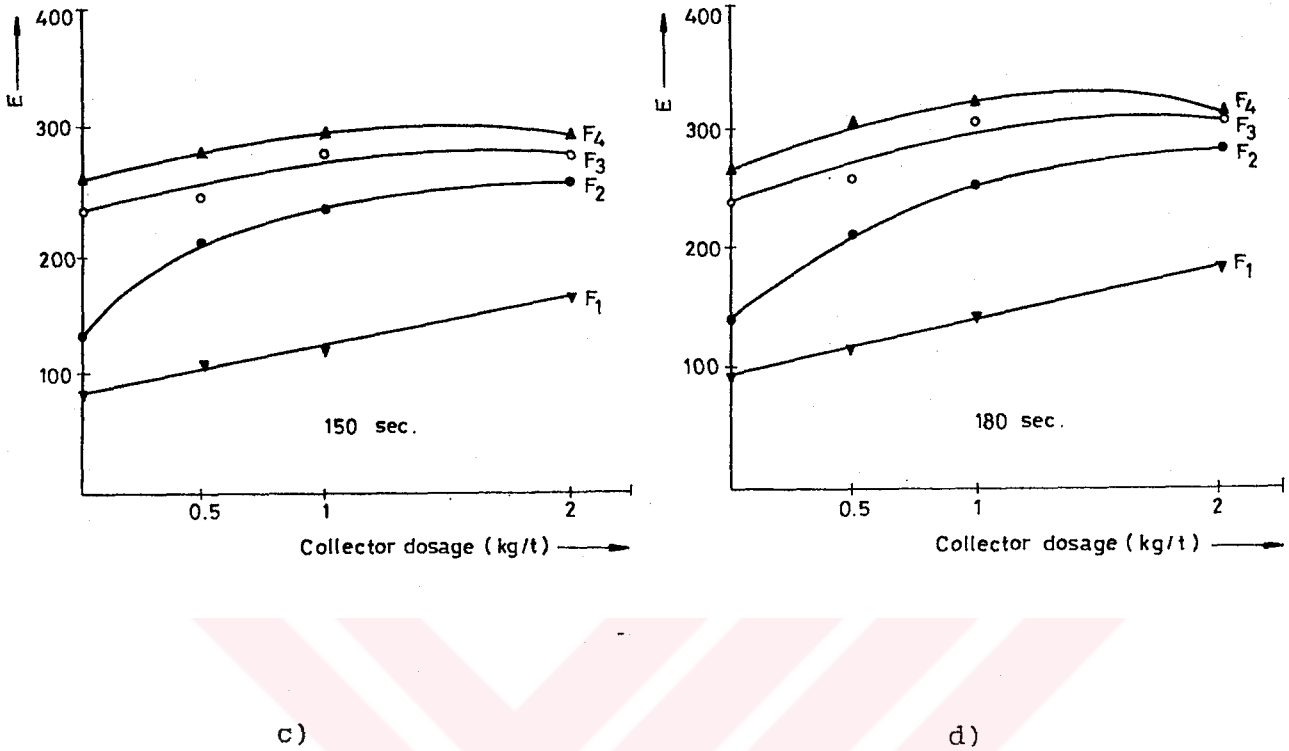


Fig 6.21 Variation of efficiency index with collector dosage.

This index increases with increasing flotation times. For a low concentration the efficiency index shows an increasing trend. The highest index values were obtained at flotation time of 180 seconds.

6.2.2.8.2 Regression Equations

In order to find the optimum collector dosage to obtain a maximum efficiency index, the efficiency index was correlated with the collector and frother dosage by using a third-degree regression equation with standard deviation of 9 percent.

The equation is given as :

$$E = a_1 + a_2F + a_3C + a_4F^2 + a_5C^2 + a_6CF + a_7C^3 + a_8F^3 + a_9C^2F + a_{10}CF^2 \dots\dots\dots \text{Eqn (11)}$$

Where, C and F are collector and frother dosages in kg/t. Here, C and F are the independent parameters and E is the dependent parameter relevant to C and F.

Using LOTUS computer programme (/DATA/REGRESSION), first, coefficients for each flotation time were determined. Then using these coefficients with the the flotation times which are independent parameters, following equations were calculated:

$$a_1 = -117.6 - 0.6253 \times t + 0.004 \times t^2$$

$$a_2 = 4440 + 15.7128 \times t - 0.105 \times t^2$$

$$a_3 = 208.469 + 1.49099 \times t + 0.0254 \times t^2$$

$$a_4 = -28715 + 8.79165 \times t + 0.29021 \times t^2$$

$$a_5 = -198.4 + 2.20526 \times t + 0.048 \times t^2$$

$$a_6 = 1014.8 + 9.108 \times t - 0.005 \times t^2$$

$$a_7 = 54.021 - 0.6785 \times t + 0.016 \times t^2$$

$$a_8 = 46223.2 - 11.271 \times t - 0.4869 \times t^2$$

$$a_9 = -160.56 + 3.77665 \times t - 0.0202 \times t^2$$

$$a_{10} = 4721.88 - 68.006 \times t + 0.206111 \times t^2$$

Thus, the coefficients of each term in eqn 11 are expressed in term of time. For the three flotation times e.g 90, 120, 150 sec. these coefficients are obtained as a linear equation time and for the four flotation times e.g 90,120,150,180 sec. as a second degree equation relative to time

To determine the coefficients in eqn 11 at any flotation times the computer programme showed in table A-6.25 was used. For the 150 sec. flotation time, equation 11 expressed as follows:

$$E = 121.935 + 4434.42x_F + 41.98xC - 20866.53x_F^2 + 24.37399xC^2 + 238.8999xCx_F - 11.75xC^3 + 33577.3x_F^3 - 48.56xC^2x_F - 841.5224xCx_F^2 \dots\dots\dots \text{Egn (12)}$$

If the frother dosage is assumed to 0.2 kg/t, eqn 12 reduces to:

$$E = 198.9 + 56.10xC + 14.658xC^2 - 11.75xC^3 \dots\dots \text{Eqn (13)}$$

To find the maximum value of E and corresponding C, eq. 13 may be differentiated and equated to zero. This equation is solved with using the computer programme including Newton-Raphson method (Table A-6.26).

From table A-6.26, C is equal to 1.74 kg/t and $E_{\max} = 278.99$. Hence a collector dosage of 1.74 kg/t gives a maximum efficiency index at a frother dosage of 0.2 kg/t for 150 sec. of flotation time.

For the other flotation times, at 0.2 kg/t frother level, C and E_{max} values are shown in table 6.4 .

Table 6.4 Calculated C and E_{max} values at 0.2 kg/t frother level

	Flotation time (sec)			
	90	120	150	180
C (kg/t)	1.15	1.74	1.74	1.80
E_{max}	195.64	245.06	278.99	300.58

At 150 sec, the optimum collector and frother dosages could be obtained by differentiating eqn 12 partially with respect to C and F, equating the two equations obtained to zero and than solving them simultaneously for C and F. The solotuion was obtained using Mewton-Raphson method applied the computer programme (table A-6.27).

From table A-6.27, the values obtained were C= 1.82 kg/t and F=0.179 kg/t and the corresponding E_{max} =282.45.

For other flotation times, C and F values were determined with the same method.

Table 6.5 Calculated C,F,Emax values at any flototation time

	Flotation Time (sec)			
	90	120	150	180
C (kg/t)	1.35	1.77	1.82	1.87
F (kg/t)	0.145	0.155	0.170	0.189
E _{max}	209.56	254.52	282.45	292.30

From table 6.5,at desired flotation time,the most proper C and F values can be obtained.

Representative comparisons of the calculated and the actual efficiency index values are presented in table A-6.28

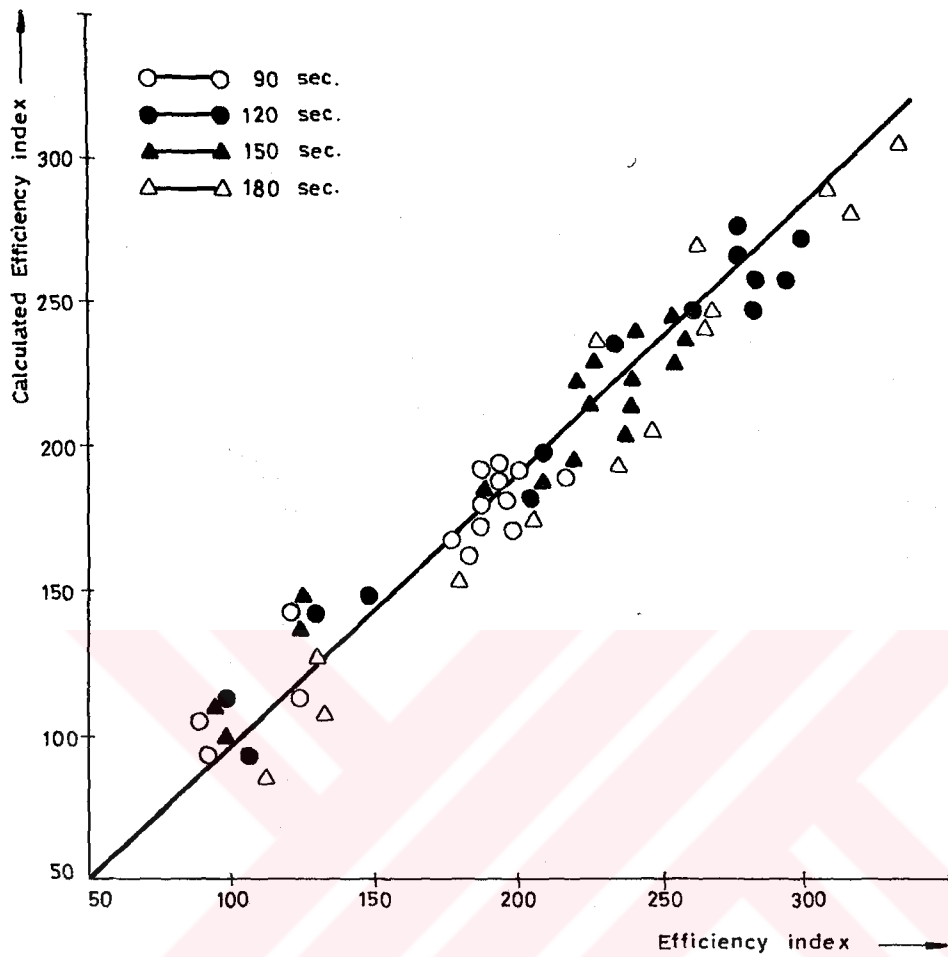


Fig 6.22 Comparison of calculated and actual efficiency index

The calculated and actual efficiency indices are plotted in fig 6.22. It may be observed that they agree within reasonable statistical limits.

6.2.3 Conclusions and Recommendations

In this study, after survey of literature about coal characteristics and coal flotation, the properties of Zonguldak coal have been presented. optimal flotation conditions for this particular coal have been determined utilizing a third-degree regression equation, the collector dosage, the frother dosage and the flotation time related with efficiency index have also been examined. Results of the experiments can be presented as follows :

- In the flotation of fine coal having ash content of 51.50 % and heating value of 3739 kcal/kg, reduction of particle size seems to have no effect in the recovery. Therefore grinding of the sample were not necessary.

- Light, medium and heavy hydrocarbon oils examined as collectors and kerosene, diesel and light neutral were chosen the most reasonable collector types respectively. Application of a mixture of these oils were not found to be as effective.

- Strain chain, aromatic, cyclic and ether type frothers were tested and the best flotation results were obtained in the use of amyl alcohol and iso octanol. To obtain low-ash content of concentrate approximately 8-9 %, the use of ethanol with 2 carbon is appropriate.

- Sodium silicate depressed the flotation of coal whereas tri-poly phosphate improved the yield of concentrate slightly. The reason for this phosphate ions create the negative charge density at the mineral surface resulting in their depression.

- From the obtained results in relation with the efficiency index, in the desired flotation time, the most proper reagent amounts have been determined by utilizing third-degree regression equation and applying computer programme. A collector dosage of 1.74 kg/t gives a maximum efficiency index at a frother dosage of 0.2 kg/t for 150 sec. of flotation time. The highest efficiency index value was obtained at 180 sec. flotation time. At this time, $C = 1.82 \text{ kg/t}$, $F = 0.189 \text{ kg/t}$ and corresponding $E_{max} = 292.30$. Comparisons of the calculated and the actual efficiency index values showed that they agree within reasonable statistical limits.

In conclusion, to obtain the best results in terms of yield and ash content of concentrate, two different flowsheets have been suggested:

- i) In the use of octanol as a frother, the ash content of concentrate is 22.67% and the yield is 90.21% ($E_{fg} = 6.23$). In addition, heating value increases from 3831 kcal/kg to 6821 kcal/kg level (Table A-6.29)
- ii) Another alternative is to use ethanol and octanol as frothers in the two-stage flotation circuit (Fig 6.24).

In the first step, using with ethanol, ash content of concentrate is 9.61% and yield is 48.94%. In this step heating value of concentrate is 7567 kcal/kg (Table A-6.30). In the second step, ash content of 38.20% and yield of 41.20% values are obtained in the use of octanol. This flotation circuit seems more appropriate when products with different ash contents are required.

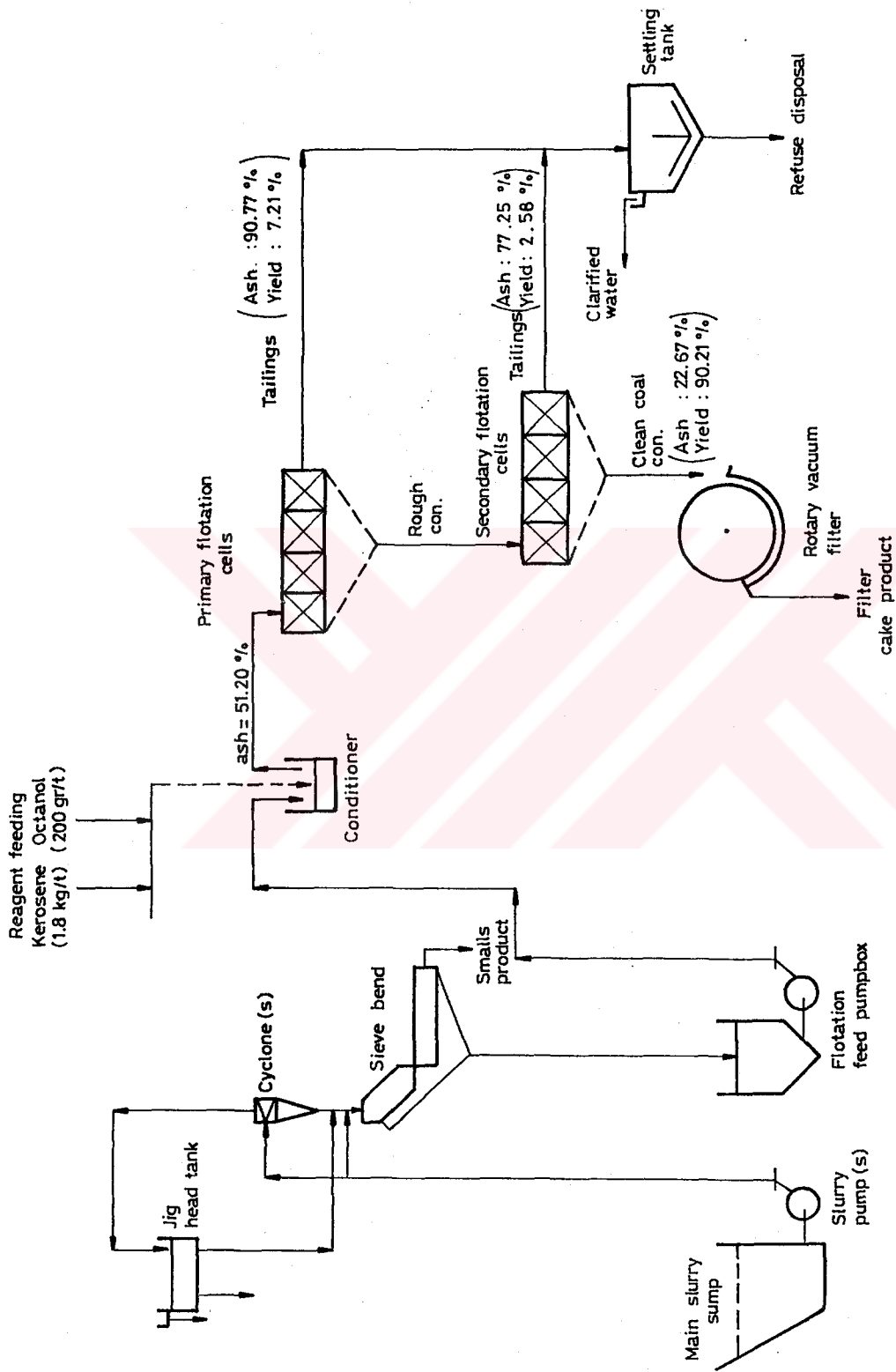


Figure 6.23. Proposed flowsheet of single stage flotation circuit.

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APPENDICES

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Table A-6.29 Analysis of concentrate obtained using with
ethanol

Table A-6.30 Analysis of concentrate obtained using with
octanol



Table A-6.1 Effect of Grain Size
(Kerosene=2.7 kg/t , Pine Oil=300 gr/t)
(Fig.6.5)

Size Range (mm)	Product	Weight (%)	Ash (%)	Recovery of Non-Ash Material (%)
Original Sample	Clean Con.	54.04	22.49	82.92
	Rough con.	66.12	72.82	90.42
	Tailing	33.88	84.22	9.58
-0.5	Clean con.	64.37	28.79	90.93
	Rough con.	70.98	33.83	93.17
	Tailing	29.02	88.14	6.83
-0.5+0.075	Clean con.	67.44	28.94	92.29
	Rough con.	74.90	34.33	94.73
	Tailing	25.10	89.09	5.27
-0.315	Clean con.	64.89	31.22	91.69
	Rough con.	72.84	37.22	93.95
	Tailing	27.16	89.16	6.05

Table A-6.2 Effect of collector type
(Pine Oil=300 gr/t , collector=2.7 kg/t)
(Fig. 6.6.a)

Collector Type	Product	Weight (%)	Ash (%)	Recovery of Non-Ash Material (%)
Diesel	Clean Con.	64.01	28.56	90.71
	Rough con.	70.70	33.76	92.90
	Tailing	29.30	87.78	7.10
Kerosene	Clean con.	63.74	30.09	91.32
	Rough con.	70.14	35.06	93.34
	Tailing	29.86	89.11	6.66
White Spirit	Clean con.	59.22	29.25	86.37
	Rough con.	68.49	35.06	91.69
	Tailing	31.51	87.21	8.31
Fuel Oil	Clean con.	63.51	31.92	89.19
	Rough con.	70.20	36.82	91.49
	Tailing	29.80	86.15	8.51
Spindle Oil	Clean con.	68.53	33.84	93.02
	Rough con.	73.96	37.73	94.48
	Tailing	26.04	89.68	5.52
Light Neutral	Clean con.	64.22	31.36	90.42
	Rough con.	71.87	36.91	93.01
	Tailing	28.13	87.89	6.99
Heavy Neutral	Clean con.	56.74	26.69	80.68
	Rough con.	65.10	34.11	83.19
	Tailing	34.90	75.18	16.81
Bright Stock	Clean con.	60.40	28.59	86.86
	Rough con.	66.50	33.39	89.20
	Tailing	33.50	83.99	10.80

Table A-6.3 Effect of collector type
(DF250=300 gr/t , collector=2.7 kg/t)
(Fig. 6.6.b)

Collector Type	Product	Weight (%)	Ash (%)	Recovery of Non-Ash Material (%)
Diesel	Clean Con.	68.65	32.87	92.52
	Rough con.	74.67	37.15	94.22
	Tailing	25.33	88.64	5.78
Kerosène	Clean con.	55.97	20.92	89.82
	Rough con.	68.42	27.59	90.10
	Tailing	31.58	83.24	9.90
White Spirit	Clean con.	57.85	25.48	78.65
	Rough con.	67.08	32.39	91.41
	Tailing	32.98	85.71	8.59
Fuel Oil	Clean con.	65.37	30.11	89.99
	Rough con.	77.70	38.48	94.15
	Tailing	22.30	86.67	5.85
Spindle Oil	Clean con.	65.61	32.57	90.24
	Rough con.	73.26	38.03	92.59
	Tailing	26.74	86.43	7.41
Light Neutral	Clean con.	59.20	26.87	86.65
	Rough con.	65.54	32.07	88.71
	Tailing	34.46	83.72	11.29
Heavy Neutral	Clean con.	64.11	30.88	89.48
	Rough con.	72.73	36.79	92.19
	Tailing	27.27	85.82	7.81
Bright Stock	Clean con.	61.90	31.79	82.47
	Rough con.	70.45	35.95	88.14
	Tailing	29.55	79.47	11.86

Table A-6.4 Effect of mixture of collector
(Pine Oil=300 gr/t , mixture of collector=2.7 kg/t)
(Fig. 6.7)

Mixture of Collector (V/V=1)	Product	Weight (%)	Ash (%)	Recovery of Non-Ash Material (%)
Diesel+Kerosene	Clean Con.	65.61	30.26	91.59
	Rough con.	71.97	35.16	93.40
	Tailing	28.03	88.24	6.60
Light+Diesel	Clean con.	57.14	26.95	86.16
	Rough con.	69.80	35.52	92.68
	Tailing	30.20	88.22	7.32
Light+Kerosene	Clean con.	66.84	33.93	91.82
	Rough con.	73.91	38.90	93.88
	Tailing	26.09	88.72	6.12
Light+Kerosene+Diesel	Clean con.	66.34	30.89	92.49
	Rough con.	72.51	35.74	94.00
	Tailing	27.49	89.18	6.00

Table A-6.5 Effect of Collector (kerosene) dosage
(Isooctanol=200 gr/t)
(Fig. 6.8)

Kerosene Dosage (kg/t)	Product	Weight (%)	Ash (%)	Recovery of Non-Ash Material (%)
0	Clean con.	50.25	19.39	71.16
	Rough con.	59.32	23.32	80.23
	Tailing	40.68	73.34	19.77
1	Clean con.	44.55	20.26	72.94
	Rough con.	63.98	30.53	91.26
	Tailing	36.02	88.57	8.74
2	Clean con.	54.48	23.06	86.41
	Rough con.	65.13	31.08	92.54
	Tailing	34.87	89.62	7.46
4	Clean con.	65.63	29.19	95.80
	Rough con.	71.70	33.94	97.64
	Tailing	28.30	95.95	2.36
8	Clean con.	68.19	32.39	94.28
	Rough con.	73.61	36.89	95.76
	Tailing	26.39	92.20	4.24

Table A-6.6 Effect of flocculant
 (Collector(Kerosene)=2.7 kg/t,
 Magnaflock=1.5 kg/t , Pine Oil=300 gr/t)
 (Fig. 6.9)

Flocculant type	Product	Weight (%)	Ash (%)	Recovery of Non-Ash Material (%)
Magnaflock	Clean con.	68.12	35.92	91.63
	Rough con.	73.82	39.53	93.69
	Tailing	26.18	88.53	6.31
No-flocculant	Clean con.	63.74	30.09	91.32
	Rough con.	70.14	35.06	93.34
	Tailing	29.86	89.11	6.66

Table A-6.7 Effect of frother carbon chain length
(frother=200 gr/t , Kerosene=1.8 kg/t)
(Fig. 6.10)

Frother Type	Carbon Length	Product	Weight (%)	Ash (%)	Recovery of Non-Ash Material (%)
Methanol	1	Clean con.	16.21	21.44	26.25
		Rough con.	18.34	24.16	28.67
		Tailing	81.66	57.63	71.33
Ethanol	2	Clean con.	22.38	8.28	42.31
		Rough con.	30.35	13.14	54.34
		Tailing	69.65	68.20	45.66
Buthanol	4	Clean con.	27.86	10.59	51.35
		Rough con.	52.60	21.21	85.44
		Tailing	47.40	85.09	14.86
Amyl Alcohol	5	Clean con.	47.41	18.00	80.14
		Rough con.	63.67	29.65	92.33
		Tailing	36.33	89.76	7.67
I.Octanol	8	Clean con.	56.59	22.67	90.21
		Rough con.	62.10	27.51	92.79
		Tailing	37.90	90.77	7.21
Phenol	6	Clean con.	58.22	25.55	85.85
		Rough con.	66.89	32.22	89.80
		Tailing	33.11	84.45	10.20
Pine Oil	10	Clean con.	56.29	24.65	87.43
		Rough con.	64.48	31.42	91.15
		Tailing	35.52	87.92	8.85
DF250	10	Clean con.	61.19	28.15	90.64
		Rough con.	66.50	32.35	92.75
		Tailing	33.50	89.50	7.25
DF1012	10	Clean con.	60.56	28.27	88.36
		Rough con.	66.14	32.26	91.14
		Tailing	33.86	87.13	8.86

Table A-6.8 Effect of Ethanol Dosage
(Kerosene=1.8 kg/t)
(Fig. 6.11)

Ethanol Dosage (Kg/t)	Product	Weight (%)	Ash (%)	Recovery of Non-Ash Material (%)
0.2	Clean con.	22.38	8.28	42.31
	Rough con.	30.35	13.14	54.34
	Tailing	69.65	68.20	45.66
1.0	Clean con.	31.58	12.18	56.93
	Rough con.	40.69	16.15	70.04
	Tailing	59.31	75.39	29.96
2.0	Clean con.	56.67	26.61	85.23
	Rough con.	64.82	29.50	93.81
	Tailing	35.18	81.43	6.19
3.0	Clean con.	59.71	27.13	89.32
	Rough con.	62.30	29.35	90.36
	Tailing	37.70	87.55	9.64
4.0	Clean con.	46.75	21.22	75.61
	Rough con.	54.66	25.32	83.80
	Tailing	45.34	76.91	16.20

Table A-6.9 Effect of octanol dosage
(Kerosene=1.8 kg/t)
(Fig. 6.12)

Octanol Dosage (gr/t)	Product	Weight (%)	Ash (%)	Recovery of Non-Ash Material (%)
100	Clean con.	53.22	24.02	83.02
	Rough con.	58.49	28.47	85.89
	Tailing	41.51	83.44	14.11
250	Clean con.	55.58	24.78	85.83
	Rough con.	60.01	28.48	88.11
	Tailing	39.99	85.52	11.89
500	Clean con.	57.92	25.31	88.81
	Rough con.	62.32	28.88	91.05
	Tailing	37.64	88.42	8.95

Table A-6.10 Effect of mixture of ethanol-octanol usage
(Kerosene=1.8 kg/t, mixture of frothers=200 gr/t)
(Fig. 6.13)

Ethanol ratio (Volume/Volume)	Product	Weight (%)	Ash (%)	Recovery of Non-Ash Material (%)
0.0	Clean con.	56.59	22.67	90.21
	Rough con.	62.10	27.51	92.79
	Tailing	37.90	90.77	7.21
0.25	Clean con.	56.71	25.81	86.14
	Rough con.	62.76	30.64	89.12
	Tailing	37.24	85.73	10.88
0.50	Clean con.	53.51	22.68	84.71
	Rough con.	59.86	27.70	88.56
	Tailing	40.16	86.09	11.44
0.75	Clean con.	36.83	14.64	64.37
	Rough con.	52.93	24.13	82.22
	Tailing	47.07	81.55	17.78
1.0	Clean con.	22.38	8.28	42.31
	Rough con.	30.35	13.14	54.34
	Tailing	69.65	68.20	45.66

Table A-6.11 Effect of Feeding at Different Stages
(Kerosene=1.8 kg/t)
(Fig.6.14)

Feeding at Different Stages (gr/t)	Product	Weight (%)	Ash (%)	Recovery of Non-Ash Material (%)
100	Rough con.	36.37	15.06	63.41
	Tailing	63.63	71.91	36.59
150	Clean con.	36.37	15.06	63.41
	Rough con.	50.15	20.43	81.92
	Tailing	49.85	82.34	18.08
200	Clean con.	50.15	20.43	81.92
	Rough con.	58.88	27.14	88.07
	Tailing	41.12	85.87	11.93

Table A-6.12 Effect of ethanol using as a pretreatment
(Kerosene=1.8 kg/t)
(Fig.6.15)

Frother Type	Product	Weight (%)	Ash (%)	Recovery of Non-Ash Material (%)
Pine Oil	Clean con.	62.49	27.15	92.43
	Rough con.	67.29	31.16	94.05
	Tailing	32.71	92.69	5.95
Octanol	Clean con.	54.32	24.52	84.17
	Rough con.	60.00	29.23	87.17
	Tailing	40.00	84.37	12.83

Table A-6.13 Gradual usage of ethanol and octanol
(Kerosene=1.8 kg/t , ethanol=200 gr/t)

Frother Type	Product	Weight (%)	Ash (%)	Recovery of Non-Ash Material (%)
Ethanol	Clean con.	31.17	9.61	48.94
	Rough con.			
	Tailing			
Isooctanol	Clean con.	37.67	38.20	41.01
	Rough con.	68.84	25.25	89.95
	Tailing	31.16	81.43	10.05

Table A-6.14 Gradual usage of ethanol and octanol
(First ethanol and than octanol)

Size Fraction (mm)	Product	Weight (%)	Ash (%)	Recovery of Non-Ash Material (%)
Original Sample	Clean con.	30.52	10.50	47.74
	Rough con.	61.17	26.71	87.00
	Tailing	38.83	18.62	13.00
-0.315	Clean con.	41.91	14.55	62.60
	Rough con.	72.08	26.64	92.42
	Tailing	27.92	84.47	7.58
Second Step . Below to 0.315	Clean con.	27.83	12.70	42.76
	Rough con.	72.68	26.65	93.83
	Tailing	27.32	87.15	6.17

Table A-6.15 Effect of Conditioning Time
(Kerosene=1.8 kg/t , octanol=200 gr/t)
(Fig.6.16)

Conditioning Time (min)	Product	Weight (%)	Ash (%)	Recovery of Non-Ash Material (%)
3	Clean con.	56.59	22.67	90.21
	Rough con.	62.10	27.51	92.79
	Tailing	37.90	90.77	7.21
6	Clean con.	58.74	22.93	90.76
	Rough con.	63.85	27.45	91.44
	Tailing	36.15	87.06	8.56
12	Clean con.	63.01	23.37	94.67
	Rough con.	66.81	26.37	96.44
	Tailing	33.19	94.55	3.56
20	Clean con.	63.52	23.45	95.53
	Rough con.	67.19	26.38	97.17
	Tailing	32.81	25.31	2.83

Table A-6.16 Effect of Solid-Liquid Ratio
(Kerosene=1.8 kg/t , isooctanol=200 gr/t)
(Fig.6.17)

Solid-Liquid Ratio by weight	Product	Weight (%)	Ash (%)	Recovery of Non-Ash Material (%)
5	Clean con.	52.43	22.34	83.61
	Rough con.	56.35	25.25	86.49
	Tailing	43.65	84.91	13.51
10	Clean con.	56.59	22.67	90.21
	Rough con.	62.10	27.51	92.79
	Tailing	37.90	90.77	7.21
15	Clean con.	58.04	19.06	96.44
	Rough con.	62.75	23.68	98.32
	Tailing	32.25	97.80	1.68
20	Clean con.	59.19	23.59	92.85
	Rough con.	64.31	28.18	94.82
	Tailing	35.69	92.93	5.18

Table A-6.17 Effect of Depressant Type
(Kerosene=1.8 kg/t, ethanol=200 gr/t,
depressant=1.5 kg/t)
(Fig.6.18)

Depressant Type	Product	Weight (%)	Ash (%)	Recovery of Non-Ash Material (%)
Na ₂ SiO ₃	Clean con.	34.94	12.90	46.63
	Rough con.	71.60	25.46	85.52
	Tailing	28.40	86.47	17.48
NoDepressant	Clean con.	30.52	10.50	47.74
	Rough con.	61.17	26.71	87.00
	Tailing	38.83	18.62	13.00
Tri-polyphosphate	Clean con.	41.20	11.25	63.90
	Rough con.	69.84	25.06	91.53
	Tailing	30.16	83.94	8.47

Table A-6.18 Effect of Tri-polyphosphate Dosage
(Fig.6.19)

Tri-phosphate dosage (kg/t)	Product	Weight (%)	Ash (%)	Non-Ash Material (%)
0	Clean con.	61.61	17.44	88.82
	Rough con.	65.01	19.60	91.28
	Tailing	34.99	85.86	8.24
0.5	Clean con.	61.89	18.14	89.76
	Rough con.	66.38	21.05	91.52
	Tailing	33.62	85.78	8.48
1.0	Clean con.	63.10	18.41	89.99
	Rough con.	67.33	21.28	92.93
	Tailing	32.67	85.87	7.07
1.5	Clean con.	66.27	19.76	93.06
	Rough con.	69.75	22.47	94.53
	Tailing	30.25	89.64	5.47

Table A-6.19 Effect of Non-ionic Emulsifier Type
(Kerosene=1.8 kg/t, octanol=200 gr/t,

(Fig.6.20)

Emulsifier/collector (Volume/Volume)	Product	Weight (%)	Ash (%)	Non-Ash Material (%)
0	Clean con.	61.61	17.44	88.82
	Rough con.	65.01	19.60	91.28
	Tailing	34.99	85.86	8.24
0.1	Clean con.	62.34	19.92	89.44
	Rough con.	66.02	20.39	91.92
	Tailing	33.98	85.26	8.08
0.2	Clean con.	63.87	18.42	90.45
	Rough con.	68.52	20.39	93.19
	Tailing	31.48	88.11	6.81
0.5	Clean con.	62.95	18.59	89.96
	Rough con.	67.20	20.89	92.28
	Tailing	32.80	86.45	7.72

Table A-6.20a Effect of some operating variables on the coal flotation

Flotation time = 90 sec

Frother dosage = 0.05 kg/t

(Fig. 6.21a)

Kerosene (kg/t)	Product	Weight (%)	Ash (%)	Recovery of Non-ash mat.	Efficiency Index (E)
0	Rough con. Tailing	22.49 77.51	20.95 49.22	31.11 68.89	73.09
0.5	Rough con. Tailing	22.38 77.62	17.90 50.05	32.15 67.85	80.80
1	Rough con. Tailing	25.18 74.82	20.70 50.31	34.94 65.06	84.92
2	Rough con. Tailing	33.89 66.11	20.44 54.35	47.18 52.82	125.45

Table A-6.20b Effect of some operating variables on the coal flotation

Flotation time = 90 sec

Frother dosage = 0.1 kg/t

(Fig. 6.21a)

Kerosene (kg/t)	Product	Weight (%)	Ash (%)	Recovery of Non-ash mat.	Efficiency Index (E)
0	Rough con. Tailing	32.95 67.05	20.02 54.08	46.12 53.98	124.58
0.5	Rough con. Tailing	40.77 59.23	18.42 59.68	58.21 41.79	188.60
1	Rough con. Tailing	47.52 52.48	19.49 64.02	66.95 33.05	216.63
2	Rough con. Tailing	45.46 54.54	20.66 61.36	63.12 36.88	187.46

Table A-6.20c Effect of some operating variables on the coal
flotation

Flotation time = 90 sec

Frother dosage = 0.2 kg/t

(Fig. 6.21a)

Kerosene (kg/t)	Product	Weight (%)	Ash (%)	Recovery of Non-ash mat.	Efficiency Index (E)
0	Rough con.	39.38	17.65	55.91	183.16
	Tailing	60.62	57.82	44.09	
0.5	Rough con.	48.61	21.48	69.87	188.65
	Tailing	51.39	61.52	34.13	
1	Rough con.	53.86	22.71	71.77	203.90
	Tailing	46.14	64.52	28.29	
2	Rough con.	52.05	22.90	69.19	189.53
	Tailing	47.95	62.73	30.81	

Table A-6.20d Effect of some operating variables on the coal
flotation

Flotation time = 90 sec

Frother dosage = 0.3 kg/t

(Fig. 6.21a)

Kerosene (kg/t)	Product	Weight (%)	Ash (%)	Recovery of Non-ash mat.	Efficiency Index (E)
0	Rough con.	40.50	18.36	57.00	180.34
	Tailing	59.50	58.09	43.00	
0.5	Rough con.	48.32	20.60	65.56	197.32
	Tailing	51.68	62.00	34.44	
1	Rough con.	49.50	21.20	67.08	198.87
	Tailing	51.50	62.85	32.92	
2	Rough con.	54.77	23.75	72.07	194.87
	Tailing	45.23	64.22	27.93	

Table A-6.21a Effect of some operating variables on the coal flotation

Flotation time = 120 sec

Frother dosage = 0.05 kg/t

(Fig. 6.21b)

Kerosene (kg/t)	Product	Weight (%)	Ash (%)	Recovery of Non-ash mat.	Efficiency Index (E)
0	Rough con.	27.78	22.84	37.51	83.02
	Tailing	72.22	50.55	62.49	
0.5	Rough con.	26.16	18.50	37.31	103.82
	Tailing	73.84	51.48	62.49	
1	Rough con.	29.54	21.37	40.65	98.65
	Tailing	70.46	51.86	59.35	
2	Rough con.	42.70	22.18	58.15	152.77
	Tailing	57.30	58.27	41.85	

Table A-6.21b Effect of some operating variables on the coal flotation

Flotation time = 120 sec

Frother dosage = 0.1 kg/t

(Fig. 6.21a)

Kerosene (kg/t)	Product	Weight (%)	Ash (%)	Recovery of Non-ash mat.	Efficiency Index (E)
0	Rough con.	35.15	20.44	48.94	131.71
	Tailing	64.85	35.01	51.06	
0.5	Rough con.	44.55	18.83	63.29	208.93
	Tailing	55.45	62.16	36.71	
1	Rough con.	52.59	20.70	72.98	237.76
	Tailing	47.41	67.44	27.02	
2	Rough con.	57.81	22.06	78.85	255.03
	Tailing	42.19	71.35	21.15	

Table A-6.21c Effect of some operating variables on the coal flotation

Flotation time = 120 sec

Frother dosage = 0.2 kg/t

(Fig. 6.21b)

Kerosene (kg/t)	Product	Weight (%)	Ash (%)	Recovery of Non-ash mat.	Efficiency Index (E)
0	Rough con.	47.65	18.97	66.57	219.67
	Tailing	52.35	62.96	33.43	
0.5	Rough con.	55.00	21.98	73.98	223.69
	Tailing	45.00	66.46	26.02	
1	Rough con.	62.32	23.38	82.31	256.75
	Tailing	37.61	72.93	17.69	
2	Rough con.	60.96	23.43	80.48	243.85
	Tailing	39.04	70.99	19.52	

Table A-6.21d Effect of some operating variables on the coal flotation

Flotation time = 120 sec

Frother dosage = 0.3 kg/t

(Fig. 6.21b)

Kerosene (kg/t)	Product	Weight (%)	Ash (%)	Recovery of Non-ash mat.	Efficiency Index (E)
0	Rough con.	50.59	19.00	70.65	243.70
	Tailing	49.41	65.54	29.35	
0.5	Rough con.	52.90	21.10	71.95	223.22
	Tailing	47.10	65.46	28.05	
1	Rough con.	59.41	22.87	80.19	252.88
	Tailing	40.59	72.12	19.81	
2	Rough con.	61.34	24.49	79.93	228.17
	Tailing	38.66	69.91	20.07	

Table A-6.22a Effect of some operating variables on the coal flotation

Flotation time = 150 sec

Frother dosage = 0.05 kg/t

(Fig. 6.21c)

Kerosene (kg/t)	Product	Weight (%)	Ash (%)	Recovery of Non-ash mat.	Efficiency Index (E)
0	Rough con. Tailing	31.32 68.68	24.03 51.44	41.63 58.37	89.11
0.5	Rough con. Tailing	33.47 66.53	22.88 52.91	45.17 54.83	104.45
1	Rough con. Tailing	35.81 64.19	22.95 53.96	48.28 51.72	113.52
2	Rough con. Tailing	46.02 53.92	22.78 60.04	62.19 37.81	163.91

Table A-6.22b Effect of some operating variables on the coal flotation

Flotation time = 150 sec

Frother dosage = 0.1 kg/t

(Fig. 6.21c)

Kerosene (kg/t)	Product	Weight (%)	Ash (%)	Recovery of Non-ash mat.	Efficiency Index (E)
0	Rough con. Tailing	38.57 61.43	22.11 55.88	52.57 47.43	132.86
0.5	Rough con. Tailing	46.25 53.75	19.49 62.96	69.16 34.84	210.49
1	Rough con. Tailing	54.87 45.13	21.30 69.07	75.57 24.43	238.03
2	Rough con. Tailing	59.17 40.83	22.28 72.68	80.48 19.52	262.54

Table A-6.22c Effect of some operating variables on the coal flotation

Flotation time = 150 sec
 Frother dosage = 0.2 kg/t
 (Fig. 6.21c)

Kerosene (kg/t)	Product	Weight (%)	Ash (%)	Recovery of Non-ash mat.	Efficiency Index (E)
0	Rough con. Tailing	50.93 49.07	19.69 65.15	70.51 29.49	233.30
0.5	Rough con. Tailing	61.14 38.86	23.46 71.15	80.67 19.39	244.66
1	Rough con. Tailing	65.70 34.30	23.62 77.10	86.52 13.48	282.78
2	Rough con. Tailing	66.95 33.05	24.33 77.78	87.34 12.66	279.22

Table A-6.22d Effect of some operating variables on the coal flotation

Flotation time = 150 sec
 Frother dosage = 0.3 kg/t
 (Fig. 6.21c)

Kerosene (kg/t)	Product	Weight (%)	Ash (%)	Recovery of Non-ash mat.	Efficiency Index (E)
0	Rough con. Tailing	54.03 45.97	89.74 68.15	74.76 25.24	258.10
0.5	Rough con. Tailing	60.99 39.01	21.75 73.65	82.27 17.73	278.58
1	Rough con. Tailing	64.83 35.17	23.09 79.31	87.26 12.74	299.72
2	Rough con. Tailing	69.12 30.88	24.65 81.00	89.87 10.13	295.31

Table A-6.23a Effect of some operating variables on the coal
flotation

Flotation time = 180 sec
Frother dosage = 0.05 kg/t
(Fig. 6.21d)

Kerosene (kg/t)	Product	Weight (%)	Ash (%)	Recovery of Non-ash mat.	Efficiency Index (E)
0	Rough con. Tailing	34.44 65.56	24.20 52.66	45.68 54.32	99.40
0.5	Rough con. Tailing	39.03 60.97	25.40 54.03	50.95 49.05	108.38
1	Rough con. Tailing	42.18 57.82	23.92 56.67	56.16 43.84	133.05
2	Rough con. Tailing	49.75 50.25	23.87 62.14	66.72 33.28	177.41

Table A-6.23b Effect of some operating variables on the coal
flotation

Flotation time = 180 sec
Frother dosage = 0.1 kg/t
(Fig. 6.21d)

Kerosene (kg/t)	Product	Weight (%)	Ash (%)	Recovery of Non-ash mat.	Efficiency Index (E)
0	Rough con. Tailing	42.37 57.63	23.28 57.25	56.85 43.11	139.90
0.5	Rough con. Tailing	52.90 47.10	22.38 65.86	71.86 28.14	211.47
1	Rough con. Tailing	56.54 43.46	21.55 70.78	77.62 22.38	254.22
2	Rough con. Tailing	63.52 36.48	23.19 77.11	85.39 14.61	283.93

Table A-6.23c Effect of some operating variables on the coal flotation

Flotation time = 180 sec

Frother dosage = 0.2 kg/t

(Fig. 6.21d)

Kerosene (kg/t)	Product	Weight (%)	Ash (%)	Recovery of Non-ash mat.	Efficiency Index (E)
0	Rough con. Tailing	54.43 45.47	20.92 67.17	74.21 25.79	238.27
0.5	Rough con. Tailing	64.53 35.47	24.16 74.45	84.38 15.62	260.02
1	Rough con. Tailing	68.08 31.92	23.69 81.04	89.57 10.43	306.11
2	Rough con. Tailing	70.15 29.85	24.79 82.43	90.96 9.04	302.45

Table A-6.23d Effect of some operating variables on the coal flotation

Flotation time = 180 sec

Frother dosage = 0.3 kg/t

(Fig. 6.21d)

Kerosene (kg/t)	Product	Weight (%)	Ash (%)	Recovery of Non-ash mat.	Efficiency Index (E)
0	Rough con. Tailing	58.63 41.37	21.40 71.18	79.45 20.55	264.26
0.5	Rough con. Tailing	64.47 35.53	22.18 77.95	86.49 13.51	303.96
1	Rough con. Tailing	67.83 32.17	23.23 84.25	91.13 8.87	330.51
2	Rough con. Tailing	71.81 28.19	25.54 85.66	92.51 7.49	308.73

Table A-6.24 Calculated Coefficients

t	E	F	C	F2	C2	CF	C3	F3	C2F	CF2
90	73.09	0.05	0	0.0025	0	0	0	0.00012	0	0
90	89.89	0.05	0.5	0.0025	0.25	0.025	0.125	0.00012	0.0125	0.00125
90	84.92	0.05	1	0.0025	1	0.05	1	0.00012	0.05	0.0025
90	125.45	0.05	2	0.0025	4	0.1	8	0.00012	0.2	0.005
90	124.58	0.1	0	0.01	0	0	0	0.001	0	0
90	188.6	0.1	0.5	0.01	0.25	0.05	0.125	0.001	0.025	0.005
90	216.63	0.1	1	0.01	1	0.1	1	0.001	0.1	0.01
90	187.46	0.1	2	0.01	4	0.2	8	0.001	0.4	0.02
90	183.16	0.2	0	0.04	0	0	0	0.008	0	0
90	188.65	0.2	0.5	0.04	0.25	0.1	0.125	0.008	0.05	0.02
90	203.9	0.2	1	0.04	1	0.2	1	0.008	0.2	0.04
90	189.53	0.2	2	0.04	4	0.4	8	0.008	0.8	0.08
90	180.34	0.3	0	0.09	0	0	0	0.027	0	0
90	197.32	0.3	0.5	0.09	0.25	0.15	0.125	0.027	0.075	0.045
90	198.57	0.3	1	0.09	1	0.3	1	0.027	0.3	0.09
90	194.87	0.3	2	0.09	4	0.6	8	0.027	1.2	0.18

Regression Output:

Constant	-132.
Std Err of Y Est	20.16
R Squared	0.927
No. of Observations	16
Degrees of Freedom	6
X Coefficient	4948.38
Std Err of Coefficient	100.7
	-25194.
	-47.39
	-261.
	8.78
	40552.5
	14.497
	384.914
	118.45
	1049.75

Table A-6.24 Calculated Coefficients

t	E	F	C	F2	D2	DF	C3	F3	D2F	DF2
120	83.02	0.05	0	0.0025	0	0	0	0.00012	0	0
120	103.82	0.05	0.5	0.0025	0.25	0.025	0.125	0.00012	0.0125	0.00125
120	98.65	0.05	1	0.0025	1	0.05	1	0.00012	0.05	0.0025
120	152.77	0.05	2	0.0025	4	0.1	8	0.00012	0.2	0.005
120	131.71	0.1	0	0.01	0	0	0	0.001	0	0
120	208.93	0.1	0.5	0.01	0.25	0.05	0.125	0.001	0.025	0.005
120	237.76	0.1	1	0.01	1	0.1	1	0.001	0.1	0.01
120	255.03	0.1	2	0.01	4	0.2	8	0.001	0.4	0.02
120	219.67	0.2	0	0.04	0	0	0	0.008	0	0
120	223.69	0.2	0.5	0.04	0.25	0.1	0.125	0.008	0.05	0.02
120	256.75	0.2	1	0.04	1	0.2	1	0.008	0.2	0.04
120	243.85	0.2	2	0.04	4	0.4	8	0.008	0.8	0.08
120	243.7	0.3	0	0.09	0	0	0	0.027	0	0
120	223.22	0.3	0.5	0.09	0.25	0.15	0.125	0.027	0.075	0.045
120	252.88	0.3	1	0.09	1	0.3	1	0.027	0.3	0.09
120	228.17	0.3	2	0.09	4	0.6	8	0.027	1.2	0.18

Regression Output:

Constant	-127.
Std Err of Y Est	24.40
R Squared	0.936
No. of Observations	16
Degrees of Freedom	6
X Coefficient	4999.16
Std Err of Co1387.12	48.79
	-24616.
	21.076
	82.14
	-12.78
	39994.6
	3.1580
	-813.01
	125.10
	543.3
	42.653
	16226.7
	143.35
	1270.43

Table A-6.24 Calculated Coefficients

t	E	F	C	F2	C2	DF	C3	F3	C2F	DF2
150	89.11	0.05	0	0.0025	0	0	0	0.00012	0	0
150	104.45	0.05	0.5	0.0025	0.25	0.025	0.125	0.00012	0.0125	0.00125
150	113.52	0.05	1	0.0025	1	0.05	1	0.00012	0.05	0.0025
150	163.91	0.05	2	0.0025	4	0.1	8	0.00012	0.2	0.005
150	132.86	0.1	0	0.01	0	0	0	0.001	0	0
150	210.49	0.1	0.5	0.01	0.25	0.05	0.125	0.001	0.025	0.005
150	238.03	0.1	1	0.01	1	0.1	1	0.001	0.1	0.01
150	262.54	0.1	2	0.01	4	0.2	8	0.001	0.4	0.02
150	233.3	0.2	0	0.04	0	0	0	0.008	0	0
150	244.66	0.2	0.5	0.04	0.25	0.1	0.125	0.008	0.05	0.02
150	282.78	0.2	1	0.04	1	0.2	1	0.008	0.2	0.04
150	279.22	0.2	2	0.04	4	0.4	8	0.008	0.8	0.08
150	258.1	0.3	0	0.09	0	0	0	0.027	0	0
150	278.58	0.3	0.5	0.09	0.25	0.15	0.125	0.027	0.075	0.045
150	299.72	0.3	1	0.09	1	0.3	1	0.027	0.3	0.09
150	295.31	0.3	2	0.09	4	0.6	8	0.027	1.2	0.18

Regression Output:

ConstRegression Output-96.0
Std Err of Y Est 20.29
R Squared 0.969
No. of Observations 16
Degrees of Freedom 6

X Coefficient4266.85 59.63 -19730. -1.253 161.3 -3.165 31440.1 -52.11 -499.24
Std Err of Co1153.93 82.89 7234.73 104.07 451.9 35.483 13498.8 119.25 1056.86

Table A-6.24 Calculated Coefficients

t	E	F	C	F2	D2	DF	C3	F3	D2F	CF2
180	99.4	0.05	0	0.0025	0	0	0.00012	0	0	0
180	108.38	0.05	0.5	0.0025	0.25	0.025	0.125	0.00012	0.0125	0.00125
180	133.05	0.05	1	0.0025	1	0.05	1	0.00012	0.05	0.0025
180	177.41	0.05	2	0.0025	4	0.1	8	0.00012	0.2	0.005
180	139.9	0.1	0	0.01	0	0	0	0.001	0	0
180	211.47	0.1	0.5	0.01	0.25	0.05	0.125	0.001	0.025	0.005
180	254.22	0.1	1	0.01	1	0.1	1	0.001	0.1	0.01
180	283.93	0.1	2	0.01	4	0.2	8	0.001	0.4	0.02
180	238.27	0.2	0	0.04	0	0	0	0.008	0	0
180	260.02	0.2	0.5	0.04	0.25	0.1	0.125	0.008	0.05	0.02
180	306.41	0.2	1	0.04	1	0.2	1	0.008	0.2	0.04
180	302.45	0.2	2	0.04	4	0.4	8	0.008	0.8	0.08
180	264.26	0.3	0	0.09	0	0	0	0.027	0	0
180	303.96	0.3	0.5	0.09	0.25	0.15	0.125	0.027	0.075	0.045
180	330.51	0.3	1	0.09	1	0.3	1	0.027	0.3	0.09
180	308.73	0.3	2	0.09	4	0.6	8	0.027	1.2	0.18

Regression Output:

Constant -72.9
Std Err of Y Est 18.33
R Squared 0.977
No. of Observations 16
Degrees of Freedom 6

X Coefficient 3941.31 16.85 -18108. 49.775 486.7 -18.95 29129.3 -136.3 -955.15
Std Err of Co 1042.38 74.87 5535.34 94.013 408.2 32.052 12193.9 107.72 954.693

Table A-6.24 Calculated Coefficients

[illegible]

a1	a2	a3
Regression Output:	Regression Output:	Regression Output:
Constant	Constant	Constant
-117.60	4440.45	208.469
Std Err of Y Est	Std Err of Y Est	Std Err of Y Est
7.49172	266.060	26.0252
R Squared	R Squared	R Squared
0.97591	0.91267	0.81177
No. of Observations	No. of Observations	No. of Observations
4	4	4
Degrees of Freedom	Degrees of Freedom	Degrees of Freedom
1	1	1
X Coefficient(s)	X Coefficient	X Coefficient(s)
0.004	-0.104	0.00254
Std Err of Coef.	Std Err of Coef.	Std Err of Coef.
0.004	40.1478	0.01445
	40.1057	3.92302

Table A-6.24 Calculated Coefficients

a4

Regression Output:
 Constant -28715.
 Std Err of Y Est 1693.09
 R Squared 0.92318
 No. of Observations 4
 Degrees of Freedom 1

X Coefficient(s) 0.290 8.79165
 Std Err of Coef. 0.940 255.214

a7

Regression Output:
 Constant 54.0210
 Std Err of Y Est 12.6557
 R Squared 0.63536
 No. of Observations 4
 Degrees of Freedom 1

X Coefficient(s) 0.001 -0.6785
 Std Err of Coef. 0.007 1.90771

a10

Regression Output:
 Constant 4721.68
 Std Err of Y Est 510.134
 R Squared 0.76009
 No. of Observations 4
 Degrees of Freedom 1

X Coefficient(s) 0.206 -68.006
 Std Err of Coef. 0.283 76.0971

a5

Regression Output:
 Constant -198.4
 Std Err of Y Est 36.706
 R Squared 0.7329
 No. of Observations 4
 Degrees of Freedom 1

X Coefficient -0.0048 2.2051
 Std Err of Coef. 0.02039 5.5330

a8

Regression Output:
 Constant 46223.
 Std Err of Y Est 3184.2
 R Squared 0.9011
 No. of Observations 4
 Degrees of Freedom 1

X Coefficient -0.4869 -11.27
 Std Err of Coef. 0.76901 479.98

a9

Regression Output:
 Constant -160.56
 Std Err of Y Est 3.33845
 R Squared 0.99921
 No. of Observations 4
 Degrees of Freedom 1

X Coefficient(s) -0.0202 3.77665
 Std Err of Coef. 0.00185 0.50323

a6

Regression Output:
 Constant -1014.8
 Std Err of Y Est 114.213
 R Squared 0.95393
 No. of Observations 4
 Degrees of Freedom 1

X Coefficient(s) -0.0050 9.10804
 Std Err of Coef. 0.06345 17.2164

Table A-6.25 Coefficients of any flotation time

```

10 PRINT "90 sn          120 sn          150 sn          180 sn"
20 PRINT "-----"
30 FOR T= 90 TO 180 STEP 30
40 A1= -117.6+ .004*T^2- .6253 *T
50 A2= 4440- .105*T^2+ 15.7128*T
60 A3= 208.469+ .00254*T^2-1.4909*T
70 A4= -28715+ .29021*T^2+8.79165*T
80 A5= -198.4-.0048*T^2+2.20516*T
90 A6= -1014.8- .005*T^2+ 9.108*T
100 A7= 54.021+ .0016*T^2- .6785*T
110 A8= 46223.2- .4869*T^2- 11.271*T
120 A9= -160.56- .0202*T^2+ 3.77665*T
130 A10= 4721.88+ .206111*T^2- 68.006*T
140 PRINT A1;A2;A3;A4;A5;A6;A7;A8;A9;A10
150 NEXT T
    
```

Coefficients	FLOTATION TIMES			
	90	120	150	180
a1	-141.77	-135.036	-121.385	-100.554
a2	5003.652	4813.536	4434.42	3366.304
a3	94.862	66.137	41.9839	22.402
a4	-25573.05	-23480.98	-20866.53	-17729.70
a5	-38.8156	-2.9007	24.3539	43.008
a6	-235.580	6.1599	238.899	2162.639
a7	5.916	-4.3590	-11.7540	-16.269
a8	41264.92	37859.31	33577.30	28418.86
a9	15.718	1.758	-48.5624	-135.5243
a10	270.839	-470.8418	-841.5224	-841.203

Table A-6.26 Finding max. value of E and C at 0.2 kg/t frother level

```

10 C=1
20 FOR I = 1 TO 10
30 X= 56.1+2*14.658*C-3*11.75*C^2
40 Y= 2*14.658-6*11.75*C
50 C= C-(X/Y)
60 PRINT "C150=";C
70 NEXT I

```

```

C150= 2.218094
C150= 1.806459
C150= 1.745535
C150= 1.744139
C150= 1.744139
C150= 1.744139
C150= 1.744139
C150= 1.744139
C150= 1.744139
C150= 1.744139

```

Table A-6.27 Finding the optimum C and F values at flotation time of 150 sec

```

10 C=1
20 F=.1
30 FOR I= 1 TO 15
40 X= 41.98+2*24.37*C+238.9*F-3*11.75*C^2-2*48.56*C*F-841.52*F^2
50 Y= 2*24.73-6*11.75*C-2*48.56*F
60 M= 4434.42-2*20866.53*F+238.9*C+3*33577.3*F^2-48.56*C^2-2*841.52*C*F
70 N= -2*20866.53+6*33577.3*F-2*841.52*C
80 C= C- (X/Y)
90 F=F-(M/N)
100 PRINT "C=";C ;"F=";F
110 NEXT I

```

```

C= 2.991181 F= .155457
C= 2.107544 F= .1475989
C= 1.885852 F= .1626148
C= 1.838961 F= .1686592
C= 1.824833 F= .1699744
C= 1.821796 F= .170292
C= 1.82107 F= .170358
C= 1.82092 F= .1703737
C= 1.820884 F= .1703769
C= 1.820877 F= .1703776
C= 1.820875 F= .1703778
C= 1.820875 F= .1703778
C= 1.820875 F= .1703778
C= 1.820875 F= .1703778
C= 1.820875 F= .1703778

```

Table A-6.28a Comparison of calculated and actual efficiency index values

Collector Kg/t C	Frother Kg/t F	90 seconds			180 seconds		
		Efficiency Index E	Calculated Efficiency Index	Absolute Relative Error(%)	Efficiency Index E	Calculated Efficiency Index	Absolute Relative Error(%)
0	0.05	73.03	61.93	11.16	99.40	81.98	17.52
0.5	0.05	89.89	83.04	7.61	108.36	80.73	25.51
1	0.05	84.81	101.57	19.75	133.05	115.39	13.27
2	0.05	125.45	112.67	10.18	177.41	153.67	13.38
0	0.1	124.58	144.42	15.59	139.90	137.19	1.93
0.5	0.1	186.60	172.85	8.34	211.47	172.66	18.35
1	0.1	216.63	187.10	13.62	254.32	210.66	17.13
2	0.1	187.43	190.81	1.78	283.03	245.47	13.26
0	0.2	183.16	166.44	9.12	138.27	190.86	19.89
0.5	0.2	168.65	167.55	0.57	260.02	233.45	10.21
1	0.2	203.50	195.27	4.23	306.11	271.83	5.07
2	0.2	189.53	188.26	0.67	302.45	287.10	5.17
0	0.3	180.34	172.18	4.52	264.26	230.95	12.60
0.5	0.3	197.32	188.68	4.37	303.96	272.26	10.42
1	0.3	198.87	192.56	3.17	330.51	302.59	8.44
2	0.3	194.82	180.25	7.50	308.73	281.45	8.82
		Standard deviation=7.66 (%)			Standard deviation=12.94 (%)		

Table A-6.28b Comparison of calculated and actual efficiency index values

Collector Kg/t C	Frother Kg/t F	120 seconds			150 seconds		
		Efficiency Index E	Calculated Efficiency Index	Absolute Relative Error(%)	Efficiency Index E	Calculated Efficiency Index	Absolute Relative Error(%)
0	0.05	83.02	71.67	13.65	89.11	75.78	14.95
0.5	0.05	103.82	83.05	20.00	104.45	81.74	21.37
1	0.05	98.65	109.75	11.26	113.52	113.83	0.27
2	0.05	152.77	136.06	10.93	163.91	149.22	8.96
0	0.1	131.71	149.36	13.40	132.86	146.42	10.20
0.5	0.1	208.93	179.10	14.24	210.49	178.56	15.16
1	0.1	237.76	204.32	14.06	238.03	211.64	11.08
2	0.1	255.03	227.66	10.73	262.54	245.39	6.53
0	0.2	215.67	191.30	18.91	233.30	198.93	14.73
0.5	0.2	223.69	214.38	4.18	244.66	229.17	6.33
1	0.2	256.75	232.92	9.27	282.70	257.93	8.78
2	0.2	243.85	243.28	0.23	279.22	275.76	1.24
0	0.3	243.70	217.93	10.57	258.10	237.04	8.16
0.5	0.3	223.70	229.59	2.63	278.58	256.98	7.75
1	0.3	252.78	236.80	6.36	299.72	273.00	8.91
2	0.3	228.17	224.76	9.74	295.31	258.07	12.60
		Standard deviation=9.47 (%)			Standard deviation=9.84 (%)		

Table A-6.29 Analysis of concentrate obtained using with ethanol

ANALYSIS TYPE	DRY
ULTIMATE ANALYSIS	
Ash (%) :	9.61
Volatile Matter (%) :	28.46
Fixed Carbon (%) :	61.93
TOTAL	100.00
HEATING VALUE (Kcal/Kg)	
Gross heating value :	7805
Net heating value :	7567

Table A-6.30 Analysis of concentrate obtained using with octanol

ANALYSIS TYPE	DRY
ULTIMATE ANALYSIS	
Ash (%) :	22.67
Volatile Matter (%) :	28.03
Fixed Carbon (%) :	49.30
TOTAL	100.00
HEATING VALUE (Kcal/Kg)	
Gross heating value :	7024
Net heating value :	6821