COAL PREPARATION

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The first principle of coal preparation is the determination of the properties of the material to be treated. Information may be obtained by examination of bore cores; from seam exposures in the case of an operating or developed mine; or by sampling the run - of-mine productio n. Details to be observed include full information on the constitution of the coal, variations which may be anticipated from different sections of the seam, the associated strata, both above and below the seam, cleat, the probable moisture content and the particle size range of the mined coal.

The hardness of the coal, which generally decreases from anthracite to lignite, can be determined by Vickers hardness diamond - indentation unit. Compression tests may be made either uniaxially or triaxially to study the strength of coal. Friability which is associated with strength and influences degradation, is less important now that the market for the large coal has declined.

Abrasiveness influences the rate of wear on screens, launders, pumps and other units. It generally is affected by the mineral impurities present. Many such minerals observed in coals, possibly the most important being muscovite, illite, montmorillonite, kaolinite, pyrite, marcasite, calcite, siderite and ankerite. It is important to ensure that what is removed actually improves the coal properties. Sometimes partial cleaning such as the removal of corrosion inhibitors, but leaving corrosion promotors, can be detrimental. The degree of clay and shale breakdown is associated with the rank of the coal. As rank increases, the tendency to compaction of these impurities increases.

Much can be done to aid the work of the preparation engineer by practising selective mining. This involves blending where possible, minimum degradation, moisture control, avoidance of material which will disintegrate when wetted and reducing to a minimum, extraneous material. In open pit mining, it may be possible to remove some impurities.

The method of mining, which is the first stage of comminution, will influence the particle size distribution. The degree of mixing which takes place during excavation, loading, transport and delivery is of major importance in wet conditions when clay may become attached to coal particles and cause water clarification problems.

A detailed petrographic analysis of the coal will normally make it possible to set the parameters for the way in which the total coal of the seam or portions thereof, may be beneficiated and utilised. Once the composition of the coal and the required product qualities are known, a preliminary process selection may be made.

Particle Breakage: The objectives of particle breakage are to free the coal from the compact seam, to reduce it to a size suitable for transport, to liberate the various components, produce particle sizes suitable for handling in the preparation processes and produce material to meet the market requirements. The energy requirements for size reduction are influenced by the physical properties of the coal constituents, cleat, joints, cleavage and stresses induced during mining.

Liberation studies will indicate the percentage of impurities which can be freed and the distribution of the coal constituents in the liberated particles, so that the potential of a coal to be cleaned, the possible method of beneficiation and the product grades can be assessed.

Normally it is advantageous to retain the coal in as large a particle size as possible since preparation and subsequent moisture contents of products generally increase as particle size decreases. The common crushers employed in size reduction are Bradford breakers, single and double roll crushers, gyratory crushers and impact crushers. In selection of a suitable unit consideration must be given to reduction ratios; maximum particle size of the feed; product size required for preparation; incidence of rock and uncrushable material; case of varying set of crusher and the possibility of using a closed circuit.

<u>Testing of coal</u>: For coals having particle size greater than 0.5 mm, the principle of separation is ~~a~~ difference in relative densities of the material to be separated and test work is based upon the sink-float analysis. For coals having particle size less than 0.5 mm, separation may be by froth flotation. For coal preparation, the most important data is that obtained sieve analysis, float and sink tests, froth flotation tests and work on flocculation and sedimentation

After sieve analysis, each size fraction is subjected to float-sink testing using liquids of specific gravity ranging from 1.3 to 2.0. The liquids commonly used for this testing are inorganic salt solutions and organic liquids. The commonly used organic liquids are

(i) toluene (sp.gr. 0.87)
(ii) carbon tetrachloride (sp.gr. 1.60)
(iii) perchlorethylene (sp.gr. 1.60)
(iv) bromoform (sp.gr. 2.79)

Since the organic liquids are miscible with each other, any intermediate specific gravity liquid can be obtained by mixing two liquids. Float and sink tests involve subjecting a coal sample in the laboratory to liquids of successively greater specific gravities, collecting the float fraction at each stage and the final sink fraction for analysis.

A typical set of washability data is shown in Table 1. This information may then be presented graphically in the form of washability curves (Fig. 1) which indicate the ease or difficulty of the proposed separation and also indicate the conditions for the process. The curves usually presented are

Cumulative floats curve: Obtained by plotting cumulative weight percent of floats at each relative density increment against the cumulative ash at that point (columns 4 and 5 of Table 1). Both scales are usually arithmetic although a logarithmic scale may be used for the ash to accentuate differences in the lower ash ranges. The curve may be used to indicate the yield obtainable for any set ash required.

Cumulative sinks curve: This curve is obtained by plotting the the cumulative weight percent of sinks at each relative density increment against the cumulative ash of the sinks for that separation (columns 6 and 7). Obviously the ash point for one hundred percent sinks must equal the ash point for one hundred percent of floats. This curve indicates the ash content of the rejects when a certain yield of clean coal is required.

Instantaneous ash curve: This curve is sometimes referred to as the elementary ash curve, characteristic ash curve or observed curve and describes the rate of change of ash at different yields (colums 8 and 9). Instantaneous ash curve gives the maximum ash in any particle in the float and minimum ash in any particle in the sinks.

Yield-gravity curve: This is obtained by plotting the cumulative percent of floats against the relative density for that separation (columns 1 and 4). It indicates the yield of clean coal for a perfect separation at a selected relative density

The distribution curve: An interval, usually $\pm$ 0.1 for relative density is specified and then the difference in yield between two relative densities, 0.2 apart, is plotted against the mean of those densities. This curve indicates the difficulty of separation. A low value, less than 10%, is satisfactory, whereas a figure in excess of 20% indicates a very difficult operation.

The combined curves can be used to indicate the parameters of separation. For example, in Fig.1, for a yield of 75 percent the clean coal would contain 15.2% ash; rejects 64% ash. The separation would be made at 1.575 relative density and the separation would be satisfactory (near-gravity material 16%)

Evaluation of Performance:

The data from float and sink analyses indicate what should be obtained under ideal conditions of operations. Such conditions do not exist in plant practice, some material being misplaced; two important reasons being degradation of coal during processing and the imperfection of the separatory unit. Useful information on degradation can be obtained by dynamic testing or pre-treatment of coal with water.

The partition or distribution curve is useful in assessing the sharpness of separation, or to predict the performance of a plant. Normally it is relatively independent of the float and sink properties of the coal being dependent on particle size distribution and the type of separating machine. Data from float and sink analyses on the raw coal, the clean coal and the reject is used to determine the partition coefficients.

$$\text{Partition coefficient} = \frac{\text{Weight of raw coal reporting to clean coal in any R.D. range}}{\text{Weight of raw coal present in that R.D. range in the feed}} \times 100$$

The yield of clean coal is calculated from the ash contents of feed, clean coal and rejects. Table 2 shows the calculation of partition coefficients from the sink-float analysis of clean coal and rejects and the yield of clean coal. The partition curve is obtained by plotting the partition coefficient against the mean density range (Fig.2). From the partition curve the cut point (separation density d_{50}) is determined. The gradient of the curve is a measure of the sharpness of separation and to indicate the accuracy of the separation, the slope of the partition curve between 75 and 25% partition coefficients is used.

This difference divided by 2 is referred to as the probable error (E_p).

$$E_p = \frac{d_{25} - d_{75}}{2}$$

The probable error is relatively characteristic of a process, units which have a low E_p such as heavy medium processes (0.02 to 0.05) being considered efficient separators. The probable error depends upon relative densities and therefore, the imperfection is used as a further method of comparing separatory processes.

By definition : $\text{Imperfection} = \dfrac{\text{Probable error}}{\text{Partition density-1}}$

$$\text{or} \quad I = \frac{E_p}{d_p - 1}$$

Table 2 gives the data for plotting partition curve using the sink-float analysis of clean coal and rejects.

Processes:

The preparation methods employed depend upon the size of coal being treated.

Size range	Method
Coarse	Heavy media jigging
Intermediate	Flowing film heavy media cyclone jigging
Fines	Froth flotation water-only cyclones.

Jigging: In terms of world tonnages of coal treated, this is the most important process. A jig consists of a suitable compartment within which the material to be stratified is supported on a sieve. A mechanism is installed so that water may be caused to palsate through the sieve mobilising the material to be separated. Pulsion is obtained by water admission, air pressure changes, reciprocating plunger or oscillation of the sieve supporting the feed. It is important that the bed becomes completely mobile as rapidly as possible and that the water velocity is sufficient to levitate all the particles in the bed. After pulsion the water flow is decelerated, the bed loses its mobility and suction takes place. The full cycle of pulsion-suction is a function of the rate of acceleration of the particles in the fluid as they attain their terminal velocity, hence particle size, shape and density are important. Normally pulsion flow equals suction flow but additional water may be added so that suction can be controlled.

The three phenomena on which stratification depends are differential acceleration, hindered settling and consolidation trickling.

For large coal, the separation is usually made with bed depths of coal upto 0.75m jigging on the sieve being practised with the stratified coal being displaced over a weir. Reject material is discharged at the end of the sieve bed by a controlled ejector. For smaller sized coal, less than 18 mm, jigging through the sieve may be practised. To prevent short circuiting, bed protection or ragging is required and this is usually composed of feldspar or material of similar density having a particle size of 50 - 75 mm. In either unit, bucket elevators combined with screws moving in the bottom of the jig hutch are used continuously to recover the sinks material from the jig. Jigs operate most satisfactorily when the difference in the differential acceleration of the particles to be separated is greatest and this is a function of the difference between their density and that of the fluid being used. Hence close particles size ra nge of feed is preferable. Also since removal of the products must be maintained at a set rate to ensure that bed depths are relatively consistent, thus avoiding changes in the resistance offered to fluid flow during pulsion, feed grades and rates should be controlled. This is not always practicable and float controllers or similar devices are used to control the depth of reject material on the sieve. Jigs usually have a high capacity and to avoid uneven distribution of the pulsion water through the sieve the overall jig is normally divided into compartments, each unit having its own pulsion mechanism. Water requirements for jigging are of the order of 2.3 to 5 m^3 per tonne with 14-25 l.p.m. per tonne per hour. This water becomes contaninated with particulate material and must be-circulated through a cleaning process of flocculation and thickening.

Heavy Media Separation:

The principle of heavy media separation is based upon float and sink. If a particle has a density less than that of the fluid it will float,if greater it will sink. The separation is quite sharp since no other factors are involved. However, because the rate at which a particle sinks or alternatively, in case of buoyancy, the rate at which it rises to the surface is a function of the size of that particles, for practical reasons the process is limited to coarse particles. Unit capacity would be too low for fine material. The fluid can be controlled at any appropriate relative density so a wide range of separations can be made. Also since the physical division between the particles is determined by the shape and size of the vessel, an accurate out point with a minimum of intermingling is possible. Provided the unit can physically handle the products (i.e., is not overloaded) it is capable of adjusting the fluctuations in feed grade and feed rate without materially effecting its capacity.

The separatory vessels have been designed with various shapes and a number of methods of product removal are in use. Factors of importance are depth of the unit, method of feed and media entry, whether feed is forcibly immersed, number of products, weir drag, upwards currents, residence time and the ratio of media flow to feed rate. Normally the feed is pretreated by sizing to remove fines. Products are drained and washed to remove medium which is recovered for re-use in a special circuit.

Four types of mediam are available, organic liquids, dissolved salts, aerated solids and suspensions in water. The suspension of solids in water, or pseudo liquids are most commonly used. Selection and application of the solid is the most important feature of the process. The solid selected must be of low cost, not decompose or degrade in water, be chemically inert with respect to its use in the process, not react with the coal, yield a liquid of satisfactorily low viscosity, be stable and have some unique property so that it can be recovered for re-use. Water requirements for heavy media separation vary from 0.05 to 2.5 m^3 per tonne with 30-50 l.p.m. per tph.

For coal preparation the densities of separation range from 1.30 to 1.90 and to manitani a volumetric concentration of between 25 to 50 percent, a material having a relative density of the order of 4.0 to 5.0 is required. Magnetite is the most commonly used material since it also has the property of being recoverable by its magnetic properties. Its density is of the order of 5.0 and the following formula may be used to calculate the R.D. of a suspension:

$$\text{R.D. of suspension} = \frac{100}{(100-M) + \frac{M}{D}}$$

where M is the percentage of magnetite by mass, and

D is the density of magnetite.

A suspension of 40 percent magnetite (D= 4.8) would yield a pulp having a separating density of 1.46. The medium must not settle rapidly, a stable low viscosity suspension being required. For magnetite, the percentage of minus 44 micron is of the order of 92 to 95% for R.D. 1.35 to 1.50 decreasing to 60-65% for R.D. 1.50 to 1.80. The susceptibility should be within the range 0.050 to 0.070 e.m.a. per gram in a field 800 oersteds. The saturation moment required is between 60 and 90 e.m.a. depending upon the purity of the magnetite (maximum 92 e.m.a.)

Magnetite separators are usually of the drum type, about 0.75m in diameter, have field strength in excess of 700 gauss measured some 50 mm from the drum surface. The feed is normally 6 to 20% solids, at a feed rate of 120 litres per minute per 0.1 m width representing a capacity of 1 tonne per 0.1 m width per hour. Secondary circuits and even tertiary units may be used.

Heavy Media Cyclones:

Cyclones were initially used as thickeners, later as classifiers and more recently introduced as heavy media units.

In a static bath the separating force is proportional to the gravitational force whereas in a cyclone it is proportional to the centrifugal force. This can be increased by increasing the pressure or velocity of flow or decreasing the radius of the cyclone. In the average cyclone the centrifugal force is some 20 times greater than the gravitational force in the cylindrical portion of the cyclone, increasingly higher in the conical section and possibly 200 times at the apex. Consequently, in the cyclone the separation of smaller particles becomes possible, it has a high capacity. Because of the increasingly high centrifugal force near the apex the actual separation density is higher than the true density of the media, that is, the relative density of separation is greater than the relative of the suspension.

The operation is different from heavy media separator in that the solid coal is mixed with the media and then pumped or allowed to flow by gravity to the separatory unit. Magnetite is commonly used as medium and it is required 90% minus 44 micorn, but care must be exercised since ultrafine magnetite may be difficult to recover. The closely sized magnetite is not classified in the cyclone, however, it will be concentrated in the underflow. Because viscosity is less important, fines can be higher and impurities greater in proportion. The sharpness of separation (i.e., probable error) is normally very satisfactory.

Normally, if D represents the diameter of the cylindrical portion, then the ratios of other sections are approximately, the inlet 0.2 D, the vortex finder 0.4 D, length of cylindrical portion 1.5 D, the underflow 0.3 D with a cone angle of the order of 20°.

The separation is influenced by the length of the vortex finder, apex diameter, relative density of the media, particle size distribution of the media, cyclone diameter and pressure. Feed particle size is within the limits of 50 mm to 0.3 mm.

The inlet pressure affects the volumetric capacity and efficiency of separation. There is a limit on pressure governed by frictional forces developed in the unit. The normal pressure ranges from 8 to 10 psi or a head of 9 times the diameter of the cyclone. The ratio of medium to coal is in the order of 5 to 1. The units are commonly arranged at angles of 10 to 15 to the horizontal to assist drainage

Capacities of heavy media cyclones are of the order of; 200 mm, 7 tph; 510 mm, 50 tph; 580 mm, 75 tph and 710 mm, 100 tph, with water requirements of 95 - 140 l.p.m. per tph.

Other examples of this principle of separation are the Dyna Whirlpool and the Vorsyl separators.

Water-only Cyclone:

The autogenous and water cyclone have been used for preparation of coal generally in the minus 6 mm particle size range. The actual unit differs from the dense media cyclones in that the vortex finder is normally longer and the cone angle greater between 60 and 135°. Separation is based upon acceleration in the fluid which is a function of the relative density of the particle so that the more dense particle moves out and is discharged via the spigot. With particle size decrease, fluid resistance becomes more dominant and separation less effective. Important controls are the length of the vortex finder and the diameter of the apex discharge orifice.

Froth Flotation:

Unlike the other methods of coal preparation froth flotation is dependent upon the surface characteristics of the coal in terms of their reaction to certain modifying agents and air. The actual separation is based upon a float-sink operation. However, this is controlled by the levitating effect of the air as attached to some particles. This reaction effect is controlled by the surface of the particle, which, due to chemical changes such as oxidation or attachment of particulate material, may have properties different from the particle as a whole. Normally, material composed of hydrocarbons, containing carbon or sulphur and crystalline has a high floatability whereas high in moisture, nitrogen or oxygen does not.

Preliminary investigations to assess surface properties, will indicate the potentialities of froth flotation and what reagents will be required. Such tests involve surface examination by microscope or scanning electron microscope, contact angle measurement, bubble pick up tests, Hallimond tube tests or similar small scale floats. The information will indicate the needs for desliming, detergents, addition of protective colloids or dispersants, such as sodium silicate, depressants, pH control and the quantities of collectors such as kerosene and appropriate frothers which are normally organic and heteropolar

Confirmatory tests may be made in small scale laboratory flotation cells (250 to 500 gram capacity) or in pilot plants where flotation kinetic data is obtained as necessary for plant design. Work hss shown that for any coal there is an optimum particle size range, generally within the range of 200 to 50 micron outside which percentage recovery and rate of recovery decrease.

While a variety of flotation cell types is available industry has principally used mechanical sub-aeration cells, al although vacuum cells and pneumatic cells have been used. Because of the relatively high rate of flotation, low density of the components, and the need to use one circuit only without benefit of recleaning cycles, low pulp densities, of the order of 5 to 15 percent solids, are used. Banks of cells in series, usually less than 5 are employed. Water requirements are 50 to 75 l.p.m. per tph.

Conclusion:

In addition to the above methods, in abroad, flowing film concentration using tables are also employed for coals below 6 mm. The preparation method and the type of circuit depend upon the feed coal washability characteristics.

Table 1: Float and sink data for washability assessment

Relative density	Weight %	Ash %	Cumulative floats Weight%	Cumulative floats Ash%	Cumulative sinks Weight%	Cumulative sinks Ash%	Instantaneous ash Weight%	Instantaneous ash Ash%
					100.0	27.6		
Floats 1.30	9.5	5.0	9.5	5.0	90.5	30.0	4.7	5.0
1.30-1.35	9.7	8.0	19.2	6.5	80.8	32.6	14.3	8.0
1.35-1.40	13.4	10.1	32.6	8.0	67.4	37.1	25.9	10.1
1.40-1.45	17.8	16.8	50.4	11.1	49.6	44.4	41.5	16.8
1.45-1.50	17.2	22.5	67.6	14.0	32.4	56.0	58.9	22.5
1.50-1.55	6.0	25.9	73.6	15.0	26.4	62.9	70.6	25.9
1.55-1.60	3.4	35.2	77.0	15.9	23.0	67.0	75.3	35.2
1.60-1.70	3.3	40.0	80.3	16.9	19.7	71.5	78.7	40.0
1.70-1.80	2.8	46.4	83.1	17.9	16.9	75.6	81.7	46.4
1.80-1.90	1.5	51.5	84.6	18.4	15.4	78.0	83.8	51.5
1.90-2.00	2.0	65.9	86.6	19.5	13.4	79.8	85.6	65.9
Sinks 2.00	13.4	79.8	100.0	27.6			93.3	79.8

Table 2: Plant performance data

Clean coal ash 13.8% Feed ash 27.6% Rejects ash 48.65

Yield of clean coal $\frac{48.65 - 27.6}{48.65 - 13.8} \times 100$ 60.4%

Relative density	Clean coal Mass%	Rejects Mass%	Cleans x 0.604	Rejects x 0.396	Feed Mass%	Partition coefficient%
Floats 1.30	15.7	0	9.5	0	9.5	100.0
1.30-1.35	15.9	0.3	9.6	0.1	9.7	99.0
1.35-1.40	21.0	1.8	12.7	0.7	13.4	94.8
1.40-1.45	24.2	8.1	14.6	3.2	17.8	82.0
1.45-1.50	17.9	16.1	10.8	6.4	17.2	62.8
1.50-1.55	3.8	9.3	2.3	3.7	6.0	38.3
1.55-1.60	1.2	6.8	0.7	2.7	3.4	20.6
1.60-1.70	0.3	7.8	0.2	3.1	3.3	6.1
1.70-1.80	0	7.1	0	2.8	2.8	0
1.80-1.90	0	3.8	0	1.5	1.5	0
1.90-2.00	0	5.1	0	2.0	2.0	0
Sinks 2.00	0	33.8	0	13.4	13.4	0

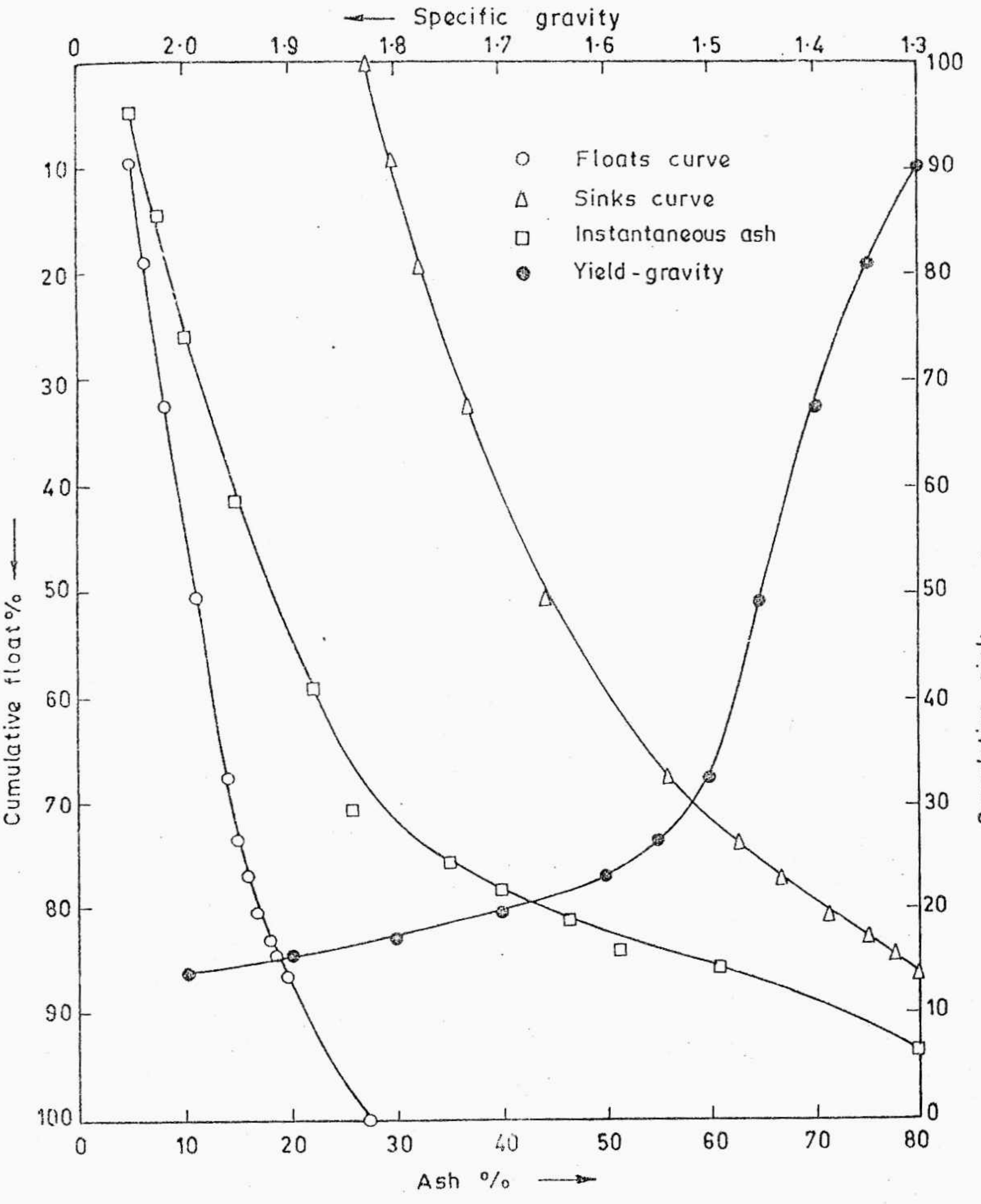

Fig.1 Washability curves

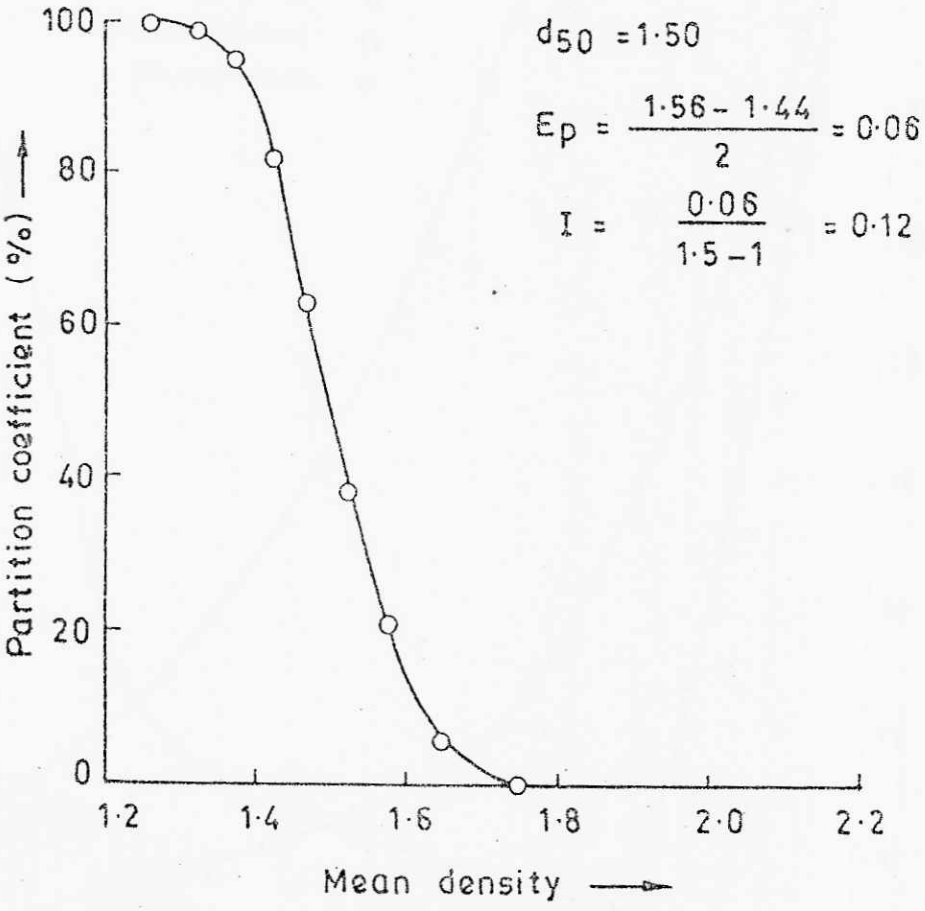

Fig. 2 Partition curve