COAL CLEANING WITH SPECIAL REFERENCE TO THE DEVELOPMENT AND APPLICATION OF THE CYCLONE WASHER.

BY

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Thesis presented to the Mechanical Engineering Department of the University of Cape Town for the degree of Doctor of Philosophy.

June, 1950.

P.O. Box 217,
Pretoria,
8th March 1951

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float and sink and relevant data" appeared in the Journal of the Chemical Metallurgical and Mining Society of South Africa, Vol 50 N°3 September 1949

(b) The whole of Part 2 entitled "A study of the operation of the cyclone washer and its application to Witbank fine coal" appeared in the Journal of the Chem. Met. and Min. Soc. of SA Vol 51, N°2, August 1950.

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COAL CLEANING WITH SPECIAL REFERENCE TO THE DEVELOPMENT AND APPLICATION OF THE CYCLONE WASHER.

SUMMARY.

The paper consists of two parts, the first being an outline of the theory and practice of coal cleaning. Various conventional processes are described and the fundamental principles upon which they depend for their operation are explained. The methods used for assessing the performance of a coal washer are compared and a new graphical representation of washery data is suggested.

In the second part of the paper, a particular South African fine coal cleaning problem is examined :- Laboratory tests have indicated that the fine coal produced by a number of collieries situated in the Witbank district contains a reasonably high proportion of coking coal and that it may be possible to recover this by washing at 1.35 to 1.4 specific gravity.

This separation represents a formidable washing problem and it was evident that the conventional processes would be unsuitable. Various newly developed fine coal washers described in the literature were consequently studied and the cyclone washer was finally selected as being the most promising.

As very little technical data were available on this type of washer, it was necessary to determine the main operating characteristics experimentally. During this phase of the investigation, emphasis was placed on obtaining maximum capacity and efficiency for separations in the specific gravity range mentioned.

Having established the operating adjustments best suited to the separations in view, cyclone washing tests were carried out on three typical Witbank duff coal samples in order to determine the suitability of the process. The washed products obtained were subjected to coking tests in an experimental coke oven.

The results of these tests indicate that it should be possible to recover the coking fraction from Witbank fine coal on a commercial scale by using the cyclone washer and that metallurgical coke can be produced from the coking coal obtained in this way.

There has been a general tendency during latter years for the demand for large coal to decrease and for smaller coal to increase in popularity. This is largely due to the development of mechanical stoking equipment and the fact that engineers began to realise that the smaller sizes can be burned more efficiently. Since the demand for smaller coal, such as nut and pea sizes, continues to exceed the quantity available as "natural arisings", it appears reasonable to suppose that recourse will have to be had to crushing of large coal in due course. In the initial stages this might take the form of reducing the top size to a limit of, say, 6 to 8 inches, or of crushing only a proportion of the rounds. In either event, it follows that the amount of fines produced will inevitably increase and the utilisation of duff will have to receive more serious consideration. One avenue worth investigating appears to be to enhance the value of this duff by cleaning it and thus to render it more attractive to the consumer.

Now, it is in the national interest to conserve the reserves by utilising all types and grades of coal in the most efficient manner. Since this includes efficient preparation at the colliery, the Fuel Research Institute decided to make a detailed study of coal cleaning methods, with special reference to those applicable to fine coal. In this way, it was hoped, the Institute would be in a position to assist collieries to operate existing washers most efficiently and to advise on future washing problems.

As a first step in this direction, a careful study was made of the available literature and a condensed summary was prepared in order to stimulate interest in coal cleaning. This outline of the theory and practice of coal cleaning constitutes Part 1 of the present paper. Since the various methods proposed for assessing the performance of a washer appeared inadequate in many respects, the author paid particular attention to this aspect of the subject and worked out a new graphical representation of washery data which may assist in clarifying the problem.

The possibility of recovering the coking fraction known to be present in some of the duff coal produced in the Witbank area on a commercial scale was the first coal washing problem to be investigated. The proved reserves of coking coal in South Africa are comparatively small and their rapid depletion is causing concern. Any possibility of augmenting these reserves consequently justifies close consideration.

The separation involved represents a formidable washing problem and it was clear that the conventional processes would be unsuitable. Various newly-developed fine coal washers described in the literature were, therefore, studied and the cyclone washer was finally selected as holding out most promise. Since little technical data were available on this washer, it was necessary to determine its main operating characteristics experimentally before washing tests could be carried out to prove the suitability of the process. This work is described in detail in Part 2 of this paper. The salient points of Part 1 which have a bearing on the problem of washing Witbank duff have been briefly summarised again in Part 2 in order that this section of the paper may be complete in itself.

PART 1.

AN OUTLINE OF THE THEORY AND PRACTICE OF COAL CLEANING.

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A N   O U T L I N E   O F   T H E   T H E O R Y   A N D   P R A C T I C E   O F   C O A L   C L E A N I N G.

C O N T E N T S.

PART 1.

AN OUTLINE OF THE THEORY AND PRACTICE OF COAL CLEANING.

COAL AND ITS MAJOR IMPURITIES.[(1)(2)(3)]

The Origin of Coal.

It is generally accepted that coal is of vegetable origin and that it has been produced almost exclusively from terrestrial plants, which, it is thought, flourished in dense swamps. The debris from these plants accumulated under water logged conditions and was transformed into peat, largely by bacterial action. During this process, the vegetable matter lost moisture, carbon dioxide was evolved and acidic or humic substances were formed. The peat bog was ultimately buried under sedimentary deposits etc., and the later stages of the conversion into coal were brought about by a number of factors (e.g. the pressure and temperature caused by overlying strata).

Rank of Coal.

The rank of a coal denotes the extent of the alteration produced in the peat deposits by geochemical processes. Since the geological conditions varied widely, solid fuels constitute a progressive series ranging from peat, through lignite, sub-bituminous coal, bituminous coal, semi-anthracite to anthracite. The position of a coal in this series determines its rank, anthracite being of high rank and peat of low rank. In general, the greater the pressure to which a coal was subjected the lower is the oxygen content and the higher is its rank.

It may be stated that the coals of the Coal Measures of South Africa (middle Ecca) are of bituminous rank.

Types of Coal.

In addition to differences in rank, coal seams also display distinctive horizontal bands of coal, which differ in physical and chemical properties. These bands are due to the kind of plant debris from which they have been formed and are said, therefore, to be different types of coal.

Four types of coal are found in normal bituminous seams. Stopes termed these vitrain, clarain, fusain and durain respectively.

Vitrain occurs..../2.

Vitrain occurs in uniform, brilliant, black bands which seldom exceed $\frac{1}{2}$ inch in thickness and are usually $\frac{1}{4}$ inch or less. This type of coal is thought to have been derived from the bark and wood of plants. Vitrain is, with clarain, of considerably lower specific gravity than the other types and is the best coking constituent of coal.

Clarain is described as a very finely laminated coal, with not quite such a brilliant lustre as vitrain, and bearing a distinct "grain" upon its surface. Strictly speaking, this is not a distinctive type of coal as it is considered to consist of alternate fine bands of vitrain and durain.

Fusain is a porous, charcoal-like substance, having a greyish black appearance. This type of coal has probably been derived from the cellular tissues of plants, but little can be said with certainty as to the manner in which it was transformed to its present condition. It has been suggested for example, that forest fires charred the plants and produced charcoal fragments, but this explanation is not generally accepted. The small cells in fusain are frequently filled with mineral matter, thus resulting in a high ash content.

Durain is hard, dull, compact coal and is usually found in comparatively thick bands. It is composed of very finely divided plant material and consequently shows little or no lamination, except where thin bands of vitrain are intermingled. This coal was probably formed during periods when the peat was submerged under water. Instead of normal peat formation, a "mud" was formed which consisted of the finest plant and mineral debris and was subsequently fossilised. The high proportion of mineral matter commonly associated with durain, and the great variations in its constitution, would be explained in this way.

For convenience, the above types of coal are normally classed as "bright" and "dull" coals and are associated as layers constituting a seam.

Bright Coal.

Bright coal consists of vitrain and clarain, but fusain may also occur in small proportions. When the latter is present, the coal tends to split along the horizontal fusain layers. Bright coal is usually well cleaved (i.e. split perpendicular to the bedding planes) and mineral matter frequently occurs as thin films along the cleavage planes. Owing to the presence of relatively coarse plant remains, this type of coal is characterised by a pronounced banded appearance.

Dull Coal.

Durain is the major constituent of dull coal, which is tougher than bright coal and is also not so highly cleaved. Dull coal frequently contains finely divided mineral matter uniformly disseminated throughout the coal substance and consequently tends to have a higher ash content than bright coal. As the percentage of mineral matter increases, dull coal passes through shaly coal to carbonaceous shale.

Impurities in Coal.

The inorganic constituents of coal appear in their most familiar form as the ash which remains after the fuel has been burned. However, the ash is not the mineral matter, it is merely the residue from the ignition in air of the inorganic compounds. It is customary to distinguish between the "inherent" and the "adventitious" impurities as follows:-

Inherent Mineral Matter.

All living plants contain a certain amount of mineral matter which is necessary to their existence. These substances consequently appear in coal as "inherent mineral matter." However, the ash yield from this source rarely exceeds one per-cent.

Adventitious Mineral Matter.

The term, adventitious mineral matter, includes all impurities which were introduced during the peat bog stage, or later. These impurities may be conveniently considered under three sub-headings, viz., primary, secondary and tertiary adventitious mineral matter.

Primary adventitious mineral matter consists of sedimentary material which was either air or water borne into the peat at the time of its formation. This mineral matter may occur either finely disseminated throughout the coal or as horizontal shaly bands or partings.

Secondary adventitious mineral matter appears as thin vein-like sheets in the cleats and joints of coal. This material is considered to have been introduced in solution subsequent to the formation of the coal.

Tertiary adventitious mineral matter includes all material which is liberated during mining operations and appears in the mine product, but which does not form part of the coal seam as such.

"Free Ash" and "Fixed Ash".

In order to distinguish between mineral matter which can not be removed by mechanical cleaning processes and that which can be separated, it is convenient to group the above forms of impurity under "fixed ash" and "free ash" respectively.

Generally speaking, the inherent mineral matter of coal together with finely disseminated primary adventitious material gives rise to the fixed ash, while the remaining forms of adventitious mineral matter constitute the free ash. In cases where the shaly bands are very thin, however, they may not be sufficiently liberated by crushing to a reasonable size and partially remain attached to the coal, thus increasing the fixed ash. It is the object of coal cleaning (or washing) to remove these "free" minerals.

Types of Impurity./....4.

Types of Impurity.

The free minerals normally associated with coals may be divided into three main types, viz.,

(1) Shales, clays and sandstone.

(2) Carbonate minerals.

(3) Sulphide minerals.

Shales are hard and relatively blackened lumps, consisting mainly of Kaolin, Micas, Felspars and some Quartz. Most of the shaly matter occurs as distinct bands and is the result of sedimentary deposition of mud from rivers, etc. Some of the shaly matter may be intimately dispersed throughout the coal, however, and when this percentage is abnormally high, the resultant material is termed carbonaceous shale. The term, shale, thus includes substances which differ widely in composition and physical properties.

Carbonate minerals usually appear as thin white, or rust coloured, plates in the cleats and joints. The most common carbonate minerals found in coal are Calcite ($CaCO_3$), Siderite ($FeCO_3$) and the "Ankerites." The latter have a wide range of composition, being double, triple or quadruple carbonates of calcium, magnesium, iron and manganese.

Sulphides usually appear in coal as silvery or golden patches, bands, or nodules, but they may also be distributed in a fine state of division through the coal substance. The most common of these minerals is Pyrites (FeS_2), which is normally of a golden colour and is often referred to as "fools gold." It is thought that pyrites may have been formed in coal by the action of hydrogen sulphide on infiltrated iron solutions, the H_2S having been produced by bacterial action.

/........5.

THE PURPOSE AND THE VALUE OF CLEANING COAL.

Although it is self evident that impurities in coal are undesirable, the value of low ash fuel to the consumer is not always fully appreciated. A few typical examples of the influence of ash content on the suitability of a coal for various purposes may, therefore, be quoted, in order to stress the importance of reducing the percentage incombustible in the raw coal as far as is possible.

Coking coal having a high ash content is undesirable as it affects the mechanical strength of the coke and also the coke's value for metallurgical smelting processes. The impurities tend to increase the percentage of breeze formed and occupy space which would have been occupied by coal and, thus, decrease the capacity of the coke ovens. One of the main advantages of a low ash content in metallurgical coke is the fact that the quantity of flux required in the blast furnace charge is considerably decreased and the effective capacity of the furnace is consequently increased. It has been claimed (4) (5) that one percent reduction in ash content of the coke will increase the rate of pig iron production from 3 to 6 percent.

In the case of coal used for steam raising, incombustible in the fuel reduces its gross calorific value, gives rise to difficulties of combustion, causes loss of combustible and involves expense in the disposal of ash. Boiler tests using raw coal and cleaned coal of the same size and from the same mine, have indicated that the boiler efficiency is increased from 0.20 to 0.34 per cent for each one per cent of ash reduction(6). The reduction of the sulphur content (by eliminating free pyrites) of a coal which is responsible for the formation of sulphur dioxide in the flue gas, also offers a distinct economical advantage because of the corrosive action of this gas, which can only be minimised by employing relatively high flue and stack temperatures.

The presence of iron pyrites, rock and clay in a coal increases the difficulty of grinding it and causes a high rate of wear in the mills. These impurities should, therefore, be removed as far as possible from coal which is to be used as pulverised fuel.

Although pyrites may not be the primary cause of spontaneous combustion in the majority of cases, it can undoubtedly be a contributary factor(7). The percentage of pyrites in coal which is to be stored in bunkers for any length of time, should, therefore, be as low as possible.

Despite the many disadvantages of a coal having a high ash content, many authorities(8)(9)(10) regard uniformity of quality and size as probably being the greatest attribute of a good fuel. For example, while a good pig iron can be manufactured with a high sulphur coke, fluctuations in the ash and sulphur content will result in variations in the quality of pig iron produced. Any unit burning coal of varying quality is normally adjusted to function with the poorest types of fuel supplied and much of the additional value of a higher grade fuel may be lost if uniformity is not maintained. It will be clear that uniformity of quality can only be ensured by preparing the coal under carefully controlled conditions.

In South Africa,/..6.

In South Africa, the high cost of transportation is probably one of the most important factors in favour of reducing the ash content. The towns and industries have largely grown around the metalliferous mining areas and the harbours, and not on the coalfields, as was the case in Europe. This has resulted in the fuel often having to be transported over very long distances to the consumers. Since the consumer is primarily interested in the cost per B.T.U. at his plant, the relative value of clean coal becomes greater as the freightage increases. Other qualities being equal, the coal with the highest heat value per unit of money will naturally be the one selected. Therefore, as South African coals generally have a high ash content, removal of adventitious dirt before transportation is of prime importance.

It will be clear from the above outline of the influence of ash content on the economic utilization of coal, that it is essential in many instances to subject the raw coal to a cleaning operation in order to reduce the ash content to an acceptable value and to produce a uniform product. The consumer in turn, should be prepared, in view of the advantages accruing, to pay a higher price for uniform, clean coal, thus compensating the producer for the additional expense incurred. In the long run both consumer and producer should benefit by accepting this policy.

In the early days of this industry, coal was mined manually and every effort was usually made to produce the cleanest product possible, by working only the best seams and by excluding the larger visible lenses or partings from the coal sent to the surface. When further preparation was required, handpicking of the larger sizes was considered adequate, the emphasis being placed on the production of saleable large coal, which commanded a higher price than small coal. However, with the development of the coking industry and mechanical stoking equipment etc., the smaller sizes of coal acquired greater value, and, since it is uneconomical to handpick small coal, mechanical cleaning became necessary. In addition, the advent of mechanical mining and loading made underground sorting practically impossible and surface preparation consequently became more important.

By resorting to mechanical coal cleaning, the producer is enabled to continue working his property after the high quality seams have been exhausted and is thus able to hold his market against more fortunate competitors. Coal cleaning is, therefore, of great importance from a national point of view, as this practice enables one to mine inferior coal seams and in this way to utilise the country's resources to the best advantage.

Having discussed the purpose and value of coal cleaning, the processes available and the principles upon which they depend for their operation may now be considered.

THE APPARENT SPECIFIC GRAVITIES OF COAL AND ITS IMPURITIES AND THE DETERMINATION OF WASHABILITY CURVES.

The extent to which impurity and coal can be separated mechanically is governed by relative differences in physical characteristics such as apparent specific gravity, surface properties, size and shape of particles, electrical properties, etc. So far, apparent specific gravity has proved the most important of these characteristics, and it is used in the majority of commercial cleaning processes to effect the separation.

Froth flotation, depending on surface wetting properties, is also extensively practiced for cleaning fine coal and will be discussed in detail in a later section. Processes depending on size, shape, electrical properties etc., are not, as a rule, capable of effecting very sharp separations and do not have a wide-spread application. However, for specific conditions they may be of importance and coal cleaning processes of this type suitable for treating the finer sizes will be described later.

Specific Gravities of Coal and Impurities.

Since coal and most of the associated impurities vary widely in composition, the various major constituents do not have definite specific gravities, but occupy a range of specific gravities. For example, the specific gravity of dull coal is influenced by the percentage of "fixed" mineral matter present, and so on. By way of illustration, typical specific gravities of some of the constituents of coal are given in Table 1. The values quoted for coal and carbonaceous shale are based on the author's experience of Witbank fine coal, while those for the remaining impurities cited appear in well known works of reference(11)(12). In addition to the "pure" substances such as bright coal, dull coal, etc., there are also composite particles consisting of, say, bright and dull coal, dull coal and shale and so on. A sample of coal thus contains particles having a wide range of specific gravity and the quantity present at any given specific gravity depends on the size of the coal considered and the nature of the seam from which it is taken.

In general, the ash content of coal increases with increasing specific gravity.

TABLE 1.

Typical Specific Gravities of Coal and its Impurities.

Constituent.	Specific Gravity.
Bright coal	1.275 to 1.4
Dull coal	1.4 to 1.7
Carbonaceous shale	1.7 to 2.2
Shale	2.0 to 2.6
Clay	2.6
Quartz	2.65
Calcite	2.7
Ankerites	2.5 to 2.7
Siderite	3.85
Marcasite	4.9
Pyrite	5.0

Float and Sink Analysis and Washability Curves.

Before one can decide on the steps to be taken to clean a given coal, it is necessary to determine its specific gravity characteristics when crushed to the size at which it is to be washed. This data is obtained by means of a "float and sink" analysis.

A sample of the coal is immersed in a liquid of known specific gravity and the quantity which floats is collected, dried and weighed. The fraction sinking in the liquid is also collected and dried and is then immersed again in a liquid of higher specific gravity than that of the first test bath. The float fraction (expressed as a percentage of the whole sample) obtained after the second separation, is termed the "fractional yield" between the specific gravities of the two test baths, while the sum of the first and second float fractions is termed the "cumulative yield" at the higher specific gravity i.e. the percentage which would have floated if the sample had been immersed directly in the second test liquid. By carrying out separations in this manner at selected specific gravities, sufficient data can be obtained to enable one to draw curves of fractional yield and cumulative yield against specific gravity. If the ash content of the various float and sink fractions is also determined, the "cumulative ash" at any specific gravity can be calculated.

It is customary to draw curves of "cumulative yield versus specific gravity" and "cumulative ash versus cumulative yield," these two curves being referred to as the "washability curves" of the coal. For convenience, the author proposes naming these curves "the quantity characteristic curve" and "the ash characteristic curve" respectively. Washability curves are used in practice to estimate the specific gravity of separation required to produce a washed coal of any desired ash content, together with the theoretical yield of this material. (As commercial processes are generally less exact than the laboratory float and sink separations, it is necessary to modify the values read off the washability curves if it is desired to estimate the results likely to be obtained in practice. A suitable procedure is explained in a later section.)

TABLE 2.

Float and Sink Analysis of a Typical Witbank Duff Coal.

Fractional.			Cumulative.		
Specific Gravity Interval.	Fractional Yield,%.	Fractional Ash,%.	Specific Gravity.	Cumulative Yield,%.	Cumulative Ash, %.
< 1.3	15.6	3.3	1.3	15.6	3.3
1.3 to 1.35	18.2	5.9	1.35	33.8	4.7
1.35 to 1.4	25.0	7.8	1.4	58.8	6.0
1.4 to 1.45	15.8	11.7	1.45	74.6	7.2
1.45 to 1.5	11.3	15.0	1.5	85.9	8.2
1.5 to 1.58	5.7	17.8	1.58	91.6	8.9
> 1.58	8.4	-	Whole	100.0	12.6

As an example, results of a float and sink analysis of a typical Witbank duff coal are given in Table 2. The fractional values were determined by actual analysis, while the cumulative figures were derived by calculation, as explained below.

The cumulative yield at any specific gravity is merely the sum of all the fractional yields of specific gravity intervals below that value,

$$\text{i.e. Cumulative yield at specific gravity X} = \sum_{0}^{X} \text{Fractional yield.}$$

Thus, cumulative yield at 1.45 specific gravity

$$= 15.6 + 18.2 + 25.0 + 15.8$$

$$= 74.6\%$$

Cumulative ash content at any specific gravity is obtained from the equation:-

$$\text{Cumulative ash at specific gravity X} = \frac{\sum_{0}^{X} (\text{Fractional Yield} \times \text{Fractional Ash})}{\sum_{0}^{X} \text{Fractional Yield}}$$

$$= \frac{\sum_{0}^{X} (\text{Fractional Yield} \times \text{Fractional Ash})}{\text{Cumulative Yield at S.G., X}}$$

$$\text{Hence, Cumulative Ash at 1.45 S.G.} = \frac{(15.6 \times 3.3) + (18.2 \times 5.9) + (25.0 \times 7.8) + (15.8 \times 11.7)}{74.6}$$

$$= \frac{51.5 + 107.3 + 195.0 + 184.8}{74.6}$$

$$= \frac{538.6}{74.6} = 7.2\%$$

The washability curves were plotted from the data in Table 2 and are shown in Figure 1. From these curves it will be seen that if a washed coal of, say, 8% ash content is required, the separation will, theoretically, have to be effected at a specific gravity of about 1.49, and that the yield of this product will be approximately 84%. Thus, by discarding 16% of the raw coal, the ash content could be reduced from 12.6% to 8%. In this example, material having a specific gravity greater than 1.49 would be regarded as "refuse," although it contains a proportion of coal.

Organic liquids, or solutions of inorganic salts in water, are commonly used for float and sink analysis. The former evaporate readily, are not corrosive and have low viscosity. The

vapours are/......10.

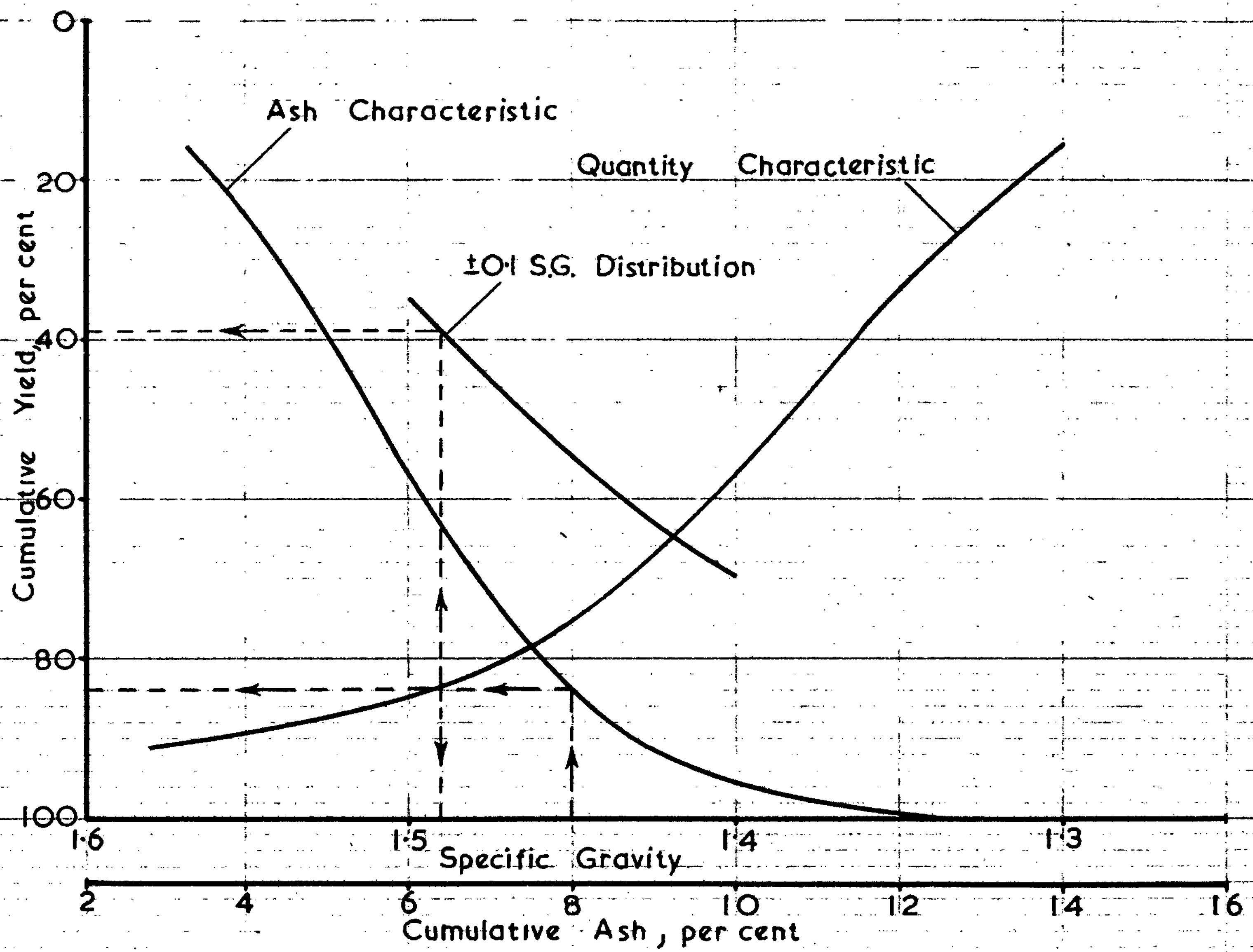

FIGURE 1 Washability curves of a typical Witbank duff coal

vapours are often toxic, however, and the liquids are expensive. Suitable inorganic salts are comparatively cheap and no harmful vapours are given off, but the solutions are corrosive, have high viscosity and liquid adhering to the coal is more difficult to remove from it. Mixtures of benzene and carbon tetrachloride are frequently used for specific gravities up to 1.6, and mixtures of carbon tetrachloride and bromoform for specific gravities between 1.6 and 2.8. Calcium chloride solutions are suitable for specific gravities up to about 1.4, while specific gravities up to 1.8 can be attained with zinc chloride solutions.

The simplest form of apparatus required for float and sink analysis consists of a beaker or bucket, the float fraction being removed by means of a suitable spoon or ladle. More elaborate equipment is available, having sliding gates etc., which mechanically isolate the float and the sink, thus largely eliminating the human element. In the case of coals having a high percentage of near gravity particles, it is desirable to immerse the apparatus in a thermostatic bath, in order to minimise specific gravity fluctuations due to temperature variations. Since coal is a porous substance, the apparent specific gravity varies with moisture content and the latter should, therefore, be carefully controlled if a high degree of accuracy is required.

Assessment of the Difficulty of a Washing Problem.

Having obtained the washability curves of the coal and having decided on the specific gravity at which the separation is to be effected, it is necessary to assess the difficulty of the washing problem in order to select a suitable washer.

According to Bird(13)(14), the ± 0.10 specific gravity distribution at the desired specific gravity of separation is an index of the difficulty of washing a coal. This value is readily derived from the washability curves as explained in the following example.

Returning to Figure 1 and taking the specific gravity of separation as 1.45, the percentage of coal within ± 0.10 specific gravity will be given by the difference between the cumulative yields at 1.55 and 1.35 specific gravity. It is found from the quantity characteristic curve that the yields at 1.55 and 1.35 are 90% and 34% respectively, and the difference is 56%. The ± 0.10 specific gravity distribution at a separation specific gravity of 1.45 is thus 56%. By proceeding in this manner, the "$\pm$ 0.10 specific gravity distribution versus specific gravity curve" may be drawn as shown in Figure 1.

Since the presence of material having a specific gravity greater than about 2.0 does not influence normal separations, it is necessary to consider only that portion of the coal having a specific gravity lower than 2.0 in determining the ± 0.10 specific gravity distribution, otherwise anomalous values may be obtained. For example, assume that the Witbank duff coal, which has already been considered, contains no material having specific gravity greater than 2.0. Suppose that an

Although assessment of the difficulty of a washing problem with the aid of the ± 0.10 distribution curve is undoubtedly valuable as a rapid guide, a more detailed study is necessary before the final selection of the washer that is to be installed is made. A suitable procedure is suggested in a later chapter.

Particle Falling Freely in a Still Liquid.[15][16]

The forces acting on a particle, which is moving in a fluid under the force of gravity, may be summarised as follows:-

(a) The force of gravity acting vertically down (mg).

(b) The buoyancy of the liquid acting vertically upwards $\left(\frac{4}{3}\pi r^3 \rho g\right)$

(c) The resistance force opposing motion and acting vertically upwards (F).

The resultant force acting on the particle and producing the acceleration is the algebraic sum of these three forces,

i.e. $$m\frac{dV}{dt} = mg - \frac{4}{3}\pi r^3 \rho g - F \quad \text{———————} \quad (1)$$

Expressing the mass of the particle in terms of the radius and density, the general equation of motion becomes,

$$\frac{4}{3}\pi r^3 s \frac{dV}{dt} = \frac{4\pi r^3 sg}{3} - \frac{4}{3}\pi r^3 \rho g - F \quad \text{———————} \quad (2)$$

which, when solved for $\frac{dV}{dt}$, the acceleration, gives,

$$\frac{dV}{dt} = g - \frac{\rho g}{s} - \frac{3F}{4\pi r^3 s}$$

$$= \left(\frac{s-\rho}{s}\right) g - \frac{3F}{4\pi r^3 s} \quad \text{———————} \quad (3)$$

It will be seen that the first term of this expression depends only on the density or specific gravity of the particle, while the second term includes the size of the particle.

The problem of determining the resistance, F, encountered by a particle moving through a viscons fluid, received a great deal of attention by physicists, notably by Stokes, Newton and Rittinger. The following expressions were obtained by these investigators for the resistance to the motion of spherical particles:-

Stoke's law, $F = 6\pi\mu rV$ ——————— (4)

Newton's law, $F = \frac{\pi}{2}\rho r^2 V^2$ ——————— (5)

Rittinger's law, $F = Q\frac{\pi}{2}\rho r^2 V^2$ ——————— (6)

where Q is an experimental coefficient of resistance.

It will be clear that Rittinger's law is merely the general form of Newton's equation.

From the observations of several investigators it was found that the resistance coefficient, Q, is not a constant but is a function of the Reynold's number, and the general equation of resistance may, thus, be expressed as

$$F = \left| fR \right| \frac{\Pi}{2} \rho r^2 V^2$$

The Reynold's number for a sphere is given by the expression

$$R = \frac{2Vr\rho}{\mu} \qquad \text{——————} \quad (7)$$

and is a dimensionless quantity.

The relationship between the coefficient of resistance, Q, and the Reynold's number, R, is shown graphically in Figure 2. Although an equation could no doubt be derived which would satisfy the major portion of the curve in Figure 2, a preliminary investigation has shown that the resultant expression is likely to be cumbersome if a good fit is desired. It is consequently more convenient, for the time being, to divide the figure into three zones as shown.

<u>Zone 1</u> (R less than 1)

Q is inversely proportional to R in this zone

$$\text{i.e.} \quad Q = \frac{k}{R}$$

$$\text{Hence} \quad F' = \frac{k}{R} \times \frac{\Pi}{2} \rho r^2 V^2$$

$$= \frac{\mu k}{2vr\rho} \times \frac{\Pi}{2} \rho r^2 V^2 = k' \Pi \mu r V$$

This expression will be recognised as Stoke's law, which, thus, only applies when R is less than one.

<u>Zone 2</u> ($R = 10^3$ to $R = 2 \times 10^5$)

Q is constant, for all practical purposes, in this region and has a value of approximately 0.4. Rittinger's law, then, applies in this zone.

<u>Zone 3</u> ($R = 1$ to $R = 10^3$)

In zone 3, the general form of the curve suggests that Q is inversely proportional to R^n, where n varies between 1 and zero. This part of the curve can be approximated by Allen's formula for the velocity of fall[17]:-

$$V = \left(\frac{4\,g}{30\rho/\alpha} \right)^{\frac{2}{3}} d\,(s - \rho)^{\frac{2}{3}} \qquad \text{——————} \quad (8)$$

where α = kinematic viscosity

and d = diameter of the particle

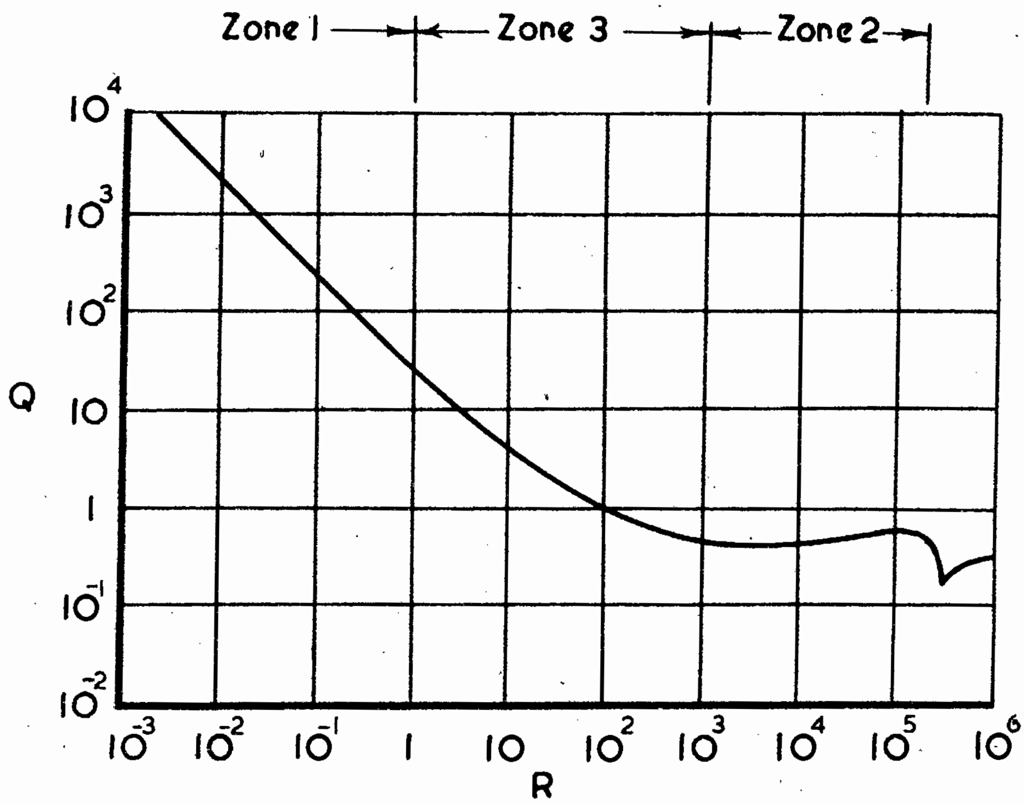

FIGURE 2 Relationship between Coefficient of Resistance and Reynolds Number

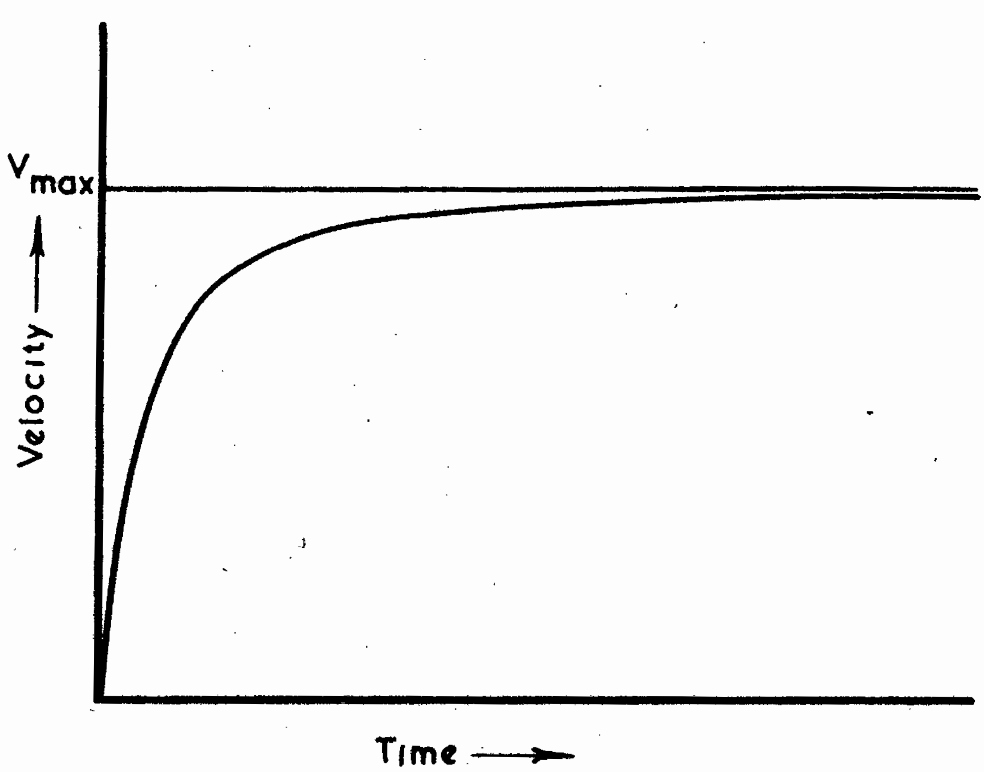

FIGURE 3 Motion of a particle in a still liquid

It will thus be seen that Stoke's law only holds for very small particles falling at low velocities or particles moving in very viscous fluids, and that Newton's law applies to a limited range of conditions. Rittinger's law, on the other hand, applies to the larger particles encountered in practice. In order to obtain a conception of the order of magnitude of particles to which the three zones of the Q - R curve apply, the author calculated the Reynold's number for coal and dirt particles moving in water at their terminal velocities. It was found that Stoke's law would apply to particles smaller than about 100 mesh, and Rittinger's law (Q a constant) to particles having diameters greater than 0.1 - 0.2 inch. Since the greatest proportion of coal normally cleaned is larger than 0.2 inch, the present analysis will be confined to particles obeying Rittinger's law (equation 6).

Substituting the value of F from equation (6) in equation (3), the acceleration is given by

$$\frac{dV}{dt} = \left(\frac{s - \rho}{s}\right)g - \frac{3\,Q\,\rho V^2}{8\,r\,s} \qquad (9)$$

Putting $\left(\frac{s - \rho}{s}\right) g = A$, and $\frac{3\,Q\,\rho}{8\,r\,s} = B$,

Equation (9) may be written

$$\frac{dV}{dt} = A - BV^2$$

$$\text{or } dt = \frac{dV}{A - BV^2} = \frac{1}{A}\,\frac{dV}{1 - \frac{B}{A}V^2}$$

$$= \frac{1}{A}\cdot\frac{dV}{1 - C^2V^2} \qquad \text{where } C = \sqrt{\frac{B}{A}}$$

$$= \frac{1}{2A}\left[\frac{1}{1 + CV} + \frac{1}{1 - CV}\right] dV.$$

Integrating,

$$t = \frac{1}{2A}\left[\frac{1}{C}\log_e(1 + CV) - \frac{1}{C}\log_e(1 - CV)\right] + P$$

$$= \frac{1}{2AC}\log_e\frac{1 + CV}{1 - CV} + P$$

When $t = 0$, $V = 0$ $\therefore P = 0$

Hence, $t = \frac{1}{2AC}\log_e\frac{1 + CV}{1 - CV}$.

Solving for the velocity,

$$V = \frac{1}{C}\,\frac{1 - e^{-2ACt}}{1 + e^{-2ACt}} \qquad (10)$$

When $t = \infty$, the expression for the terminal velocity is obtained.

/.........17.

and could

Putting $s_2 = 2.0$, $s_1 = 1.4$ and taking

$\rho = 1$ for water (using suitable units, S.G. and density have the same numerical value).

$$\frac{r_1}{r_2} = \frac{2.0 - 1}{1.4 - 1} = \frac{1.0}{0.4} = 2.5$$

or $r_1 = 2.5\, r_2$

i.e. the "light" particle and the "heavy" particle would fall with equal terminal velocities in water if the former has 2.5 times the radius of the latter.

It appears, then, from the above analysis, that a true separation according to specific gravity, irrespective of size, is only possible during the initial acceleration period, as the velocity of fall increases, there is compensation between size and density and particles of different specific gravities (or densities) will tend to fall together. A density separation of particles which have attained their terminal velocities will only be possible if the feed is closely graded according to size. As close size grading would be an undue limitation in practice, impulses may be applied to the washing medium so that it moves periodically upwards and downwards. As will be shown later the particles are prevented from attaining their terminal velocities in this way and the equivalent of the free fall acceleratory period, may be employed to effect a density separation.

Motion of a Particle in Upward and Downward Currents of Liquid. (16)

Since the majority of practical coal cleaning processes employ rising or falling currents of water, or both, to effect the separation, the influence of these conditions must now be considered.

(a) The Motion of a Particle in a Vertically Upward Current.

For the purpose of this analysis, the velocity of the particle is taken as positive in an upward direction. Let the velocity of the upward current be represented by W and assume that the particle obeys Rittinger's law (equation 6). The general equation of motion is then

$$\frac{mdV}{dt} = -mg + \frac{4}{3}\pi r^3 \rho g + Q\frac{\pi}{2}\rho r^2 \left[W - V\right]^2$$

$$\text{or } \frac{dV}{dt} = \frac{3}{8}\frac{Q\rho}{rs}\left[W - V\right]^2 - \left[\frac{s - \rho}{s}\right] g \qquad (15)$$

The terminal/.......19.

The terminal velocity of the particle is readily obtained from this expression by equating the acceleration to zero and solving for the velocity,

$$\text{i.e. } V_{max} = W - \sqrt{\frac{8g}{3Q} \frac{(s - \rho)}{\rho} r}$$

$$= W - K\sqrt{\frac{r(s - \rho)}{\rho}} \qquad \text{———————} \qquad (16)$$

From equation 16, it will be clear that

(1) The particle will rise if

$$W > K\sqrt{\frac{r(s - \rho)}{\rho}}$$

(2) The particle will fall if

$$W < K\sqrt{\frac{r(s - \rho)}{\rho}}$$

(3) The particle will remain in suspension if

$$W = K\sqrt{\frac{r(s - \rho)}{\rho}}$$

During the first instant after the particle has been subjected to an upward current (i.e. $V = 0$) the acceleration is

$$\frac{dV}{dt} = \frac{3}{8}\frac{Q\rho W^2}{rs} - \left[\frac{s - \rho}{s}\right] g \qquad \text{———————} \qquad (17)$$

As the velocity of the particle increases, the acceleration decreases, W^2 in equation 17 being replaced by $(W - V)^2$, i.e. the first term in equation 15 decreases with increasing particle velocity. Since this term involves the size of the particles, it follows that as the velocity of the particles increases, the separation is effected more and more according to specific gravity.

For a separation according to specific gravity only, independently of size, it would be necessary for the first term in equation 15 to be equal to zero i.e. $W = V$. However, equation 16 indicates that the maximum velocity which the particles can attain in an upward current is less than W, so that V can never equal W. Washers employing upward currents of water only would, therefore, not be able to wash unsized coal.

Consider a coal and a dirt particle of equal size, having densities S_1 and S_2 and accelerations g_1 and g_2 respectively, rising in an upward current.

$$\text{Then } g_1 - g_2 = \frac{dV_1}{dt} - \frac{dV_2}{dt}$$

where V_1 and V_2 are the velocities of the two particles at any instant. At the beginning of the motion V_1 and V_2 would be very nearly equal and, as a first approximation, $V_1 = V_2 = V$

From equation (15),

$$g_1 - g_2 = \left\{\frac{3Q\rho}{8rs_1}\left[W - V\right]^2 - \left[\frac{s_1 - \rho}{s_1}\right]g\right\} - \left\{\frac{3Q\rho}{8rs_2}\left[W - V\right]^2 - \left[\frac{s_2 - \rho}{s_2}\right]g\right\}$$

$$= \frac{3Q\rho}{8r}\left[W - V\right]^2\left[\frac{s_2 - s_1}{s_1 s_2}\right] + g\left[\frac{s_2 - s_1}{s_1 s_2}\right]\rho \qquad (18)$$

Since $s_2 > s_1$ this expression is always positive and $g_1 > g_2$ i.e. a coal particle ascends more rapidly than a dirt particle of equal size. Similarly, it can be shown that the dirt particle falls more rapidly than the coal particle if the upward current velocity is not sufficient to cause the particles to rise. In other words, at the inception of an upward current, at whatever speed the current is moving, coal particles tend to attain a position above dirt particles of equal size.

The motion of two particles which are not of the same size may also be deduced from equation 15. If the coal and dirt particles have radii r_1 and r_2 respectively, it can be shown that separation will take place provided that

$$\frac{1}{r_1 s_1} > \frac{1}{r_2\, s_2}$$

$$\text{or} \quad \frac{r_1}{r_2} < \frac{s_2}{s_1} \qquad (19)$$

This indicates that very close sizing would be required in order to separate coal from dirt in an upward current.

(b) <u>The Motion of a Particle in a Vertically Downward Current.</u>

For the purpose of this analysis the velocity will be taken as positive in a downward direction.

The motion of a particle in a vertically downward current may be divided into two phases, viz.

(1) Acceleration/..21

(1) Acceleration from rest until the particle has attained the velocity of the current.

(2) Acceleration after this instant until the terminal velocity is reached.

During the first phase, the equation of motion is

$$\frac{m\,dV}{dt} = mg - \frac{4}{3}\pi r^3 \rho g + Q\frac{\pi}{2}\rho r^2 \left[W - V\right]^2$$

$$\text{or } \frac{dV}{dt} = \frac{3Q\rho}{8rs}\left[W - V\right]^2 + g\left[\frac{s - \rho}{s}\right] \qquad (20)$$

During the second phase, the equation of motion is given by

$$m\frac{dV}{dt} = mg - \frac{4}{3}\pi r^3 \rho g - Q\frac{\pi}{2}\rho r^2 \left[V - W\right]^2$$

$$\text{or } \frac{dV}{dt} = g\left[\frac{s - \rho}{s}\right] - \frac{3Q\rho}{8rs}\left[V - W\right]^2 \qquad (21)$$

Putting $\frac{dV}{dt}$ equal to zero and solving for the velocity,

$$V_{max} = W + \sqrt{\frac{8rg}{3Q}\frac{(s - \rho)}{\rho}} \qquad (22)$$

At the beginning of the first phase, V is negligible,

$$\text{hence } \frac{dV}{dt} = \frac{3Q\rho}{8rs}W^2 + g\left[\frac{s - \rho}{s}\right] \qquad (23)$$

Considering two particles of equal size having densities s_1 (coal) and s_2 (dirt) and accelerations g_1 and g_2 respectively.

From equation 23,

$$g_1 - g_2 = \left\{\frac{3Q\rho}{8rs_1}W^2 + g\left[\frac{s_1 - \rho}{s_1}\right]\right\} - \left\{\frac{3Q\rho}{8rs_2}W^2 + g\left[\frac{s_2 - \rho}{s_2}\right]\right\}$$

$$= \left[\frac{1}{s_1} - \frac{1}{s_2}\right]\left[\frac{3Q\rho}{8r}W^2 - \rho g\right]$$

Since $s_2 > s_1$, $\left[\frac{1}{s_1} - \frac{1}{s_2}\right]$ will always be positive

and $g_1 - g_2$ will be positive when $\frac{3Q\rho}{8r}W^2 > \rho g$

and negative when $\frac{3Q\rho}{8r}W^2 < \rho g$.

In other/.....22.

In other words, if the current velocity, W, is high the coal particle will fall faster than the dirt and when the current velocity is very low, dirt will accelerate faster than coal of equal size.

At the beginning of the second phase, $V = W$ and the acceleration is given by

$$\frac{dV}{dt} = g\left[\frac{s - \rho}{s}\right]$$

i.e. the acceleration depends only on the density of the particle, and dirt would fall faster than coal irrespective of size. From equation 21 it will be clear that the size of the particles would be of greater importance in modifying a density separation as the value of V increases.

Hindered Settling.

In the foregoing discussion of the motion of a particle in a liquid, it was assumed that the particles were spherical and free to move when acted upon by the forces indicated. In practice, however, the particles are not spherical and since a large number of particles are present in the washing medium they affect one anothers motion and give rise to hindered settling conditions. In addition, differences in shape also tend to modify the separation. A detailed analysis of the various factors involved consequently becomes very complex.

Although the formulae which have been derived are not directly applicable to practical conditions, they serve to illustrate the fundamental principles employed in several types of washers in common use and many of the conclusions which have been drawn are substantiated in practice.

A detailed discussion of hindered settling conditions is beyond the scope of the present paper, but one important established principle should be mentioned viz. a suspension of solids in a fluid behaves as though it were a liquid having a density equal to the mass of the suspension divided by its volume and a particle immersed in the suspension is buoyed up in accordance with Archimede's law. This conclusion is amply borne out by experiments with particles which are large compared with the suspension particles. Many modern washers employ this fact to effect a true density separation and the results obtained compare favourably with float and sink separations in a heavy liquid.

/......23.

TYPES OF COAL WASHERS.

As explained earlier, the mechanical cleaning of coal is effected by taking advantage of differences in the physical properties of the coal and dirt. The more important differences are those of

(1) Specific gravity.

(2) Shape.

(3) Surface wetting properties.

(4) Frictional resistance on a metal slope.

(5) Resilience.

(6) Electrical properties.

Differences in specific gravity and shape of coal and dirt, form the basis on which the majority of cleaning processes operate. Both wet and dry processes are used, but the former are most common.

Coal washers in general use may be broadly classified as follows according to the physical properties employed to effect the separation.

A. Specific Gravity and Shape.

(a) Washers employing a continuous upward current of water (classifier type).

(b) Washers employing continuous horizontal currents of water (Trough type).

(c) Washers employing a pulsating vertical current of water (Jig type).

(d) Tables using a horizontal stream of water and vibrating in a direction at right angles to the direction of the current.

(e) Reciprocating tables in conjunction with an upward current of air.

B. Specific Gravity.

Washers in which the separation is according to specific gravity and is but little influenced by size and shape of particles. Washers of this type are frequently referred to as "Float and Sink" processes as a bath of intermediate specific gravity is employed to effect the separation of coal from dirt. The separating bath usually consists of a suspension of a fine solid in water. It has also been suggested that air could be used instead of water.

C. Surface Wetting Properties.

Froth flotation processes employ this property. Air bubbles are formed in water by aeration or mechanical agitation and are reinforced by the use of oils such as gas oil or pine oil. These bubbles become attached to the coal particles and raise them to the surface while the dirt sinks.

/.........24.

D. Miscellaneous Processes.

Washers which employ differences in physical properties such as friction, resilience, electrical properties etc. are included in this group.

Typical examples of the more important of the above types of washers will be briefly described in this section, while a later section will be devoted to a discussion of the relative efficiencies of the various types.

Coal Washers of the Classifier Type.

Washers of this type employ upward currents of water to effect the separation of coal from dirt. It will be clear from equation 19, that the feed would have to be very closely sized before efficient separation could be effected by a single rising current of water. As this would be an undue limitation, washers of this type usually consist of conical vessels which allow the use of different velocities at different points in the apparatus. In this way the size ratio of material which can be treated is widened substantially.

Although coals, 4 to 5 inches in size, have been washed successfully in classifiers, they are particularly suited to the cleaning of the finer sizes of coal.

The following are typical examples of this type.

Robinson Washer. (18) (19)

This washer was developed by Robinson in England in 1885 and soon won wide acceptance in that country and in the United States.

The washer is illustrated diagrammatically in Figure 4. The separating vessel consists of the inverted frustrum of a cone, A, constructed of cast iron or steel. Units having a capacity of 30 to 40 tons of feed per hour, have a diameter of some 10 to 12 feet at the upper end, depth of 8 to 10 feet and an apex angle of about 60 degrees.

The lower end of the vessel is fitted with an annular chamber, B, perforated on the inside wall with several rows of holes. Water is supplied under pressure to this chamber and is forced up through the cone in a number of jets.

A vertical drive shaft, C, carries four cross beams, D, from which wrought iron bars, E, project into the cone. Four short bent arms are also attached to the lower end of the driving shaft. This shaft is rotated at about 14 R.P.M., thus giving the water a circular motion in addition to its upward movement. The chief function of the slow rotation is to break up agglomerations of particles and to carry the cleaned coal round the cone to the discharge lip.

The coal is fed into the cone through an iron baffle tube, F, attached to the cross beams and projecting about 15 inches beneath the surface of the water. In this way, all particles are made to come under the influence of the rising currents and masses of unseparated particles can not float along the surface to the discharge point.

The raw coal/.....25.

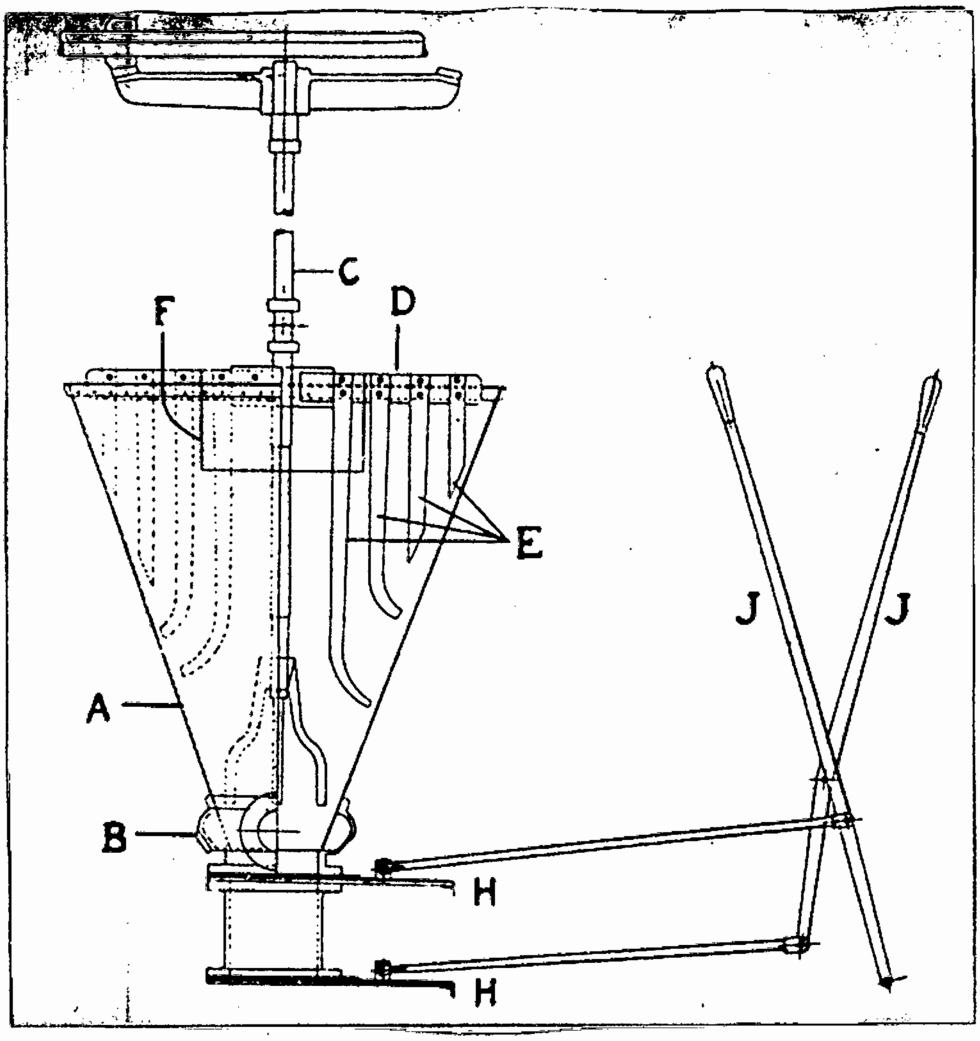

FIGURE 4. ROBINSON WASHER.

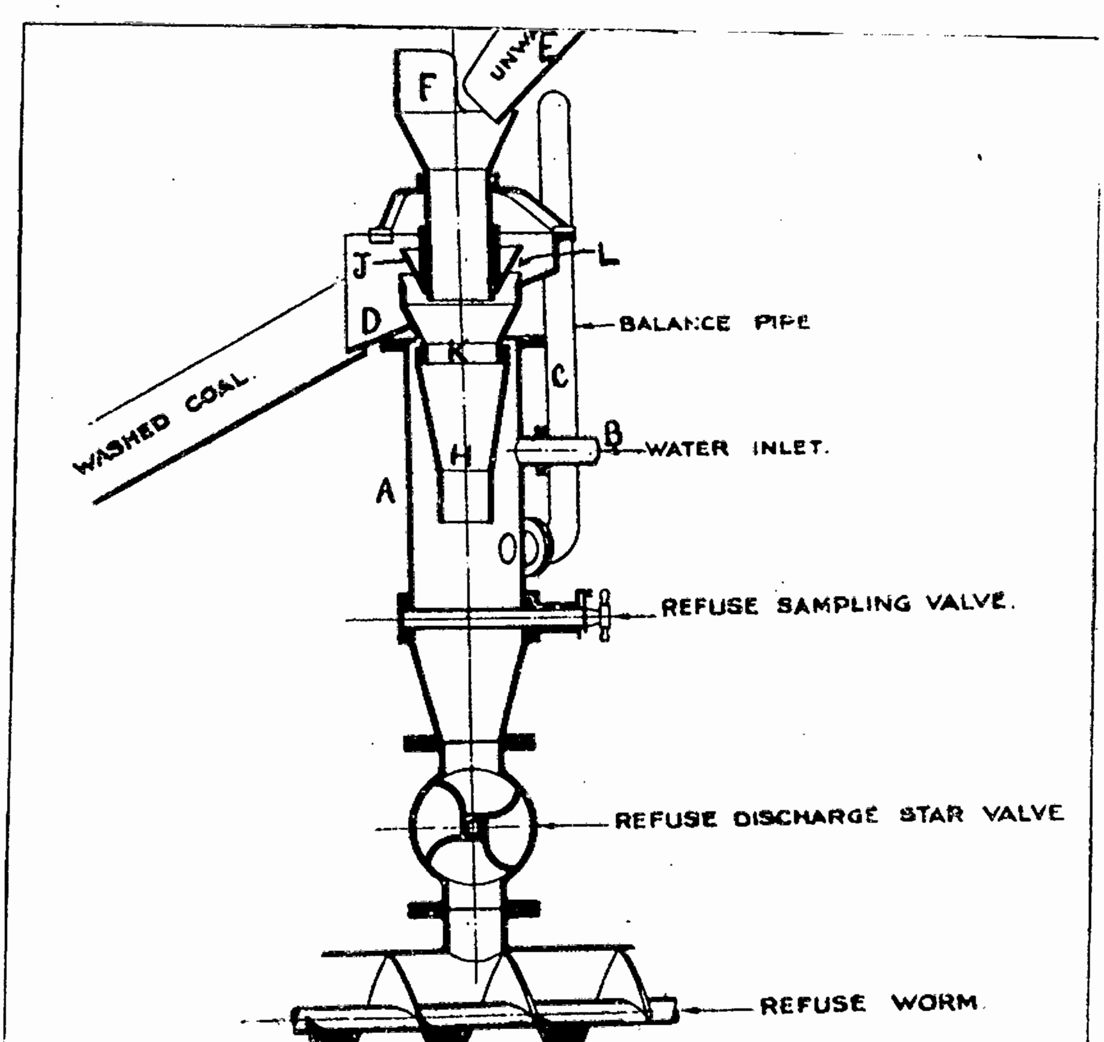

FIGURE 5. DRAPER TUBE.

The raw coal is separated by the upward currents of water into clean coal and refuse. The clean coal overflows at a lip cut in the rim of the cone, while the refuse sinks through the water into a refuse chamber and is discharged by operating the sliding valves, H, by means of the levers, J. The sequence of refuse valve operation is as follows:- the upper valve is open while the lower valve is closed and refuse collects in the chamber, periodically the upper valve is closed and the lower valve opened in order to discharge refuse.

Water overflowing with the washed coal is removed on a drainage screen and is returned to the supply tank via a water clarifier, in order to remove fine coal particles which are in suspension.

Although the Robinson washer may be used for unsized coal ranging from about 2½ inches top size, the results are more satisfactory if the coal is screened into suitable size fractions before washing. From this it appears that the necessity for screening or not, in any particular case, would depend on the difficulty of the washing problem. This washer has the advantages of low capital and operating cost. It is compact and easy to operate. However, these advantages are somewhat counterbalanced when it is necessary to wash the coal in a number of size fractions.

In 1934 Menzies brought out a separator which is generally similar to the Robinson washer, except that water is introduced at several horizons in the cone by rings of nozzles, as well as at the bottom through a classifier column.

The Draper Washer [18] [19]

In England, M.J. Draper, about 1917, invented the washing tube bearing his name. The latest form of this washer is shown diagrammatically in Figure 5.

Raw coal is fed from a chute, E, into a feeding funnel, F, from which it falls into the washing tube, H. An adjustable cone shaped sleeve, J, is attached to the neck of the feeding funnel, and controls the annular area, L, at the mouth of the discharge tube, K. This area is adjusted so that the velocity of the ascending current of water is increased at the discharge point, thus rapidly discharging the clean coal.

Water is admitted to the washer at a horizontal and tangential feed pipe, B. The upward current of water in the washing tube, H, consequently has a spiral motion. This causes flat, plate-like particles, such as calcite, to turn over and fall more rapidly than would otherwise be the case.

The water below the washing tube remains relatively still and the refuse particles fall rapidly to the bottom of the washer where they are discharged by a revolving star valve into a screw conveyer. The latter collects the refuse from a battery of tubes and delivers it into a refuse hopper.

This washer has the serious disadvantages of requiring close sizing of feed, large water circulation per ton treated, and small capacity per unit. It is claimed that Draper tubes can wash coal ranging in size from 3 inches to $\frac{1}{100}$ inch with equal ease (22), each unit being capable of dealing with 5 to 10 tons of feed per hour.

Sy-Vor Washer (20) (21)

The Sy-Vor washer was recently developed in England by R.L.R. Slacke for cleaning fine coal. Although upward currents of water are also employed in this process, the separation of coal from dirt depends mainly on the creation of a fluid mass or "quick sand". This condition also influences the separation in other types of classifier washers to a certain extent.

The working principle of the washer is shown in Figure 6. The apparatus consists essentially of a three legged syphon. Two open water tanks, a and b, are connected by a syphon formed by the conical vessel, c, and the U tube, c_1. Water flows through the spigot valve, d, through the conical vessel and U tube due to the head h_1. A clean coal discharge pipe is fitted into the separating vessel as shown, and forms the third leg of the syphon. Flow through this tube is controlled by the head h_2.

The coal to be cleaned is fed to the constant head tank, e, and the granular coal and shale gravitates to the separating vessel. Fine slimes overflow directly to the tank, b. Since the rising velocity through the spigot, d, is sufficient to prevent discharge of solids, and the greatly reduced velocity in the enlarged area of the cone, c, is insufficient to carry the granular particles, these accumulate at the bottom of the cone and form a high specific gravity suspension medium sufficient to support the coal. The latter thus eventually builds up to the end of the clean coal collector and is discharged.

As the shale accumulates in the bottom of the cone, the mean density of the mixture increases in relation to the density of the overflow mixture in the U tube. The effective head h_1 is consequently reduced with corresponding reduction of rising velocity through the spigot, thus automatically discharging excess shale.

A slight pulsating motion has been found to assist the rapid sorting of coal from shale. This motion is effected by a diaphragm, k, connected to the upper portion of the conical vessel.

As far as is known at present, only a small pilot plant has been operated and little technical data are consequently available. It is claimed, however, that material ranging from $\frac{1}{8}$ inch to about $\frac{1}{200}$ inch can be successfully treated. A few preliminary laboratory scale experiments were carried out at the Fuel Research Institute and this work indicated that the Sy-Vor would only be suitable for effecting separations at point in the washability curve where the $\pm$ 0.1 S.G. Distribution is low, i.e. with the South African coals tested, at a high specific gravity. In any event, the simplicity of the apparatus and its operation is appealing and it may possibly be of great value for certain applications.

Jig Washers/......27.

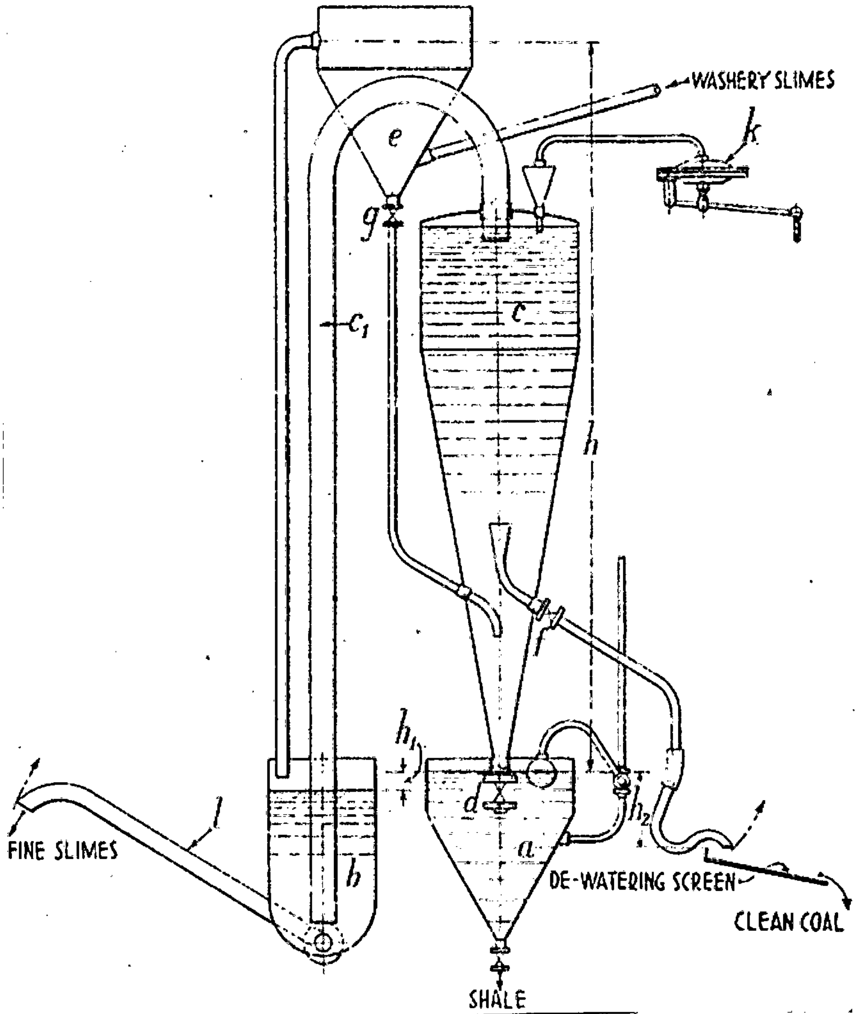

FIGURE 6: DIAGRAM OF SY-VOR WASHER.

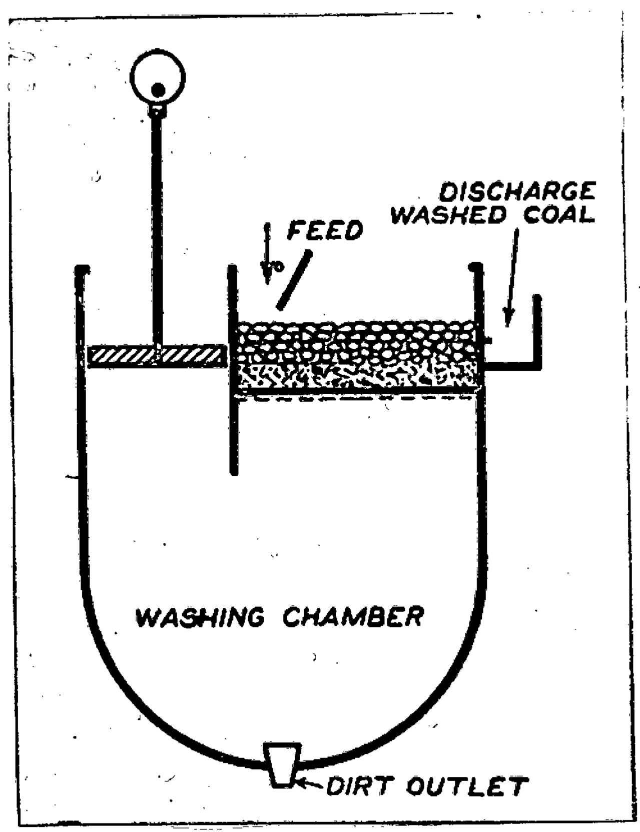

FIGURE 7: DIAGRAM OF A PLUNGER JIG.

Jig Washers.

Jig washers depend on pulsating currents of water for their action and are among the earliest processes used for the concentration of minerals and the cleaning of coal. Since this type of washer requires little supervision, has low capital and operating costs and is able to wash unsized feed, it is extensively used in modern coal cleaning practice.

The forerunner of the modern jig consisted simply of a crude wickerwork basket, which, when about half filled with coal, was placed in a waterbath and was given an up and down motion for such a period as the operator considered necessary. This movement in the water caused a rough stratification, the lighter material, appearing in the upper layers, was then scraped off. In the next stage of the evolution of the jig, a box, having a perforated base, was suspended from a lever, which was given an up and down motion manually. Later forms of this washer consisted of a curved box, having a vertical partition, virtually converting the vessel into a U tube. The material to be cleaned rested on a perforated plate in the one leg and the water was given a pulsating motion by a mechanically driven plunger situated in the other leg of the U tube. The general arrangement is shown in Figure 7. Modern jigs are essentially of the same form, except that the plunger has been replaced by a sealed chamber, air being admitted and exhausted alternately to provide the pulsating motion. This method of operation has the advantage that the jig cycle can be readily altered. Provision was also made in the later types for continuous disposal of clean coal and refuse.

In order to explain the operating principle of a jig, it is necessary to return to the fundamental equations which were derived for the behaviour of a particle in upward and downward currents of water. To facilitate reference, the relevant equations of motion are listed below.

(a) Upward Current

$$\frac{dV}{dt} = \frac{3Q\rho}{8rs}\left[W - V\right]^2 - \left[\frac{s-\rho}{s}\right]g \qquad (15)$$

(b) Downward Current

First phase of motion

$$\frac{dV}{dt} = \frac{3Q\rho}{8rs}\left[W - V\right]^2 + \left[\frac{s-\rho}{s}\right]g \qquad (20)$$

second phase of motion

$$\frac{dV}{dt} = \left[\frac{s-\rho}{s}\right]g - \frac{3Q\rho}{8rs}\left[V - W\right]^2 \qquad (21)$$

(c) Still Water

$$\frac{dV}{dt} = \left[\frac{s-\rho}{s}\right]g - \frac{3Q\rho}{8rs}V^2 \qquad (9)$$

The early/......28.

The early theory[23] of correct jig operation was that the separation should be effected mainly during the upward current of water and that the downward current was a disadvantage and should, therefore, be minimised. Now, it will be clear from equation (15) that the velocity of the water and the velocity of the particle should be nearly equal in order to effect a separation in an upward current as closely as possible according to density and to avoid the necessity of sizing the feed. The velocity of the upward current was consequently kept as low as possible at its inception and was allowed to increase as the particles accelerated. As the water changes its motion from an upward to a downward current it becomes momentarily still and if the particles are also still relative to the water, conditions are produced for a brief interval, during which the separation is strictly according to density i.e. $V = 0$ in equation 9. Therefore, at the beginning of the downward current, the particles are arranged so that the coal forms an upper layer and the dirt a lower layer. As explained before, if the downward velocity is high, the coal will fall faster than the dirt and thus the separation effected during the upward current would be destroyed. To avoid this, the downward current through the coal was minimised by admitting water from an external source underneath the plunger during its upstroke. Excess water was discharged with the clean coal during the downstroke of the plunger (upward current).

More recently, however, Bird[24] found experimentally that a separation, which is nearer to a true density separation can be obtained in the downward current of water. His cycle of events and explanation of the separation are as follows. At the commencement of the downstroke, the water is given a sharp upward acceleration so that the bed of coal is lifted en mass. The acceleration of the water is then allowed to decrease rapidly, thus allowing the coal bed to open from the bottom upward and in this way the particles are suspended in the water ready for separation. The water and particles are then allowed to fall together and the separation is effected in three phases. In the initial phase, the water and coal fall together and density is the only factor influencing the separation (i.e. $V = W$ in equation 20). Then follows a phase in which the particles gain on the water and size influences the separation ($V > W$ in equation 21). The particles then come to rest on the screen plate, and just as the bed is closing, the particles and the water pass through a stage in which they again fall at the same speed and size has no influence. The last phase is considered to be of great fundamental importance in jigging unsized feeds.

The various mechanical devices which are employed to effect the separation in accordance with the above principles, are best illustrated by means of an example of a modern jig.

The Baum Jig.

The Baum Jig was first introduced in 1892 and, with various modifications, is probably the most common type of jig in general use at present. The distinctive feature of this jig is the fact that the pulsations of water are produced with compressed air. This has the advantage that the moving parts are reduced to a simple air valve which can not come into contact with abrasive particles and the operating cycle can be more readily varied in accordance with the requirements.

The diagrammatic view of an early Baum jig shown in Figure 8 will serve to illustrate its construction. The jig box is U shaped and is divided into a number of compartments and cells. A compartment usually consists of two or more cells and is, in fact, a complete jig in itself. A multiple compartment jig is, therefore, in reality a series arrangement of two or more jigs. Thus, in the example illustrated, the jig consists of two compartments, the first having three cells and the second two cells. A screen plate, d and e, is fitted on one side of each compartment near the top and the air valves, a, are situated at the top of the other side of the U, one to each cell. These valves are actuated by means of eccentrics, c, and admit and exhaust the air that produces the pulsations. The amount of air can be regulated by the stroke of the valve, the intensity of the pulsation by the pressure of the air, and the frequency of the pulsations by the speed of the eccentric shaft.

Raw coal is fed to the larger of the two compartments and the heavier dirt is discharged at the feed end of the jig through the gate, f, and falls through the chute, g, into the elevator pit, h. The lighter material overflows into the second compartment where the clean coal is separated and overflows at i. Lighter refuse and middlings are discharged at K and fall through the chute, l, into the elevator pit, m. Fine refuse which passes through the screen plates is transported by the conveyors, n1 and n2, to the elevator pits, h and m.

The rate of refuse discharge is controlled by a float which operates a refuse ejector as shown in Figure 9. This float is simply a large hydrometer which measures the specific gravity of the solids-water mixture at a selected level in the jig bed.

The various features described above are clearly shown in the ghost view of a modern jig, Figure 10.

Jigs are used in practice to wash coal ranging in size from 8 inches to zero and it is claimed that material down to 65 mesh can be cleaned effectively.

Trough Washers.[27] [28]

Trough or launder washers were among the earliest types used for the concentration of ores and were first applied to the cleaning of coal in about 1841.

These washers consist essentially of an inclined trough or channel down which a stream of water, bearing coal, is made to flow. Due to turbulent flow among the particles, they become stratified according to size and specific gravity, in a manner similar to that occuring in a natural stream. Differences in the coefficient of friction and the shape of the particles also play an important part in this stratification. Large dirt particles tend to concentrate at the bottom of the trough, while the lighter and smaller coal particles seek the uppermost layers. A mixture of larger coal and smaller dirt particles occur in the intermediate layers. By suitably adjusting the speed of the water current, the upper layers can be made to travel forward at a faster rate than the dirt and thus a separation is effected. The tendency for stratification in a trough washer can not be easily analysed mathematically on account of the number of assumptions that must be made, and such an analysis, will consequently not be attempted in this paper.

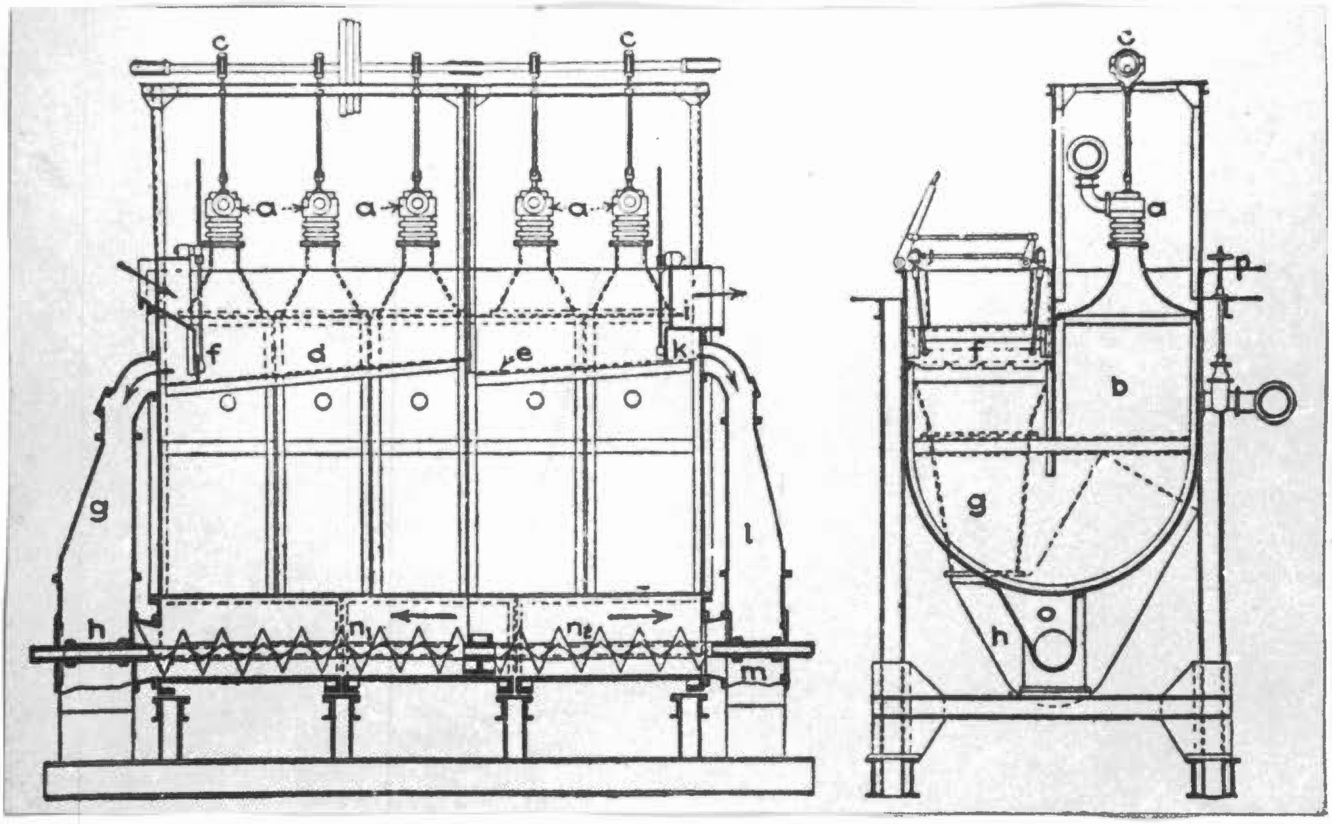

FIGURE 8: DIAGRAMMATIC ARRANGEMENT
OF A BAUM JIG.

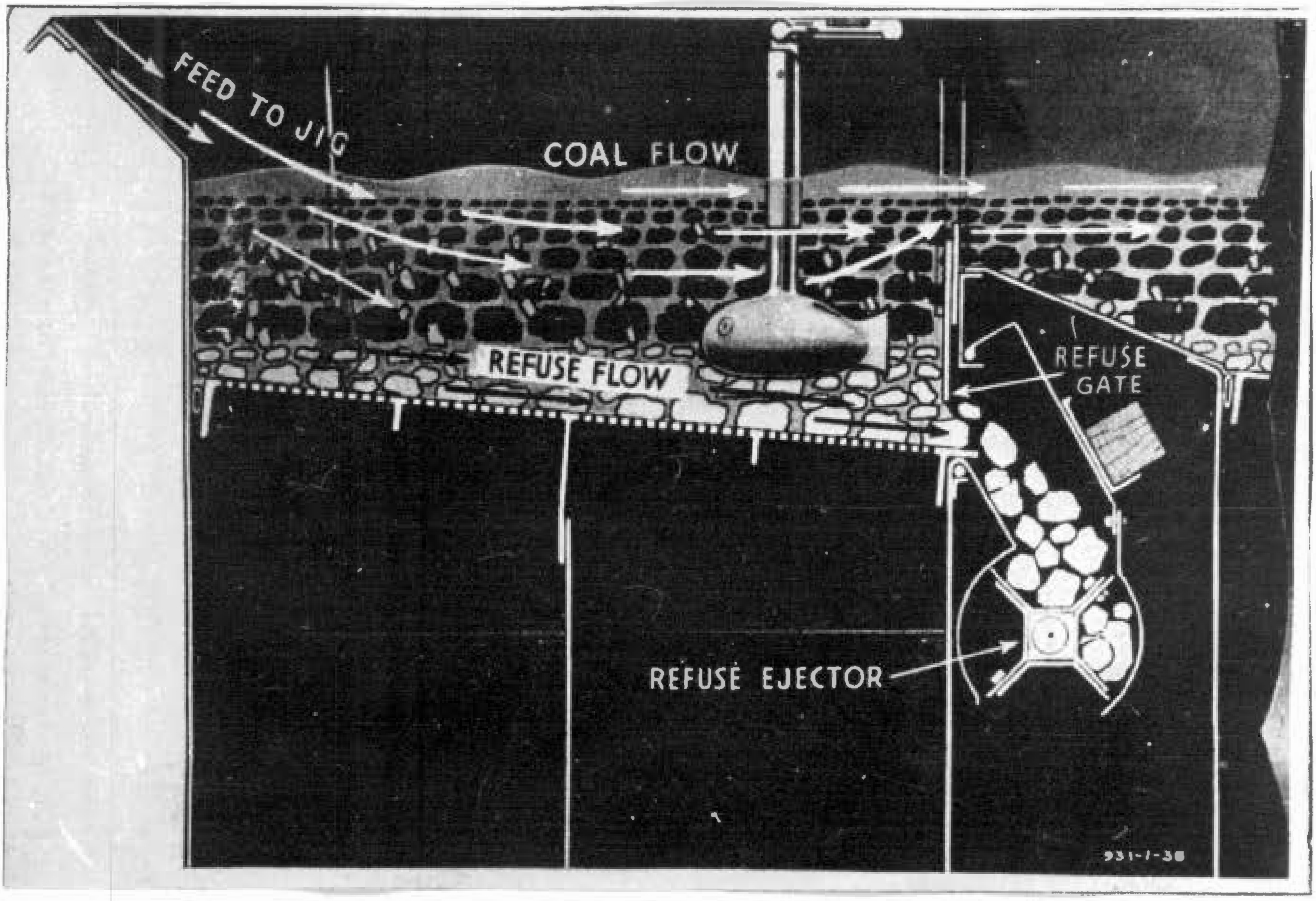

FIGURE 9: JIG FLOAT CONTROL.

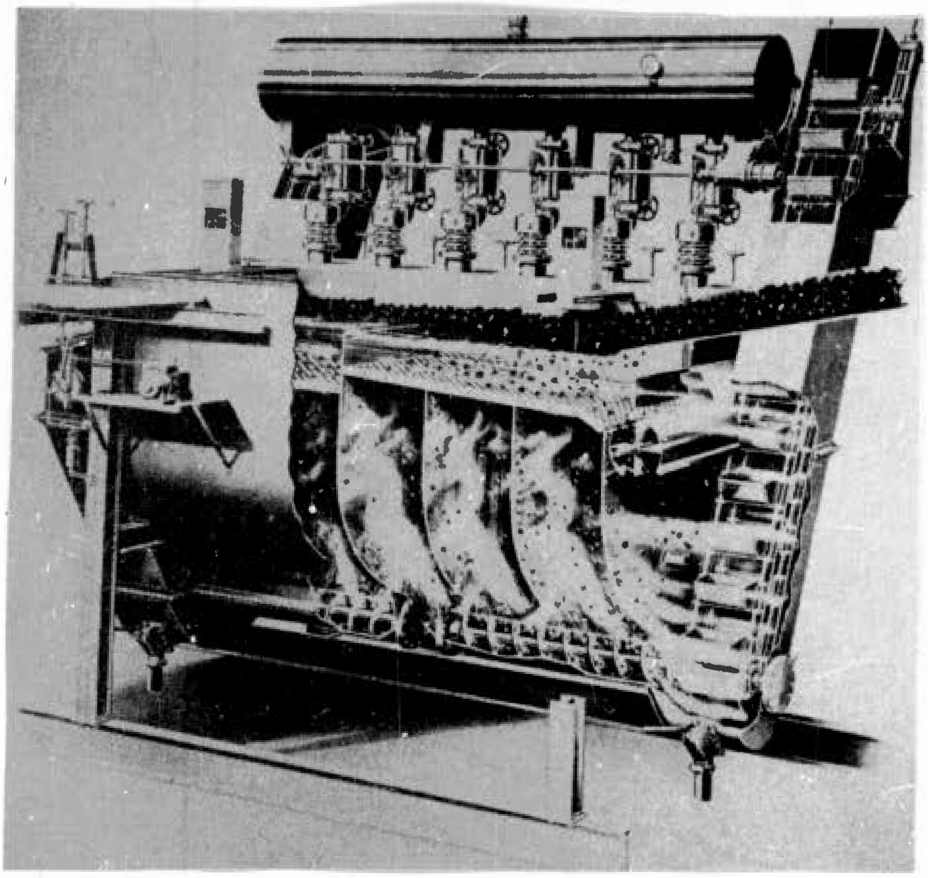

FIGURE 10: BAUM JIG.

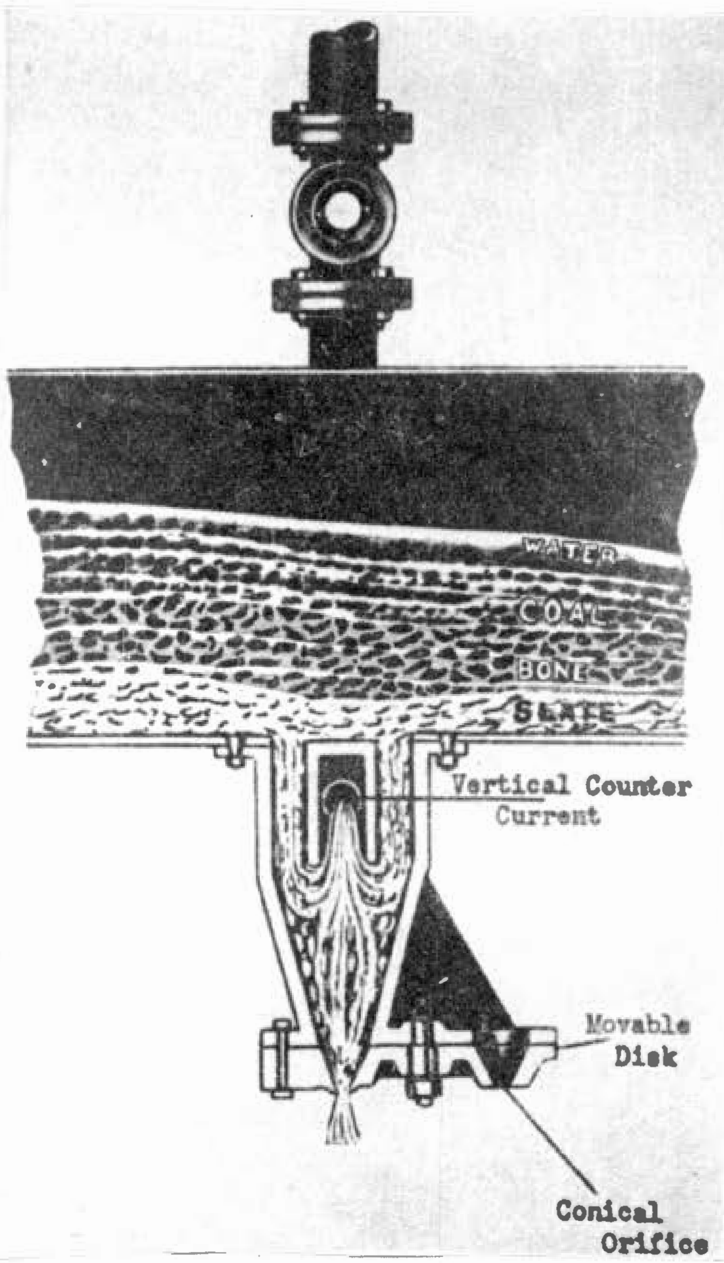

FIGURE 11: SECTION OF FREE-DISCHARGE LAUNDER AND RHEOLAVEUR.

Several washers of this type have been developed over a period of many years and they differ mainly in respect of the methods employed to discharge the refuse. In earlier types, shallow dams were constructed across the trough at various points and the dirt was allowed to accumulate behind them. When sufficient dirt had been deposited, the feed was either shut off or diverted to another trough while the refuse was shovelled off or otherwise removed. As coal tended to be trapped among the refuse, it was necessary to agitate the latter in order to liberate as much coal as possible. In later types, it was attempted to discharge the refuse mechanically, either intermittently or continuously, by the use of moving belts, scrapers, etc., and other more ingenious devices. However, most of these appliances interfered with the stratification and it was not until the Rheolaveur box was invented that continuous mechanical disposal of refuse was successfully accomplished.

As an unduly high percentage of coal tends to be lost in the refuse, provision is made in modern installations to rewash a large proportion of the discard and this gives rise to fairly complex flow sheets.

Trough washers have the advantage that a wider range of sizes can be separated in one operation than in an upward current classifier. Operating costs are usually low, but the capacity per unit area is small.

The Rheolaveur.

The Rheolaveur washer was developed about 1912, in Belgium, by Antoine France. This washer proved so successful, particularly for cleaning the finer sizes of coal, that it was widely accepted and several units are still in operation.

The principal feature of this washer is the method employed to draw off the dirt continuously while preserving the stratification and preventing the loss of coal. An example of the device used for fine coal is shown in Figure 11. Water is supplied to the dirt receptacles and some of it passes upwards through the communicating slots to replace the dirt evacuated, while the remainder passes downwards with the displaced dirt. The water current passing into the trough provides an upward current at the aperture and may be adjusted so as to prevent coal particles from falling and being discharged with the refuse. The discharge orifice at the base of the box is adjusted in accordance with the size and quantity of dirt passing in order to minimise the quantity of water leaving with the refuse. Free discharge of refuse can not be permitted in the case of coarse coal, as the orifices would be large and the quantity of water required would be excessive. This is overcome by allowing the draw off box to discharge into a water sealed dirt chamber which is emptied mechanically.

The actual washing trough is constructed in two or more sections, the inclination of each length diminishing progressively. While the particles are moving with relatively high velocities, they are able to stratify rapidly and on reaching a less steeply inclined portion of the trough, the speed of the water current decreases and the heaviest particles forming the lowest layer are deposited and may be drawn off. In a subsequent portion of the trough the next layer of dirt is deposited and so on. In this way, large dirt, smaller dirt and eventually a mixture of smaller middlings and coal may be drawn off separately.

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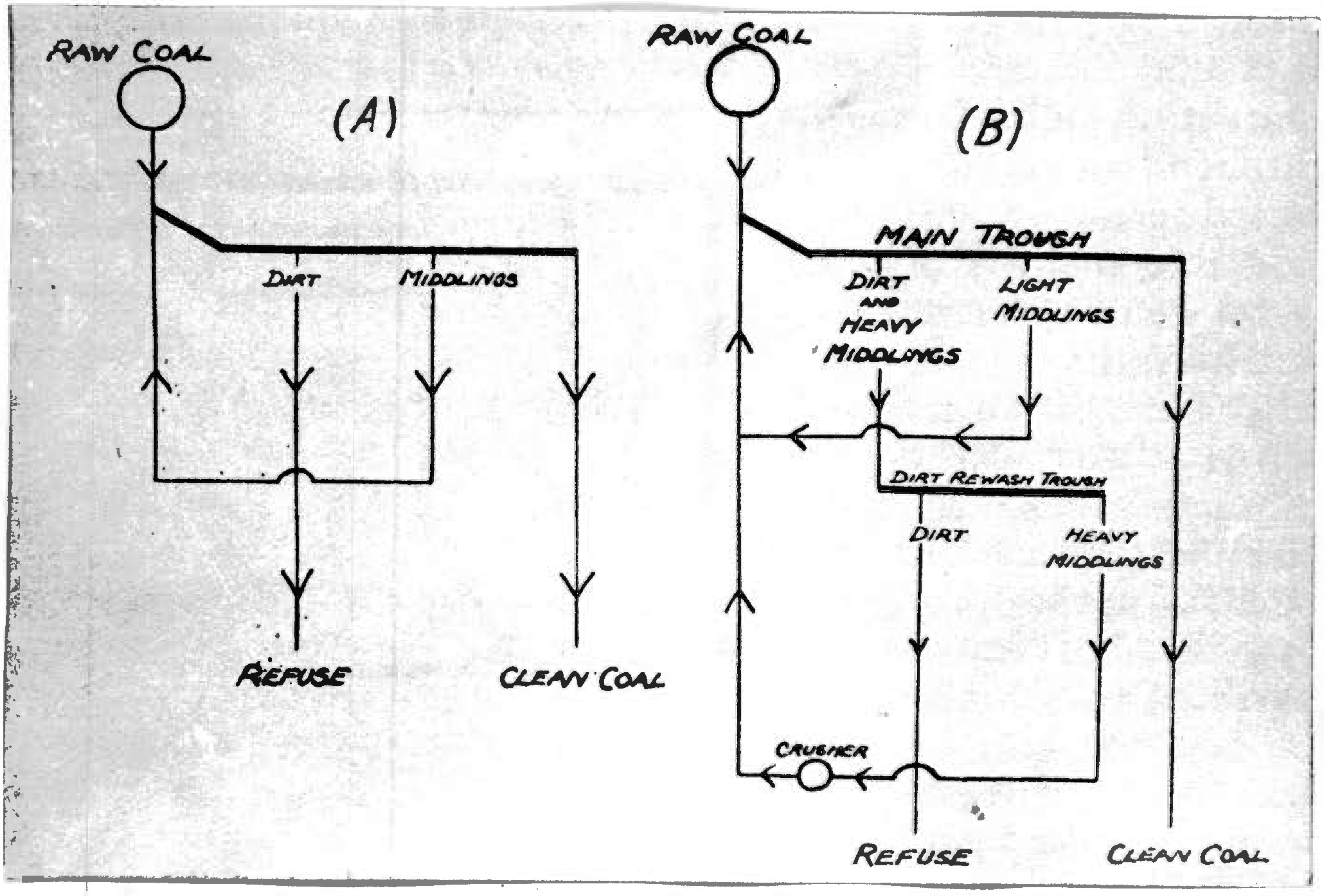

FIGURE 12: FLOW SHEETS; RHEOLAVEUR NUT-COAL WASHER.

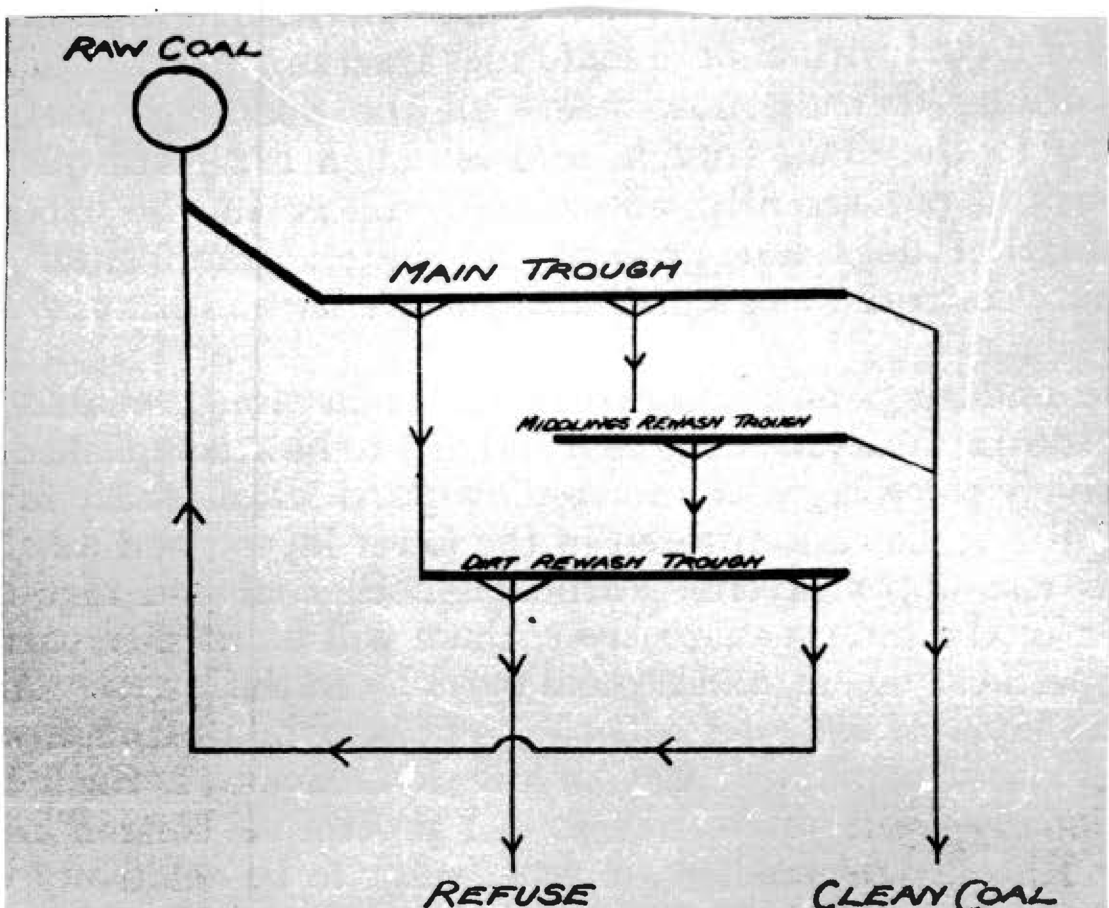

FIGURE 13: FLOW SHEET; RHEOLAVEUR FINE COAL WASHER.

The number and arrangement of troughs and refuse boxes required, depends entirely on the nature of the coal, principally on the size and specific gravity distributions. In the case of coarse coal one trough may be sufficient, while fine coal may require 3 or more troughs. Typical flow sheets for nut coal are shown in Figure 12 and a flow sheet for fine coal in Figure 13. It will be clear from these diagrams, that the process may be divided into two operations, viz.

(1) The removal of all dirt and some coal in order to ensure a product of the desired quality.

(2) Recovery of coal from the refuse.

The recirculation of middlings etc. is an important feature of this type of washery.

In earliest practice, the coal was screened into a number of size fractions and washed in separate troughs. Better results were obtained, however, by washing the coal unsized and then removing fines from the product for rewashing in a fines washer. It is claimed that coal ranging in size from 4 inches to zero can be successfully treated.

Concentrating Tables.

Concentrating tables in various forms have been used extensively for ore-dressing for many years and tables suitable for cleaning fine coal have also been developed. They are similar to trough washers in that they depend for their action on a current of water flowing down an inclined plane. In order to separate the particles after stratification, the table is given a jerking motion, usually in a transverse direction to the flow of water.

The general arrangement of this type of washer is shown diagrammatically in Figure 14 and the principle of operation (29) is as follows.

A particle placed on a horizontal plane which moves slowly backwards and forwards, will remain stationary relative to the plane, provided that the frictional force exceeds the momentum acquired by the particle. If, however, the plane moves to the right, say, with a gradual acceleration and is then suddenly arrested, the momentum of the particle may exceed the frictional force and the particle will continue to move to the right. If the plane is made to return to its original position by a sudden jerk, while the particle is still in motion, the plane is virtually withdrawn from underneath the particle and it will finally come to rest at a point to the right of its original position. By combining and repeating these two movements, the particle may thus be made to travel across the plane.

If the plane is now inclined to the horizontal in such a manner that water will flow in a transverse direction to the reciprocating motion, a particle on the plane will be subjected to two forces at right angles and will travel in the direction of the resultant of these forces. Due to stratification, the lighter particles are uppermost and, since the flow of water has greater influence on them than on the heavier particles forming the lower layers, they are carried away in the direction of flow, while the heavy

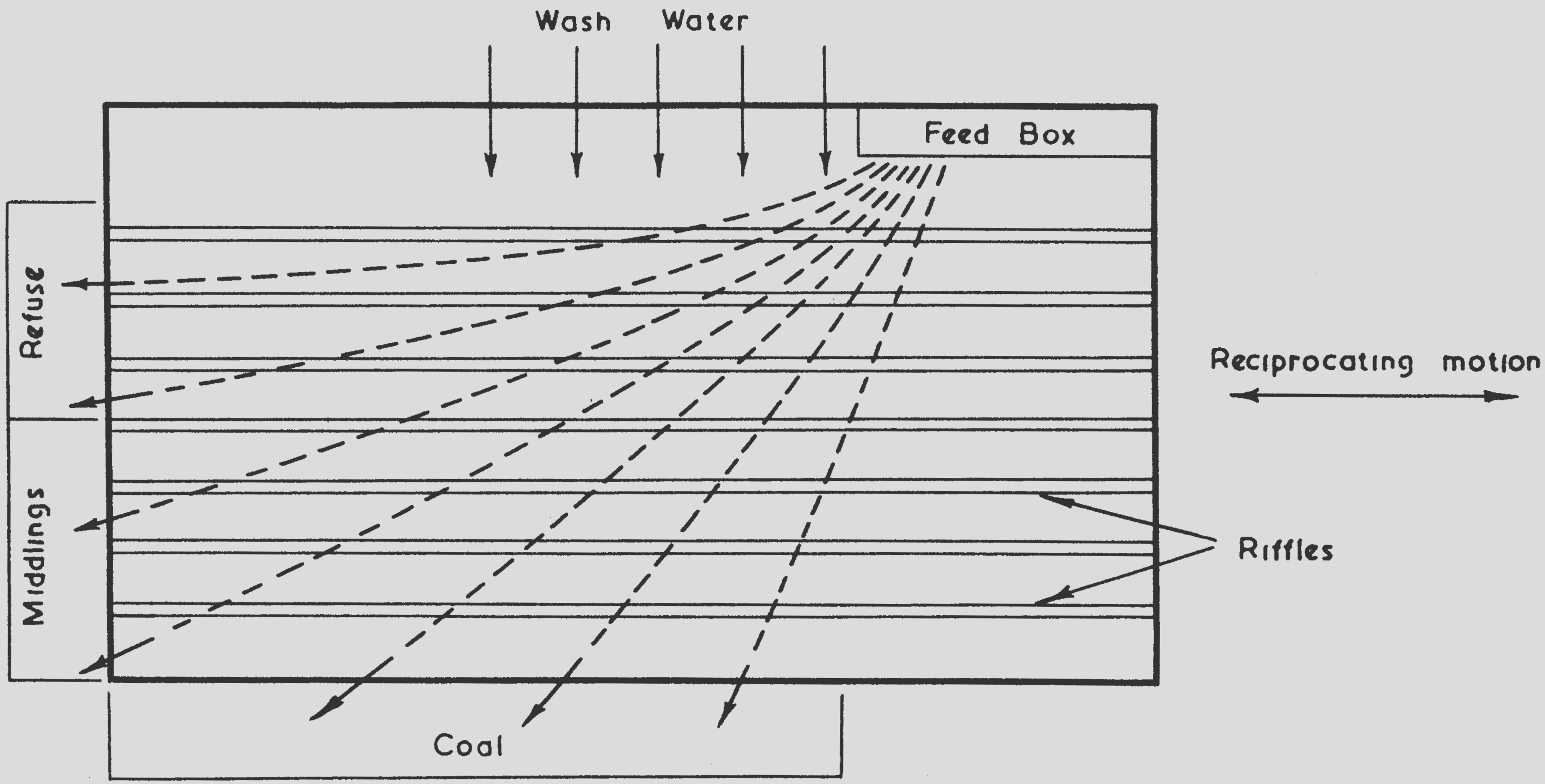

FIGURE 14 Operating principle of a Concentrating Table

particles remain on the table and are made to move by the jerking motion. In this way, the feed particles become spreadout in a fan shape according to specific gravity and size and are discharged from the table as shown in Figure 14. By collecting the overflow over a suitable section of the table any desired product may thus be obtained.

The tables are usually constructed of wood and are covered with linoleum or rubber. Slats of wood, of rectangular cross section, are tacked to the top of the table to form riffles which assist the separation. There are several different forms of concentrating table, which differ mainly with respect to the nature of the reciprocating motion and the arrangement of the riffles.

Concentrating tables are suitable for treating minus $\frac{1}{4}$ inch coal and are effective down to about 100 mesh. However, if there is a considerable amount of near gravity material, cleaning is not always satisfactory.

Heavy-Media Washers.

It has already been pointed out that separation in upward and downward currents of water are not according to density differences only, but are also influenced by the size and shape of the particles. By suitably regulating the water currents in a classifier and the stroke of a jig, it is possible to increase the size ratio which can be treated to a considerable extent. It will be clear, however, that a particularly sharp separation according to specific gravity can not be expected unless the influence of other factors such as size etc. can be eliminated to a large degree.

A possible solution to this problem was to immerse the raw coal in a liquid of intermediate specific gravity between that of coal and dirt. Owing to the high cost of suitable heavy liquids, and other difficulties, little headway was made with commercial processes of this type, until it was found that suspensions of finely ground solids in water behaved as heavy liquids in many respects. Thus, an artificial liquid of any required specific gravity could be obtained readily by adding a suitable quantity of solids (or heavy medium) to water and the heavy medium adhering to the clean coal and refuse could be recovered easily by rinsing these products on screens. The possible use of such a "heavy liquid" led to the development of the socalled "Heavy Medium" or "Float and Sink" processes.

The Chance process, patented in 1917, was the first washer of this type to be used commercially and it is probably still the most popular. Several different processes employing the same principle have been developed in recent years. These differ only in mechanical arrangement and in the material used as heavy medium. Although each of these processes has its own particular advantages, there is probably little to choose between them as regards efficiency.

The great accuracy of the separations achieved is one of the chief advantages of heavy medium washers and has led to their adoption on an increasing scale during recent years. This feature is of great importance in the case of South African coals, which are, in general, very difficult to wash. Other advantages are the elimination of costly hand-picking for the larger coal and the great uniformity of quality, which characterises the clean pro ucts due to the close limits within which the separating specific gravity can be maintained. The feed rate can also be varied widely without loss of efficiency. Depending on the heavy medium employed, any specific gravity of separation between 1.25 and 3.4[30] can be obtained.

The top size of the coal is usually limited by the design of the device employed to dispose of the refuse settling in the separating vessel. Some of these washers can deal with minus $3\frac{1}{2}$ inch coal, while others can handle coal up to 8 inches and even 12 inches in size. Although the usual types of heavy medium washers can clean coal effectively down to $\frac{1}{8}$ inch (in some cases $\frac{1}{16}$ inch), they are usually applied to coal above about $\frac{1}{2}$ inch[31]. The lower limit of size is imposed by the fact that the velocities of small (and especially near gravity) particles is very low under static conditions and the viscosity of the suspension increases this difficulty.

This limitation has been overcome recently with the development of the cyclone washer which is essentially a heavy medium washer employing large centrifugal forces to aid the separation of small particles. The cyclone washer is able to effect a separation of coal ranging in size from about $\frac{1}{2}$ inch to 100 mesh with efficiencies almost equal to those obtained with the conventional heavy medium washers treating coarse coal. It will be clear, then, that a combination plant consisting of a cyclone and a conventional heavy medium separator will be able to wash all coal ranging in size from about 12 inches to 100 mesh and, as will be shown later, with almost theoretical efficiency.

Before describing typical plants, it is necessary to discuss the main properties of suspensions in order that the problems involved in the selection of a suitable heavy medium and the limitations of the various processes may be appreciated.

Heavy Medium Suspensions.

Factors affecting the choice of a heavy medium include cost, availability, resistance to abrasion, resistance to corrosion, chemical inertness and shape of particles. In addition, an ideal suspension should have the following properties:-

(1) Stability or low settling rate, i.e. requiring the minimum agitation to maintain a constant specific gravity.

(2) Low viscosity, especially if small sizes of coal are to be washed.

(3) Constant effective specific gravity for a wide range of coal sizes.

(4) Easy recovery for re-use after dilution i.e. particles should have a high settling rate.

(5) Medium should be easily freed from coal slime.

As will be shown, most of these properties are conflicting and the suspensions used in all commercial heavy medium processes consequently represent a compromise between the opposing requirements.

DeVaney and Shelton[32] investigated the influence of the specific gravity of suspension on the viscosity and settling rate for various heavy media (graded -200 + 325 mesh). The results of this work are shown graphically in Figures 15 and 16. It will be clear from these curves that the viscosity increases very rapidly when a certain critical specific gravity of suspension is exceeded for each medium. It will also be noted that the critical suspension specific gravity depends on the specific gravity of the material, the higher the specific gravity of the medium the higher is the critical specific gravity. Thus, quartz (S.G. 2.65) would be suitable for separations up to about 1.5 specific gravity, magnetite (5.18) up to 2.4, ferrosilicon (6.8) to 3.1 and so on.

The settling rate, on the other hand, decreases with increasing specific gravity of pulp, and the suspension becomes very stable when the critical specific gravity is exceeded, i.e. when the settling rate is low, viscosity is high. It will also be seen that the settling rate increases with increasing specific gravity of medium for any given suspension specific gravity. Thus, a quartz suspension of 1.5 specific gravity would be more stable than a magnetite suspension of the same specific gravity, but its viscosity would be higher. The higher settling rate of the magnetite would be an advantage, however, when the suspension has been diluted and the medium particles have to be recovered for re-use, as a smaller thickening tank would be required.

In order to determine the influence of particle size on viscosity and settling rate, DeVaney and Shelton carried out the tests indicated in Figures 17 and 18. These curves show that the critical specific gravity of suspension increases with increasing particle size, the settling rate, however, also increases. Thus, the use of coarse particles would decrease the viscosity at the expense of stability.

Hirst[33] found that the effective density of a suspension was not constant, but increased with decreasing feed particle size. This relationship is given by the expression

$$\sigma = \rho + (s - \rho) \left(\frac{d}{D}\right)^{1 - \phi}$$

where σ = effective density of the suspension with respect to a particle of diameter D,

ρ = density of the suspension.
s = density of the heavy medium particles.
d = diameter of the medium particles.
ϕ = -1 for medium particles obeying Stoke's Law.
= -0.5 for medium particles obeying Allen's Law.
= 0 for medium particles obeying Newton's Law.

As an example, the effective specific gravity of a suspension of 48 to 60 mesh sand having a specific gravity of 1.40 is practically constant at 1.40 for feed particles down to about 3/8 inch in size, but is 1.48 for $\frac{1}{16}$ inch particles.

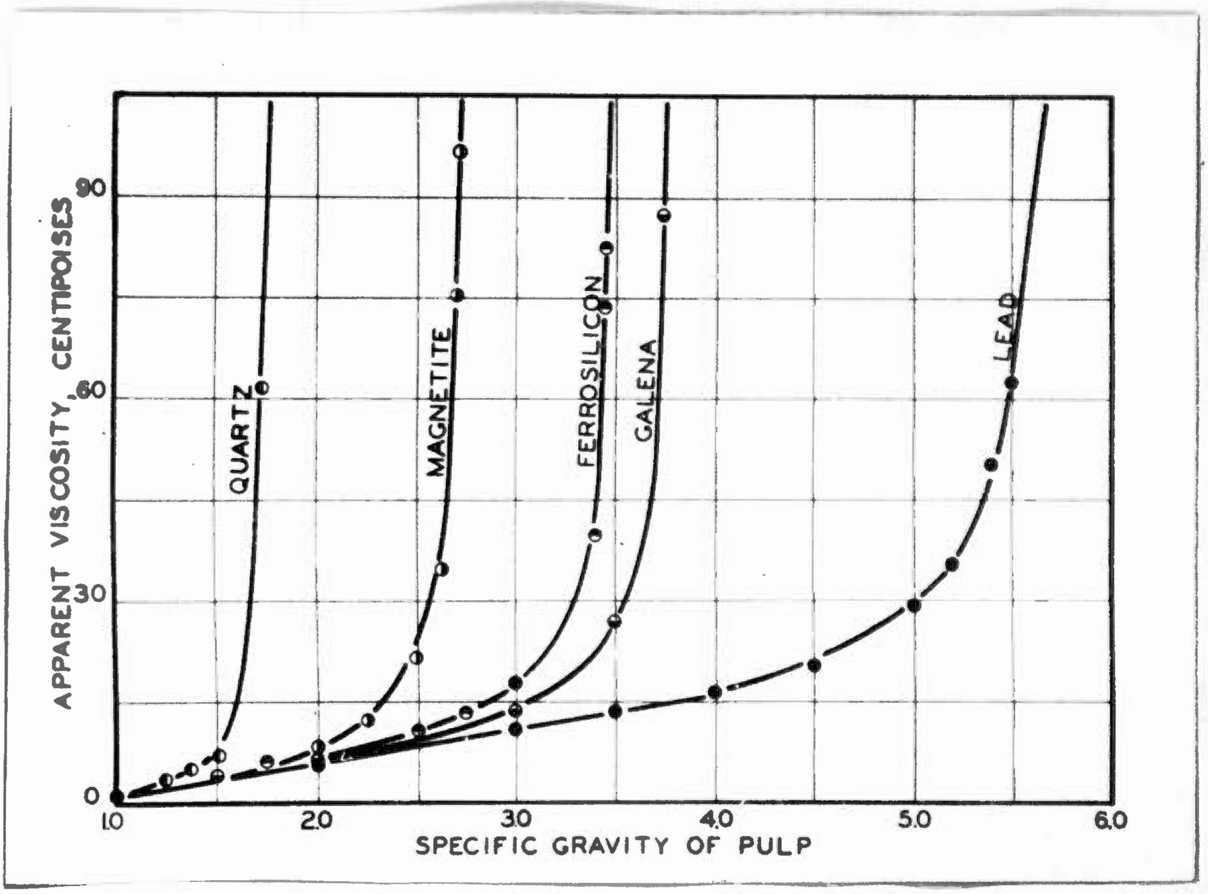

FIGURE 15: EFFECT OF SPECIFIC GRAVITY OF PULP ON CONSISTENCY OF SEVERAL SUSPENOIDS (-200 + 325 MESH).

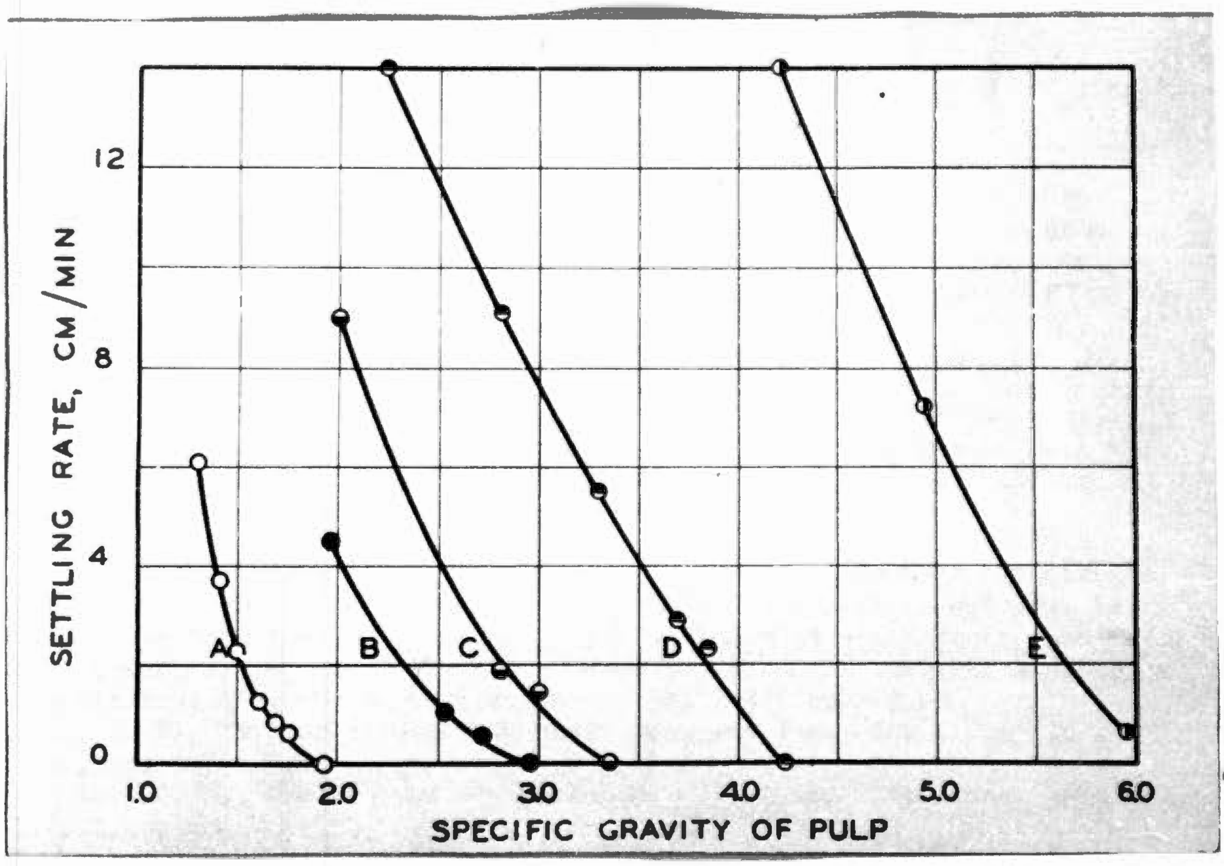

FIGURE 16: EFFECT OF SPECIFIC GRAVITY OF PULP ON SETTLING RATE:

A = Quartz; B = Magnetite;

C = Ferrosilicon; D = Galena;

E = Lead.

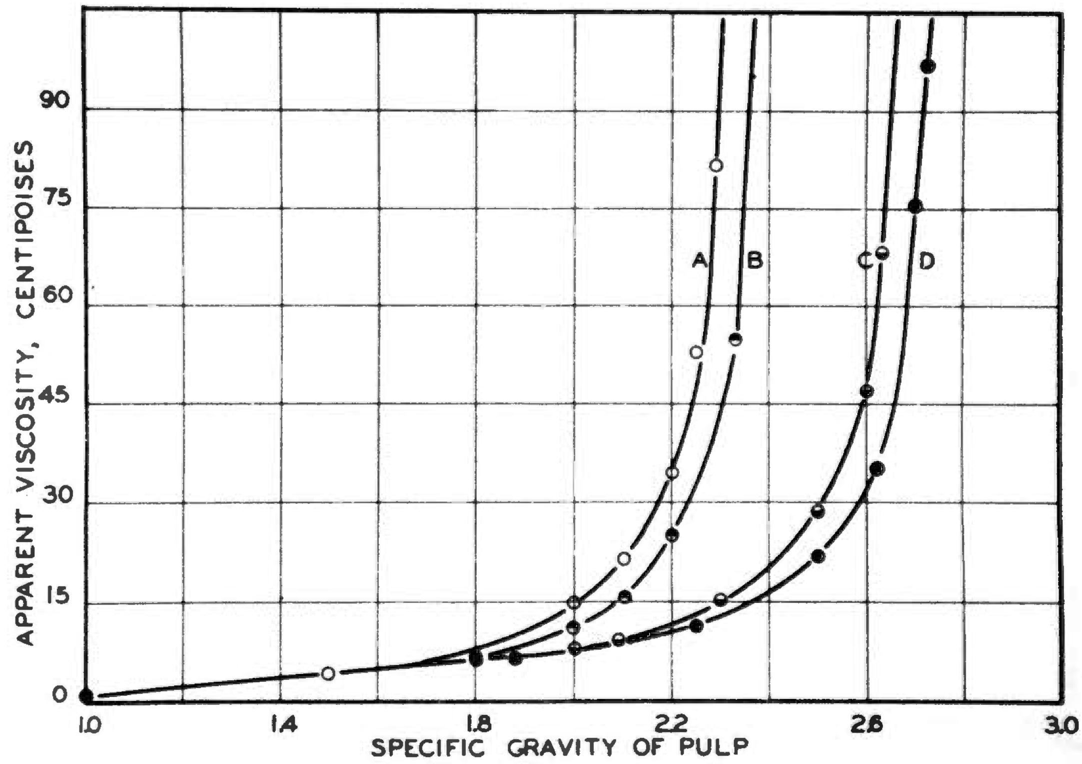

FIGURE 17: VISCOSITY CURVES FOR MAGNETITE SUSPENSIONS.

(Average diameter in Microns)
(A=15.8; B=25.7; C=37.6; D=51.7)

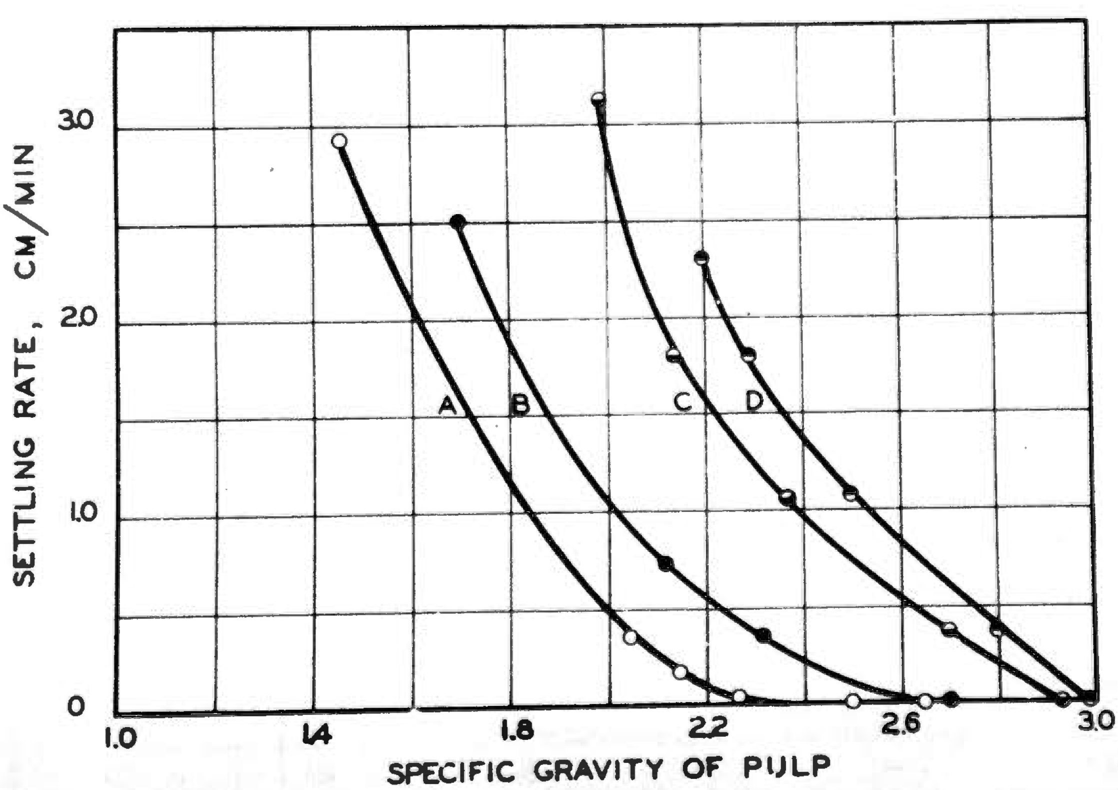

FIGURE 18: SETTLING RATE CURVES FOR THE MAGNETITE SUSPENSIONS SHOWN IN FIGURE 17.

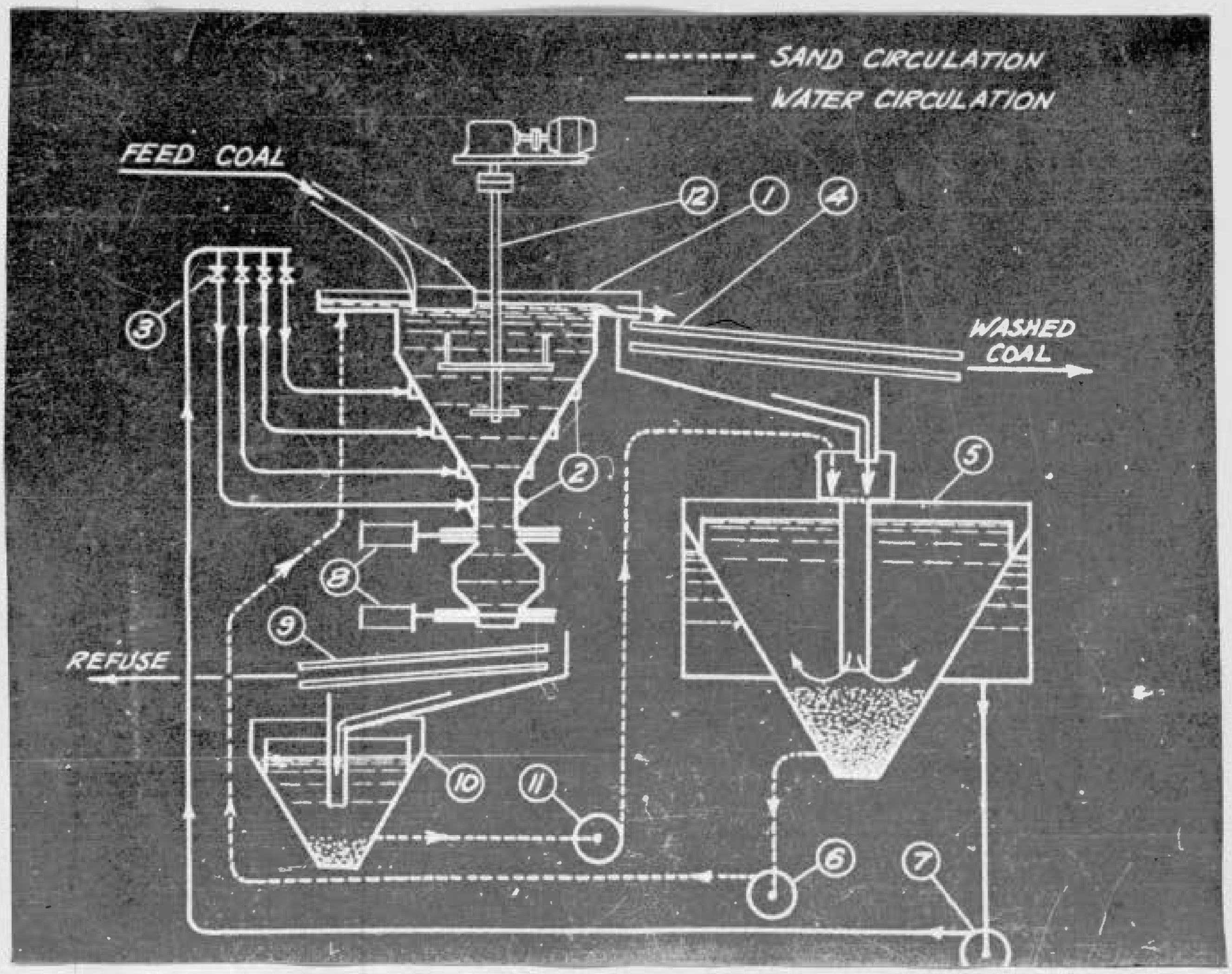

FIGURE 19: DIAGRAM OF THE CHANCE PROCESS.

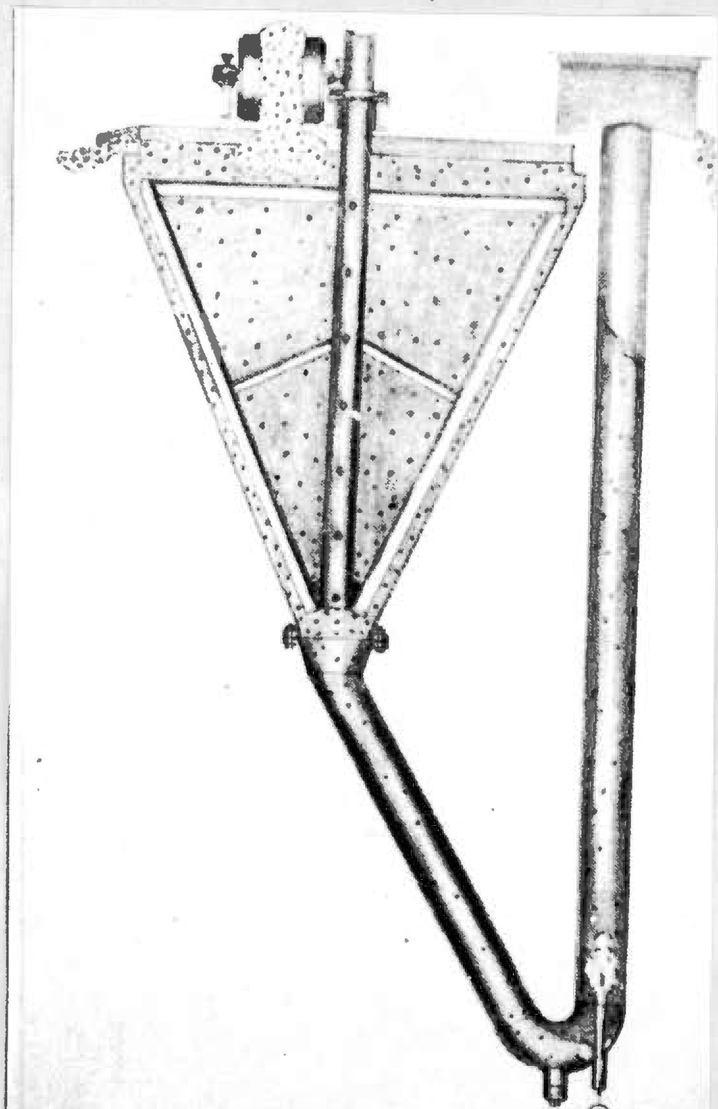

FIGURE 20: CYANAMID SEPARATING CONE

The distribution of the balancing water to each manifold is adjusted to suit the grading of the sand and a nearly constant specific gravity can be maintained. Hydraulic classification takes place in the cone to a certain extent and influences the separation of small feed particles. Although the efficiency of separation falls off with decreasing particle size, coal down to about $\frac{1}{16}$ inch can be treated satisfactorily in most cases. The top size which can be washed is not clear from the literature, but reference is made to cases where minus 6 inch coal was fed to a Chance plant.

When two saleable products are required (i.e. clean coal, middlings and refuse) the separations are usually effected by two Chance cones arranged in series, the first cone being operated at the higher specific gravity. However, if the quantity of middlings does not exceed 15% of the input, the separations may be effected in a single cone fitted with a middlings column. This is achieved by adjusting the balancing water to give two zones of specific gravity in the cone. It is claimed that a difference of 0.2 specific gravity between the two zones can be maintained. It is also possible to shut down the middlings column and revert to single gravity separation when required.

The Chance washer has proved so efficient and inexpensive to operate that many are in use at present to deal with those coals which are normally too difficult to wash in jigs etc.

It will be clear that the Chance process really occupies a position intermediate between an upward current classifier, such as the Robinson washer, and true dense medium processes in which the separation is virtually static.

Cyanamid Heavy Medium Washer[36][37]

The Cyanamid heavy medium process was first used for concentrating metalliferous ores in 1939 and was applied to coal cleaning somewhat later (about 1944). This process employs finely ground magnetite or ferrosilicon (or mixtures of these) as heavy medium, depending on the specific gravity of separation required. The advantage of these media lies in the fact that their magnetic properties provide an easy and, it is claimed, inexpensive method of reclaiming medium which has been contaminated by coal slime, clay etc. Suspensions prepared from these materials, when finely comminuted, are stable and require little agitation to maintain constant density. Almost static separations are, therefore, possible. Magnetite is suitable as medium for separations between 1.25 and 2.20 specific gravity. Mixtures of magnetite and ferro silicon are used for separations in the range 2.20 to about 2.85, and ferrosilicon alone for specific gravities of 2.85 to 3.4. Magnetite is used in coal cleaning as the specific gravity of separation rarely exceeds 1.6 or 1.7. This material is a naturally occuring mineral and is found in abundance in many countries including South Africa.

One type of separating cone is shown in Figure 20. Clean coal overflows at the top, while refuse sinks and is discharged continuously (together with suspension) by means of an air lift. Since the largest particle which can pass through an air lift column, without causing obstruction, is about $\frac{1}{3}$ its diameter, the maximum size of feed which has been found to be practicable is about $3\frac{1}{4}$ to $3\frac{1}{2}$ inches. The cone is equiped with slowly revolving sweeps or rakes. These rakes serve three purposes; (a) refuse and magnetite are prevented from settling on the cone wall, (b) the bath is made to revolve slowly thus carrying float coal to the overflow, (c) they assist in maintaining a uniform suspension. Maintenance of a constant density is further aided by slow circulation of the suspension through the cone and air lift.

A typical flow sheet of a plant designed to wash feed from which excessive fines have been removed is shown in Figure 21. The clean coal and refuse is discharged on to two sets of vibrating screens, (4) and (5), where about 90% of the entrained suspension is drained off into a common sump and returned to the separating vessel(2). The remainder of the adhering medium is removed by water sprays and the finished products are discharged. Diluted suspension from the washing screens is passed between magnetising coils(7) to a thickening tank(8). The medium particles become mutually attracted on account of the magnetic charge they have acquired and floculate (i.e. form adhering groups of particles). In the floculated condition, the medium settles rapidly and a smaller thickener is required than would otherwise be the case. The thickening tank also serves as storage space for medium and can take care of variations in the amount of medium in circulation, due to alteration of the separating specific gravity.

Overflow from the thickener is used as spray water, while the underflow passes through magnetic separators(11)(12) for removal of non-magnetic slimes. The specific gravity of the cleaned suspension is controlled by a densifier(13) from which it is returned to the separating cone via demagnetising coils(14) to disperse the medium particles.

In cases where unsized feed is washed, fine coal and refuse particles pass through the drainage and rinsing screens and may be recovered separately, if desired, by providing twin medium cleaning circuits.

It is claimed that this washer can clean coal effectively from $3\frac{1}{2}$ inches down to about 20 mesh. The loss of magretite is comparatively low, being of the order of $\frac{1}{2}$ lb. per ton of raw feed. This figure varies with the size grading of the feed and may be as high as 1 lb. per ton or higher for finer feeds.

Link Belt Washer[38]

The Link Belt washer is of comparatively recent origin and, as far as can be ascertained, only a pilot plant of 110 tons per hour capacity has been in operation. The American Cyanamid Company are also the technical and sales representatives for this washer. Magnetite is used as heavy medium, the grind consisting of approximately 100 per cent minus 50 mesh of which 70 per cent is minus 325 mesh.

A description of this washer is given as it is an example of a separating vessel which is not conical in form. Raw coal enters one end of a horizontal revolving drum which is

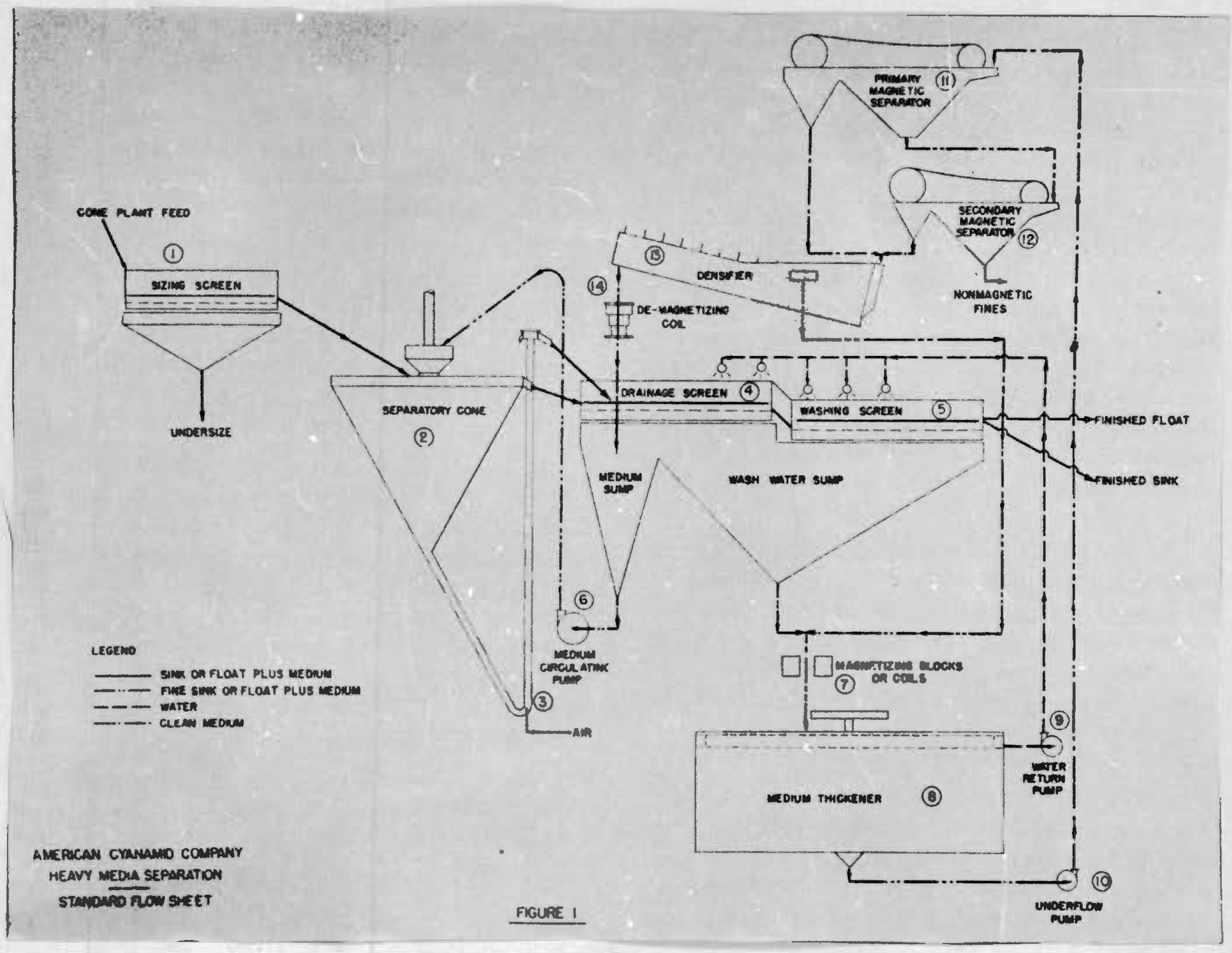

FIGURE 21: FLOW SHEET OF CYANAMID HEAVY MEDIUM PROCESS.

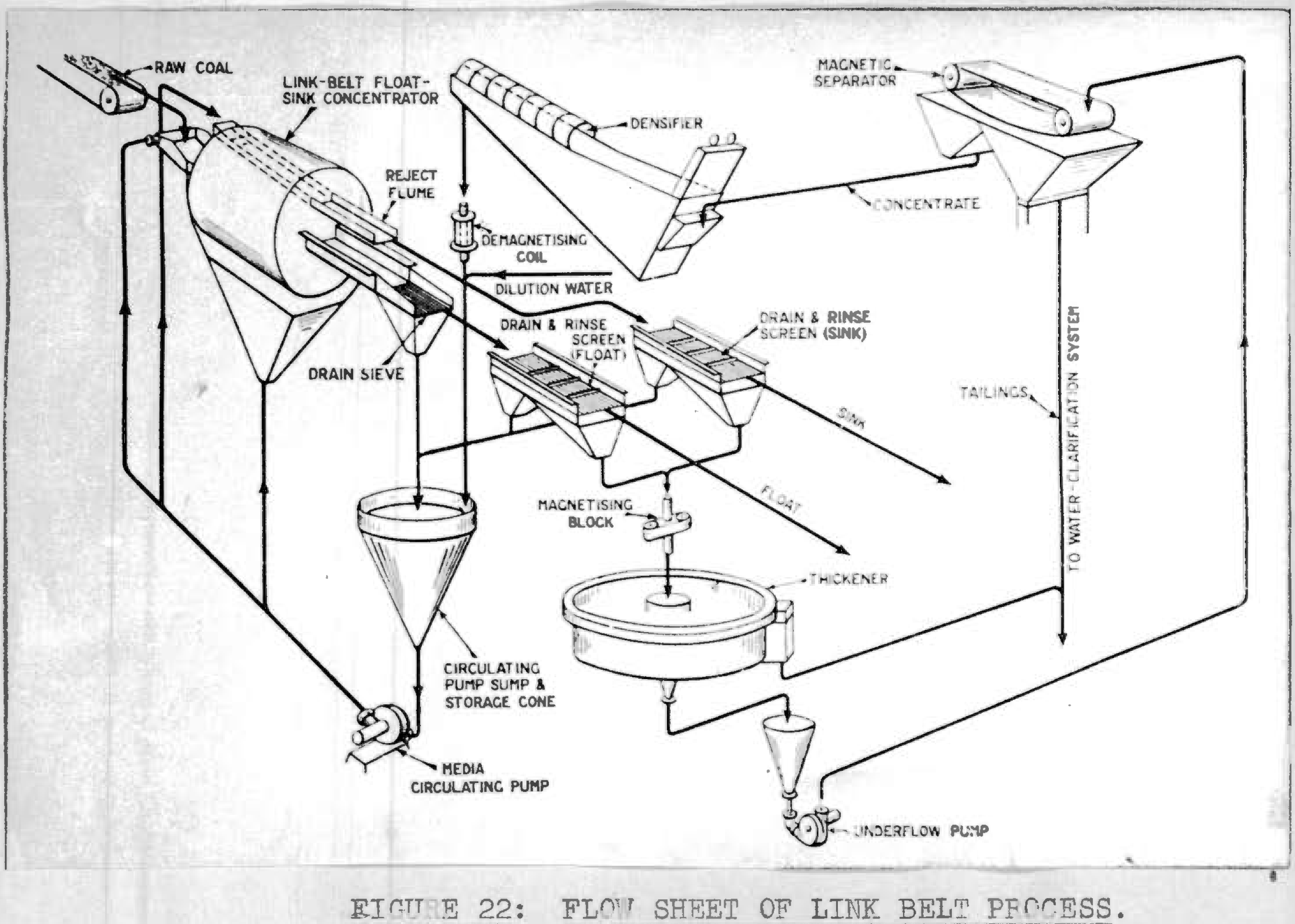

FIGURE 22: FLOW SHEET OF LINK BELT PROCESS.

half immersed in a bath of medium. Clean coal floats through the drum and flows over a weir. The refuse sinks to the bottom of the drum and is elevated by perforated lifting compartments attached to the inside of the drum. As the refuse is elevated above the bath most of the entrained suspension drains off. Near the top of the drum, the refuse falls off the lifters into a flume or launder and is discharged hydraulically to vibrating screens. Skirt boards project into the bath of medium and prevent the contamination of refuse with clean coal.

A flow sheet of a typical plant is shown in Figure 22. The magnetite recovery circuit will be recognised as being similar to that already described.

It is claimed that this washer can successfully handle coal ranging in size from 8 inches to $\frac{3}{16}$ inch. The top size is only limited by the size of the compartments in the elevator and these can be increased in size if desired. Medium consumption is also about $\frac{1}{2}$ lb. per ton of feed.

Froth Flotation.

All the washers which have been described so far, depend principally on differences in specific gravity to effect a separation of coal from dirt. Froth flotation is the most important of the processes which rely on physical properties other than specific gravity for their action.

This process was first applied to fine coal about 1920 and was soon widely accepted, particularly in Europe. Froth flotation has not yet been used for coal cleaning in South Africa, the reason probably being that it would be uneconomical in view of the high operating cost as compared with the relatively low value of coal in this country. If, however, this process were used to produce a saleable product from material which would otherwise be waste, it may possibly be of great value. In the past the demand for fine coal has not even approached the normal production of natural fines, and their market value was low so that there has been no incentive to clean fine coal, particularly if the process is expensive. As more uses are found for fine coal, however, and the value increases, froth flotation will undoubtedly come into the picture.

In this process, advantage is taken of differences in the surface wetting properties of coal and refuse to effect the separation. Thus, shale is more readily wetted by water than is coal, and certain oils have a greater affinity for the latter. Therefore, if a suitable oil (or collector) is added to a suspension of raw coal in water, the oil will coat the coal particles and make them water-repellent, while shale particles, etc., will be wetted by the water. If bubbles of air are now passed through the pulp, the oil-coated coal particles will tend to adhere to them and will be raised to the surface, while the shale particles remain in suspension. By adding suitable reagents (or frothers) the

air bubbles/....40.

air bubbles are reinforced and, instead of bursting, remain on the surface of the water in the form of a froth to which the coal is attached. This coal-bearing froth can then be scraped off and dewatered to recover the coal. A similar technique may be employed to separate bright coal from dull coal.

Frothers commonly used in coal flotation are cresylic acid, pine oil and certain alcohols. Collectors used are low-grade petroleum fractions, such as partly refined fuel oil or lubricating oil, and paraffin. The governing factors in the selection of reagents are the quantity necessary for the process and the price, both of which govern the overall reagent cost. Reagent consumption varies considerably but is usually of the order of 1 or 2 lb. per ton of raw coal.

The maximum size of coal which can be treated is governed by the size of particle which can be lifted to the surface by the air bubbles, and is usually about $\frac{1}{8}$ inch.

Although the various makes of froth flotation plant differ considerably in design and operation, the essential features are the same, viz. provision for agitation and aeration of the pulp.

Denver Flotation Machine (39)

One cell of a Denver "Sub A" flotation machine used for ore separation is shown in Figure 23. The pulp flows into the cell by gravity through a feed pipe and drops on top of a rotating impeller situated below a stationary hood. Suction created by the impeller draws air through a standpipe surrounding the impeller shaft, and the pulp is thus thoroughly aerated when it is discharged into Zone 1 of the cell. In the central zone, the pulp is quiet and the mineral laden bubbles separate and rise to the surface, while middlings are recirculated to the agitation zone. The froth accumulates in zone 3 and is removed by rotating paddles. The machine used for fine coal is the same in principle, but has a double froth discharge on opposite sides of the cell.

A complete flotation plant consists of a number of such cells usually arranged in series, the separation being effected in stages. Thus, the froth from the first cell would be composed of the highest quality coal and would become progressively poorer for succeeding cells. Froth from the end cells is often recirculated to the first cell for re-cleaning.

The froth flotation process is not equally applicable to coals of every rank. Coals of high oxygen content and low rank are more difficult to treat and when they can be treated tend to require an excessive amount of reagent.

Miscellaneous Processes.

There are several cleaning processes which depend on physical properties such as shape, coefficient of sliding friction, resilience, friability, electrical conductivity, etc., or on combinations of some of these properties and specific gravity. The majority of these devices are limited to the coarser sizes of coal and the separation is not particularly

sharp/......41.

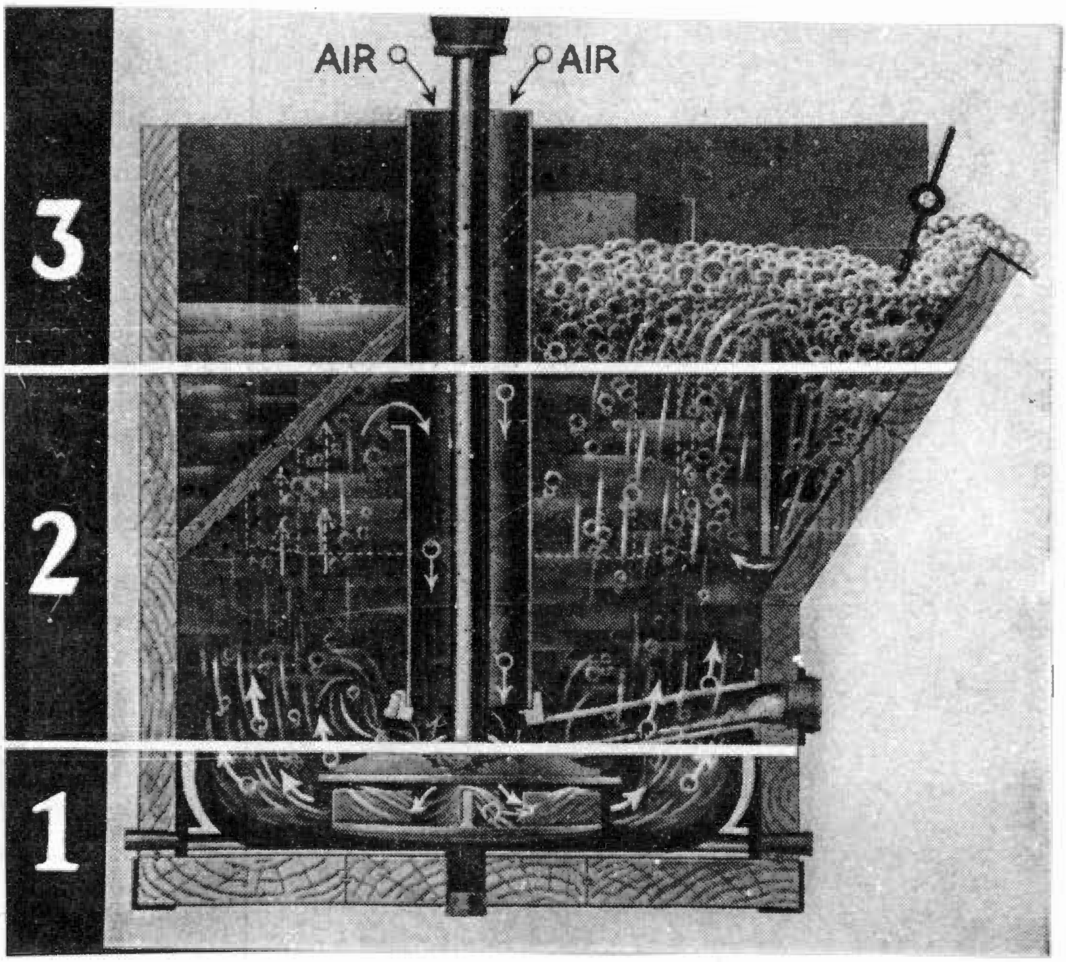

FIGURE 23: DENVER "SUB.A" FLOTATION CELL.

sharp in most cases. Although they may be of importance in certain specific cases, their application is rather limited. A description of the various types of separator included in this class will, therefore, not be attempted in the present paper. Processes employing the electrical properties of coal and dirt are of some interest, however, as they may possibly be of importance for fine coal cleaning in the future.

An electrostatic cleaner[40] was erected in Germany during the second World War for the production of ultra clean coal suitable for the manufacture of electrodes. The principle of operation is illustrated in Figure 24. Dry coal is fed onto a an electrically charged roll and is then allowed to fall past an electrode, the potential difference being 30,000 to 50,000 volts. Particles of high ash content are deflected towards the electrode, whereas coal particles fall vertically. A series of rolls and electrodes, arranged as shown, complete the separation. Although this plant had not yet been operated in 1948, preliminary pilot plant tests indicated that coal of 1.0 to 0.1 millimetre could be treated satisfactorily.

The Johnson process[41], although somewhat different in arrangement, employs the same principle. This process can handle coal up to 3/8 inch, but feed graded minus 1/8 inch plus 200 mesh is preferred. Internal moisture does not affect the process but the feed should be surface dried. It is claimed that the plant is foolproof and easily controlled and that operating costs are low.

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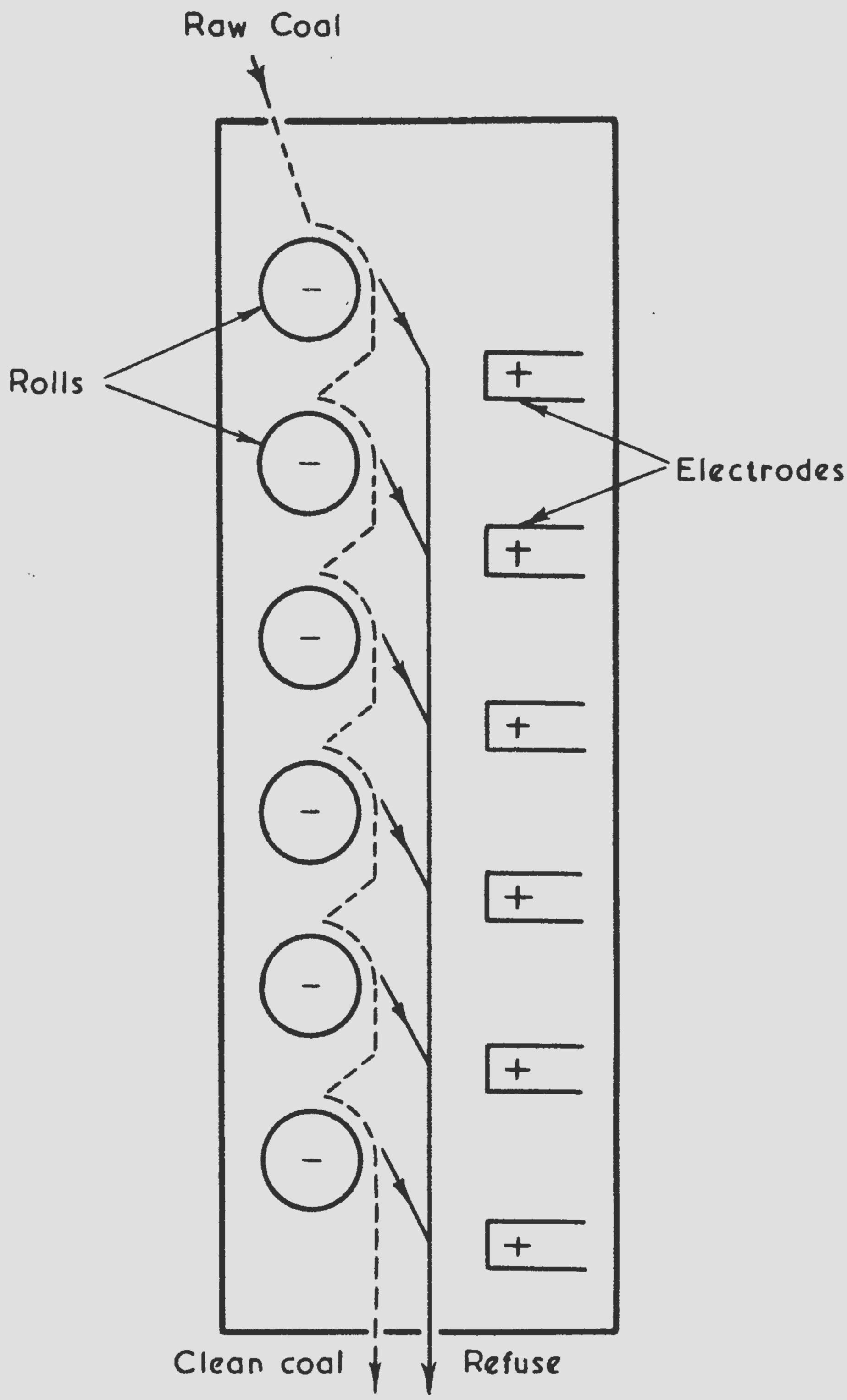

FIGURE 24 Diagram of an electrostatic cleaner

ASSESSMENT OF THE EFFICIENCY OF A WASHING PROCESS.

In order to determine whether the operation of a washer is satisfactory as regards the separation it is required to effect, it is necessary to analyse the results obtained and deduce from these the efficiency of the process. However, as coal is a heterogeneous substance, the basis on which the efficiency should be assessed is not self evident, and has consequently given rise to many, and in some cases, conflicting opinions as to the best procedure. It is not proposed to describe and discuss in detail the various methods of evaluating washery performance which have been suggested from time to time. Only typical examples of those used in practice will be given together with the author's views on this subject.

The methods commonly used to interpret washery data may be divided broadly into 2 groups:-

(a) Formulae which reduce the washery data to a single figure representing the degree of effectiveness of the recovery of clean coal and elimination of refuse from the feed.

(b) Graphical representations of washery performance.

Although the various efficiency formulae are still used extensively, there has been a tendency in later years for graphical methods to increase in popularity.

Efficiency Formulae.

In the various efficiency formulae employed, the relative proportions of clean coal and refuse actually obtained from a washer or the ash contents of these products (or combinations of these) are compared with corresponding values obtained by float and sink analysis of the feed, the latter being regarded as representing an ideal separation.

For simplicity, the following symbols will be used to define the formulae:-

Ya = actual yield of washed coal, as percentage of feed.

Yt = float in feed coal at the S.G. of separation, percent.

Yaf = float in washed coal at the S.G. of separation, as percentage of washed coal.

Ra = actual refuse, as percentage of feed.

Raf = float in actual refuse at S.G. of separation, as percentage of refuse.

Rt = sink in feed at S.G. of separation, as percentage of feed.

Yas = sink in washed coal at S.G. of separation, as percentage of washed coal.

Aa = ash content of washed coal, percentage of washed coal.

At = ash content of float in feed at the S.G. of separation.

Ar = ash content of actual refuse.

As = ash content of sinks in feed at S.G. of separation.

Af = ash content of feed.

Drakeley Efficiency Formula[42]

This formula has been used extensively in practice to estimate the efficiency of a washing operation, and is composed of two terms viz., a qualitative and a quantitative efficiency factor. The former measures the improvement in the washed coal over the raw coal with respect to the elimination of refuse, while the latter is a measure of the efficiency of recovery of the washed coal. These factors are defined as follows:-

$$\text{Qualitative Factor} = \frac{100 \times \left[Yaf - Yt \right]}{100 - Yt}$$

$$= \frac{100 \times \left[Yaf - Yt \right]}{Rt}$$

$$\text{Quantitative Factor} = 100 \times \frac{\left[Yt - \frac{Raf \times Ra}{100} \right]}{Yt}$$

The overall or general efficiency of the operation is given by the product of these two factors.

The original Drakely formula, shown above, was later modified as follows[43]:-

$$\text{Qualitative Factor} = \left[\frac{Rt - \frac{Ya \times Yas}{100}}{Rt} \right] \times 100$$

$$\text{Quantitative Factor} = 100 - \frac{Ra \times Raf}{100}$$

The relative merits of the original formula and the modified version will be discussed later. This formula is an example of one based only on the quantities of the various fractions obtained by analysis of the feed and washery products.

Chapman and Mott Formula[44]

In view of the practical difficulty involved in obtaining truly representative samples of the washed coal and refuse and in determining the true yields, Chapman and Mott devised a formula which is based only on the ash contents of the different samples, it being considered that the ash contents are less liable to variation and sampling would, therefore, be simplified.

Efficiency of the operation is defined as follows:-

$$\text{Overall Efficiency} = \frac{At}{Aa} \times \frac{Ar}{As}$$

Fraser and Yancey Formula[42]

In the Fraser and Yancey formula, both quantity and quality (ash content) of washed coal is taken into account to determine the efficiency, as follows:-

$$\text{Quantitative Factor} = \frac{Ya}{Yt}$$

$$\text{Qualitative Factor} = \frac{Af - Aa}{Af - At} = \frac{\text{Actual ash reduction}}{\text{Theoretical ash reduction}}$$

Overall efficiency is also given by the product of the two factors.

While it is not intended to enter into a detailed discussion of the relative merits of the formula shown above, attention should be drawn to the limitations of efficiency formulae of the type illustrated.

It will be clear that the specific gravity of separation must be known accurately before the above formulae can be applied. Since the S.G. of separation is liable to variation in many processes, its actual value at the time of sampling must also be determined, and this requires considerably more analytical data than mere inspection of the relevant formula would suggest. The use of an assumed S.G. of separation to eliminate this additional work will, in all probability, give rise to a completely erroneous value for the efficiency.

The main criticism of the various formulae lies in the fact that, with few exceptions, they have no physical significance and it is consequently difficult, if not impossible, to interpret the figure obtained to any greater extent than as a mere indication of increase or decrease of separating efficiency. Material variations in the specific gravity composition of the feed may even nullify this function and the operator will not know whether the performance of the washer has deteriorated and the plant requires attention or otherwise. The term, efficiency, applied to these largely empirical expressions is, therefore, misleading and the author prefers to regard them as "performance factors." These formulae can, however, be of great assistance in washery control when used by competent persons who are fully aware of their limitations.

It can be shown that the value of the efficiency as calculated by means of any of these formulae depends not only on the operation of the washer but also on the specific gravity of separation and the specific gravity composition of the feed coal. It follows, then, that differences in the actual separating ability of two or more washers will only be indicated by the "efficiency" figure if they are washing the same coal at the same specific gravity, i.e. operation under conditions rarely obtained in practice. The necessity for distinguishing between the "mechanical" efficiency of the washer and the efficiency of the operation as a whole (with respect to any given coal) will, therefore, be clear. The author has found it convenient to refer to these as "washer efficiency" and "washing efficiency" respectively, the former being the basis on which different washers should be compared.

Washing efficiency may be determined by means of one of the formulae already mentioned, but the following expression[45] is considered to be of greater practical value, i.e.:

$$\text{Efficiency} = \frac{\text{Actual yield of washed coal}}{\text{Percentage of float coal in the feed having the same ash content as the washed coal.}}$$

The value obtained from this formula shows, by difference, the exact percentage of saleable coal lost to the refuse and may, therefore, be applied in economic calculations etc. in the usual way. This important information is not obtained from the formulae previously described.

No formula is known to the author which may be used to calculate the washer efficiency as a percentage, but it is possible to deduce a numerical value from the Tromp distribution factor curve which provides a qualitative measure of this efficiency and is, therefore, useful for comparative purposes.

(b) Graphical Representation of Washery Data.

Graphical representation of washery data is of great value as the operator can clearly see what is taking place in the plant and can consequently decide on the appropriate action. A number of methods have been proposed, but only a few of the more useful will be considered.

The Mirror Diagram[46]

In order to apply this method, it is necessary to carry out detailed float and sink analysis of the raw coal and the washery products. This data is then depicted graphically by using squared paper, ruled ten squares to the unit of length, whereby one small square can represent, say, one hundred weight of material and the complete figure, say, 100 tons of raw coal.

A horizontal line, AB, is drawn on the paper as shown in Figure 25 and vertical lines are drawn at convenient intervals to mark the specific gravities at which the float and sink analysis was carried out. The diagram is then divided into vertical zones, representing the washery products, by lines drawn at the specific gravities (or gravity) of separation. Thus, a two stage washing operation is shown in Figure 25 and the three zones represent clean coal, middlings and discard respectively.

The float and sink results of the raw coal are now plotted below the line AB in area units, the area representing each specific gravity fraction being given a distinctive colour or shading. The results of the analysis of the washery products are plotted above AB in a similar manner except that misplaced material is plotted in the appropriate zone, e.g. all specific gravity fractions of the clean coal are plotted in the clean coal zone. In this way, all losses etc. are clearly indicated. For example,

Figure 25/......46.

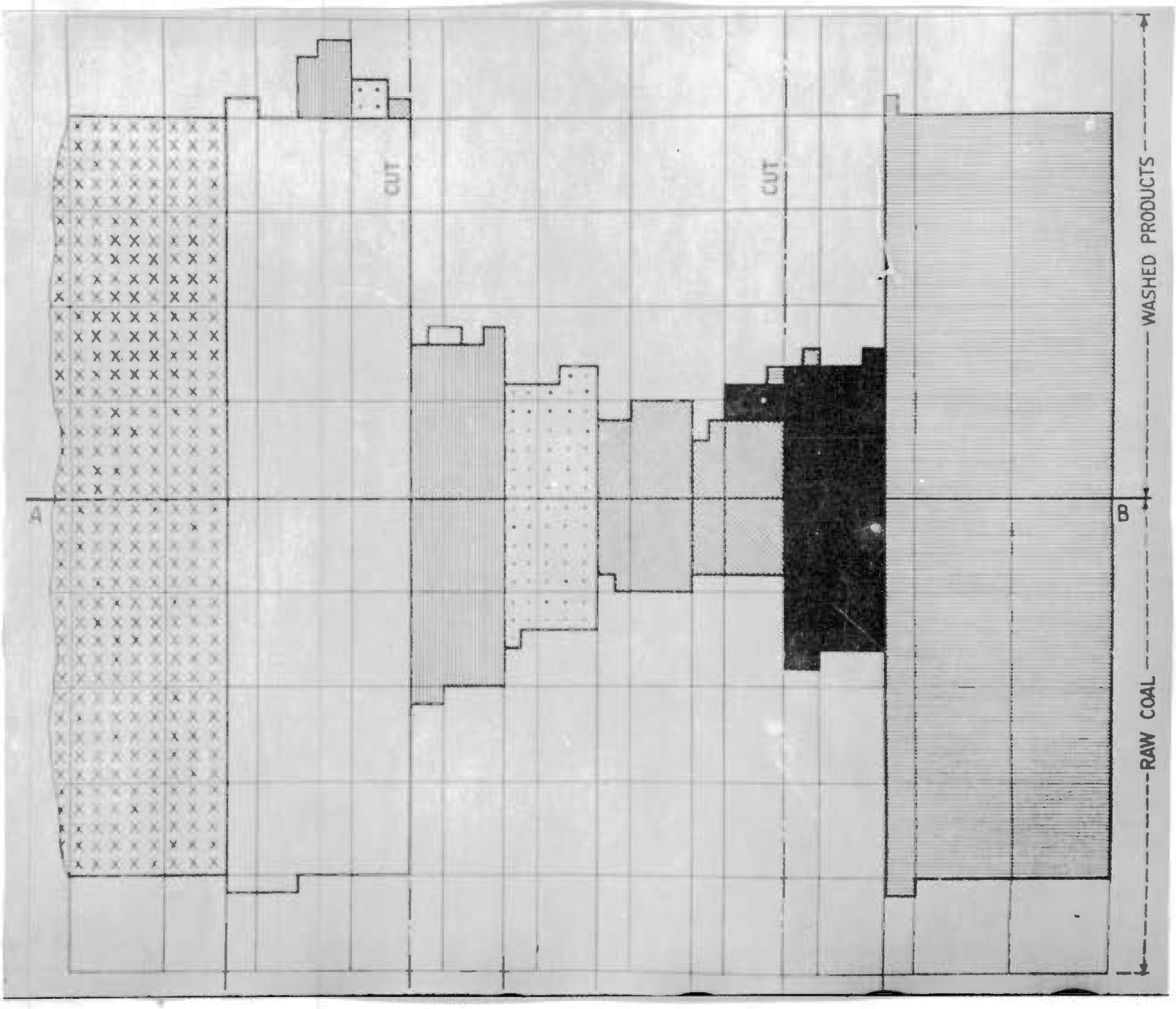

FIGURE 25: MIRROR DIAGRAM.

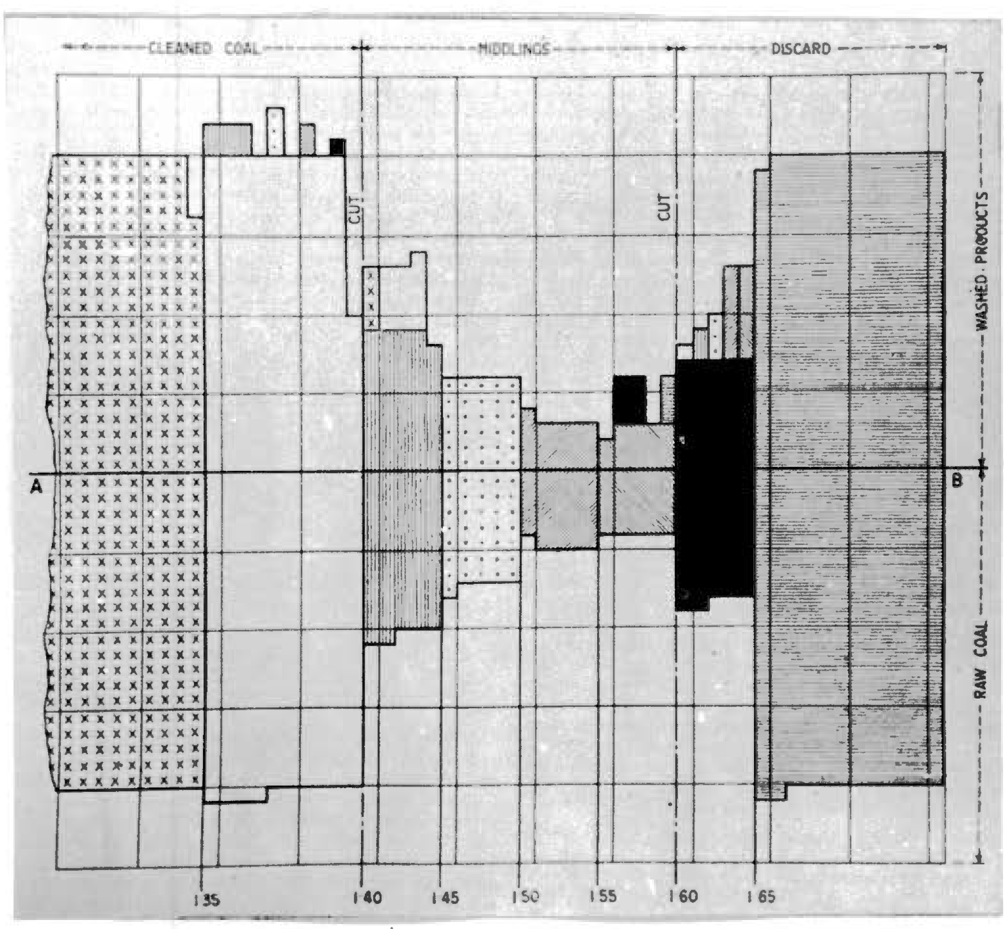

FIGURE 26: MIRROR DIAGRAM SHOWING LESS EFFICIENT SEPARATIONS THAN FIGURE 25.

Figure 25 shows that part of the material having specific gravity 1.45 to 1.5 was recovered in the clean coal, while, theoretically, all material of this specific gravity interval should have remained in the middlings, and so on. If the separation were perfect, the areas above AB would be a mirror-image of the areas below it.

This method provides a picture of the separations actually obtained under practical conditions, and, with experience, the efficiency of the operation could be assessed at a glance. By way of illustration, the mirror diagram for the same coal as that used for the tests in Figure 25, but cleaned by a less efficient process, is shown in Figure 26. The difference in the overall efficiencies will be apparent.

Tromp Distribution Factor Curve[47]

The mirror diagram described above shows the distribution of the material between the various washery products. It will be clear, therefore, that the appearance of the diagram depends on the specific gravity composition of the feed as well as on the efficiency of the process or washer efficiency.
It follows that washers can only be compared by means of this method when they are operated under the conditions previously mentioned. This difficulty was overcome by Tromp, who devised a method to show the performance of the washer, irrespective of the specific gravity composition of the feed.

The distribution factor curve proposed by Tromp, indicates what percentage of each specific gravity fraction of the feed was recovered in the clean coal, or product, and what was rejected in the tailing. This curve is, therefore, independent of the actual quantity of material present in each specific gravity interval.

In order to determine these percentages, it is necessary, as a first step, to carry out separate float and sink analyses of the washed product and of the tailing. The distribution factor (or percentage recovered) for each specific gravity interval is then calculated from the float and sink data as illustrated by the numerical example shown in Table 4. In this case the yield of washed product was 63 per cent of the feed coal.

TABLE 4.

Method of Determining the Tromp Distribution Factors.

Specific Gravity Interval.	Product. Percentage of Product.	Product. Percentage of Feed a x 0.63	Tailing. Percentage of Tailing.	Tailing. Percentage of Feed c x 0.37	Feed Percentage of feed b + d.	Distribution Factor (Percentage recovered) (b÷e)x100
	a	b	c	d	e	f
To 1.3	13.5	8.51	0.3	0.11	8.62	98.7
.3 to 1.35	47.9	30.18	1.8	0.67	30.85	97.8
.35 to 1.4	26.9	16.95	5.3	1.96	18.91	89.6
.4 to 1.45	8.3	5.23	20.1	7.44	12.67	41.3
.45 to 1.5	1.9	1.20	18.0	6.66	7.86	15.3
.5 to 1.58	0.8	0.50	20.3	7.51	8.01	6.2
> 1.58	0.8	0.50	34.2	12.65	13.15	3.8
Total	100.1	63.07	100.0	37.00	100.07	-

/47

The distribution factor for each specific gravity interval is now plotted against the mean specific gravity of that interval; thus, the distribution factor for the interval 1.3 to 1.35 would be plotted at 1.325 specific gravity. In the case of most South African coals, it may be assumed that there is no coal having a specific gravity lower than about 1.28 so that the mean of the first stage in the example is taken as 1.29. The distribution factor curve shown in Figure 27 has been plotted in this manner from the data in Table 4.

Tromp defines the specific gravity of separation as the specific gravity at which half of the material present goes to the product and half goes to the tailing, i.e. the distribution factor is 50% at the specific gravity of separation and the latter is simply read off the curve (i.e. 1.425 in Figure 27). The "Tromp curve" thus provides a method of determining the true specific gravity of separation and makes it possible to apply the various efficiency formulae with greater accuracy.

The shape of the Tromp curve is virtually independent of the specific gravity distribution of the feed and the specific gravity of separation. It is influenced, however, by the size grading of the feed and in the case of some washers, such as jigs, by the load to a certain extent. This curve may, therefore, be regarded as a characteristic of a washer when treating any particular size grading of feed and provided that the operating adjustments are not to be altered materially, may be used to predict the results likely to be obtained when washing any other coal of similar size grading. It will also be clear, that this is a valuable method of comparing the performance of washers or of determining the influence of alterations to the operating adjustments on separating efficiency.

Since the distribution factor curve, in effect, shows graphically the deviation from the ideal separation (i.e. a vertical line through the specific gravity of separation), the shape of the curve will clearly be a measure of the washer efficiency. Although visual comparison of Tromp curves will undoubtedly indicate differences in the efficiency, it is nevertheless, desirable to express the washer efficiency numerically.

This may be achieved by redrawing the Tromp curve as shown in Figure 28, i.e. the upper portion of the curve in Figure 27 has merely been rotated through 180 degrees around the 50 per cent distribution factor line. The shaded area, usually termed the "error area," will then be a measure of the deviation from a perfect separation and hence of the washer efficiency. It will be clear that the smaller the error area, the greater will be the washer efficiency.

It is customary to plot all Tromp curves that are to be compared to standard scales and then to measure the error areas by means of a planimeter. However, the error area may be expressed in "distribution factor x S.G." units, if desired, and Tromp curves drawn to any scale may then be readily compared.

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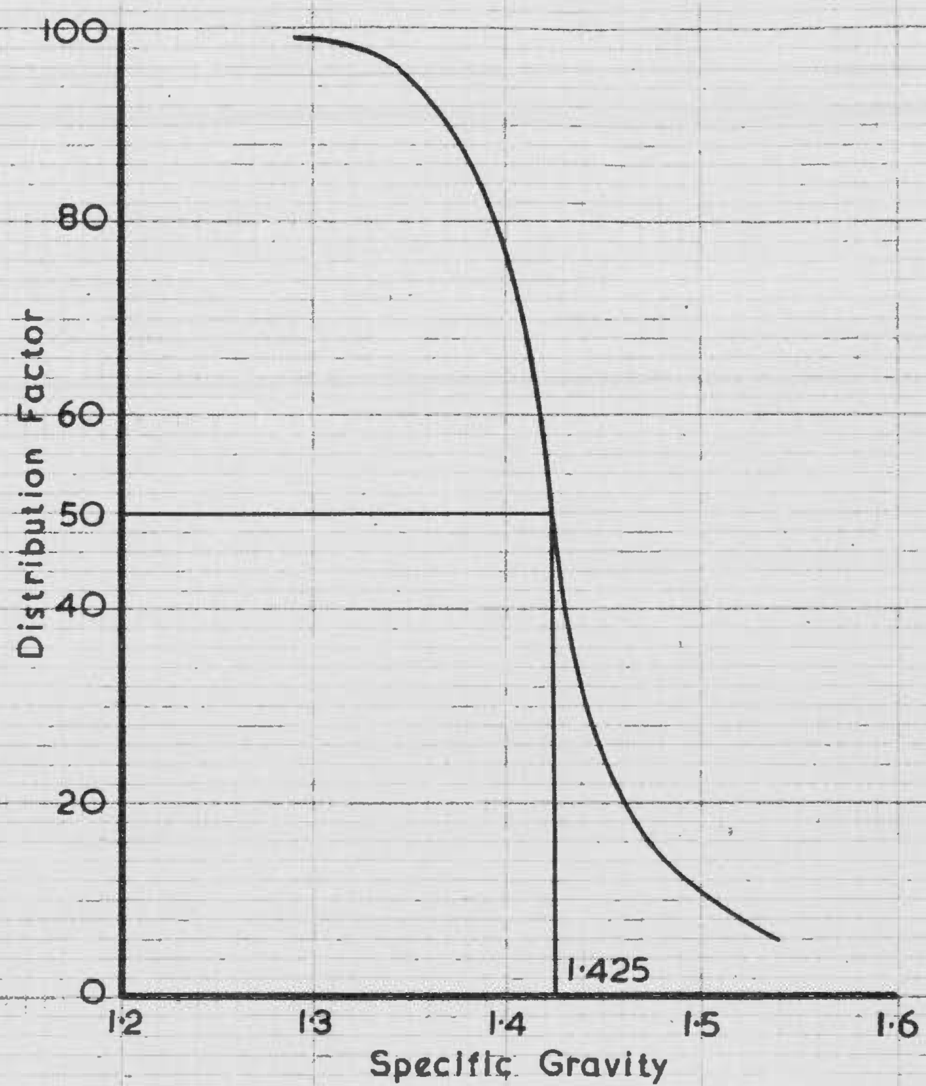

FIGURE 27 A typical Tromp curve

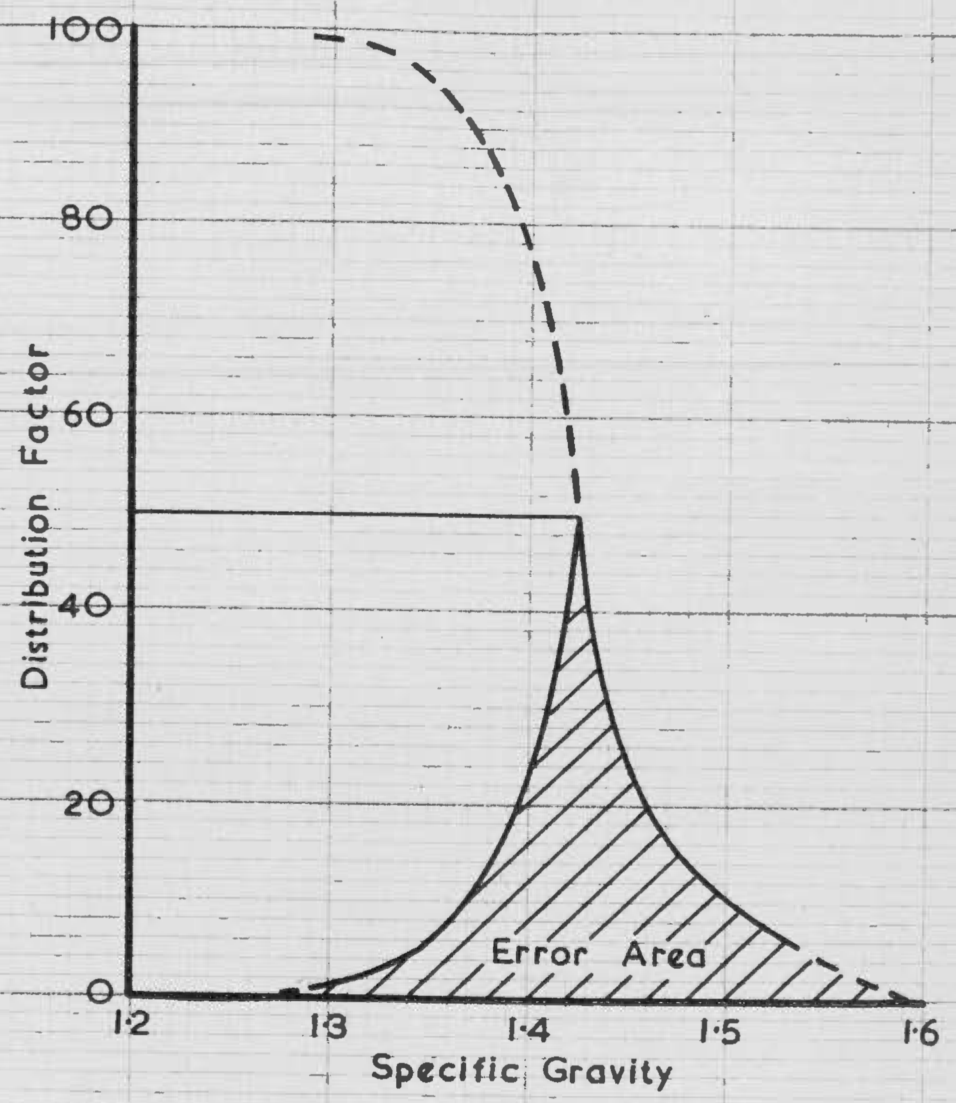

FIGURE 28 Tromp curve showing Error Area

By relating the shape of the distribution factor curve to a hazard curve, plotted on Gauss's formula, Tromp has found the distribution factor curve to conform to a definite mathematical equation, viz.:

$$V = 50 \times e^{\frac{-7854 \times f^{1.2}}{s^{1.2}}}$$

where V = relative frequency of yield deficit or distribution factor.

f = deviation from the density of separation

s = deviation factor.

The deviation above and below the specific gravity of separation are not necessarily the same (i.e. the curve is not always symmetrical). Tromp claims that the complete distribution factor curve can be drawn mathematically if three suitably selected points are determined by actual float and sink analysis. The author has found, however, that float and sink analysis is generally not accurate enough to enable one to adopt this procedure with confidence and it is considered, therefore, that as many points on the curve as possible should be determined by experiment.

A New Graphical Method for Representing Washery Data.

As stated previously, if the Tromp curve of a washer has been obtained for any particular feed coal, it may be used to estimate the results likely to be obtained in practice from this washer when treating any other coal of similar size grading. In this way it should then be possible to arrive at reasonably accurate conclusions regarding the suitability of various processes for any given feed coal. The necessary calculations could be carried out using a tabular method[48], but as this is likely to be laborious and subject to error, it was considered that a graphical method for arriving at the required data may be of some value. Suitable graphical representation would have the added advantage of clearly showing all losses, etc., and would thus facilitate interpretation of the results. The following procedure was consequently worked out.

As a first step, it is necessary to carry out detailed float and sink analysis of the raw feed and hence to obtain its washability curves, i.e. the "cumulative yield - specific gravity curve" and the "cumulative yield - cumulative ash curve." In order to facilitate reference to these curves, it is proposed to name them the "quantity characteristic curve" and "ash characteristic curve" respectively.

Two additional curves may now be derived from the characteristic curves to enable one to express the washability data in terms of areas. These curves will be termed the "quantity distribution curve" and the "ash distribution curve."

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The Quantity Distribution Curve.

Mathematically the "quantity distribution curve" is the derivative of the "quantity characteristic curve" and shows the rate of change of yield with respect to specific gravity. Theoretically this curve should be derived by determining the slope of the quantity characteristic curve at various specific gravities. It will be sufficiently accurate for all practical purposes, however, merely to determine the fractional yield over a small interval of specific gravity at each point selected as shown in Figure 29.

It will be seen that the quantity distribution (D_X) at specific gravity, X, is given approximately by

$$D_X = \frac{Y_b - Y_a}{X_b - X_a} \quad \left(\text{theoretically } \frac{\Delta y}{\Delta x}\right)$$

By proceeding in this manner, the complete quantity distribution curve can be obtained. It will be clear that the area of the elementary strip, X_aLMX_b, is equivalent to the fractional yield between the specific gravities Xa and Xb, i.e. elementary area $= D_X \, x(X_b - X_a)$

$$= \frac{Y_b - Y_a}{X_b - X_a} \, x \, (X_b - X_a)$$

$$= Y_b - Y_a.$$

Therefore, since the cumulative yield at a specified specific gravity, X, is given by the sum of the fractional yields up to that point, it will be appreciated that the cumulative yield, Y_X, is given by the shaded area, P, under the quantity distribution curve. This may be expressed mathematically as:-

$$\text{cumulative yield at specific gravity X,} = Y_X$$

$$= \int_{X'}^{X} D dX$$

$$= \text{Area P.}$$

As the quantity distribution curve could generally be expected to have no simple equation, it is proposed that this integration should be done mechanically by means of a planimeter or by some other recognised method of determining areas.

The Ash Distribution Curve:

As a first step in obtaining, what is termed, the "Ash distribution curve" it is necessary to determine a curve indicating the instantaneous ash at all specific gravities, i.e. a curve showing the actual ash content of the particles. The "Instantaneous ash - specific gravity curve" may be determined from the float and sink data as shown in the following example:

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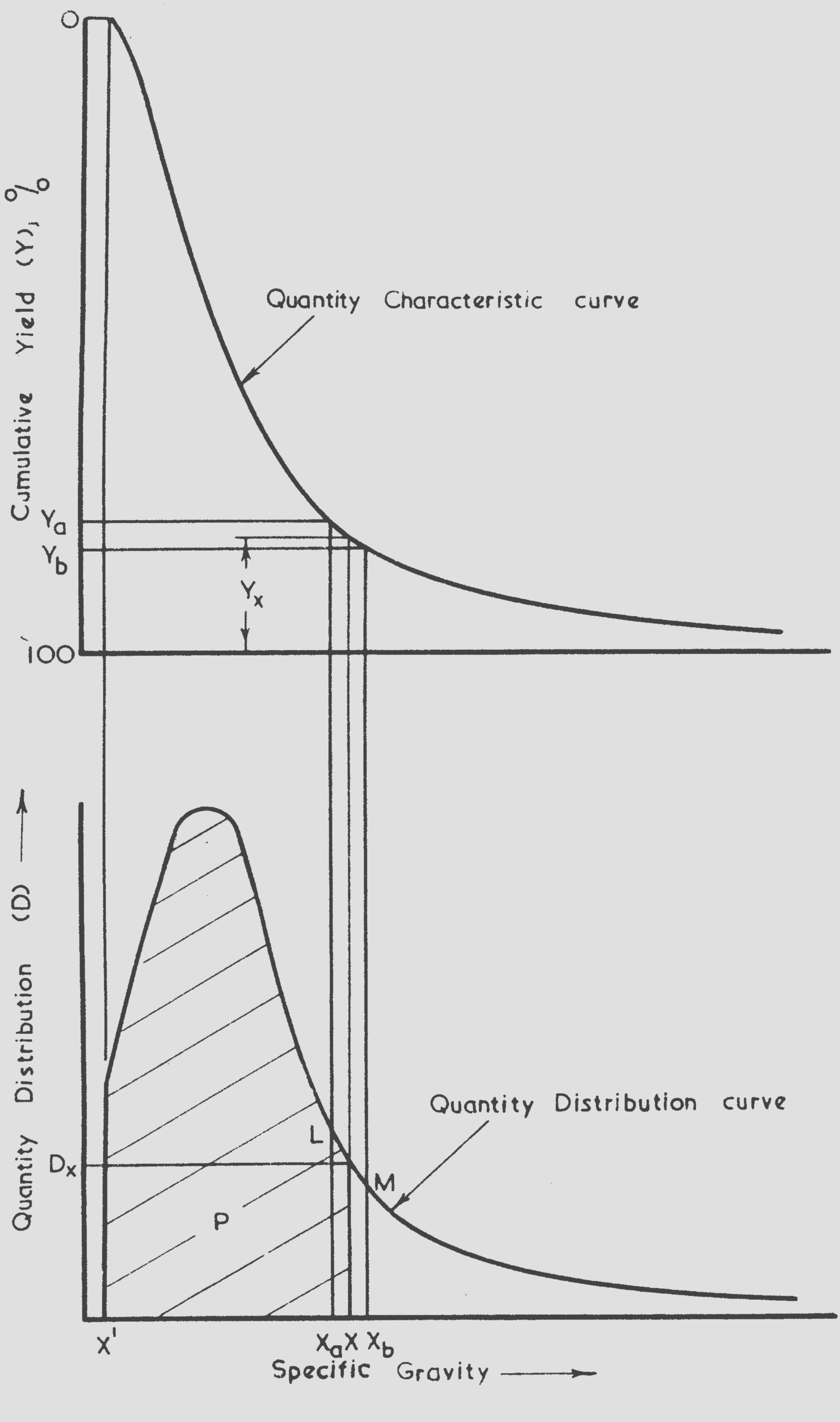

FIGURE 29 Derivation of the Quantity Distribution curve

TABLE 5.

Specific Gravity Interval.	Fractional Yield %.	Fractional Ash %.
1.275	0	0
1.275 - 1.28	1.2	3.1
1.28 - 1.30	5.7	3.4
1.3 - 1.35	21.6	5.7
1.35 - 1.4	17.7	8.1

If the float and sink analysis of the raw coal has been carried out fractionally, the results may be used directly but if the float and sink analysis was done cumulatively it will be necessary to determine the relevant fractional values by calculation. Assume that Table 5 represents the fractional float and sink analysis of the feed. It will be seen that 1.2% of the coal lies in the specific gravity range 1.275-1.28 and has an average ash content of 3.1%. Now all particles of this fraction do not have an ash content of 3.1%. The lightest particles present have a lower ash content while the heaviest particles have a higher value. If a linear relationship is assumed over the specific gravity range 1.275 to 1.28, it follows that the particles represented at the mean specific gravity would have an ash content of exactly 3.1% i.e. particles having a specific gravity of 1.2775 would have an ash content of 3.1%. Similarly, the instantaneous ash content at 1.29 specific gravity would be 3.4%. By proceeding in this way the Instantaneous ash - specific gravity curve of the general form shown in Figure 30 can be drawn. The method is approximate but is sufficiently accurate if the interval of specific gravity used for the float and sink analysis is small (say, 0.05 specific gravity).

Now, the ash content of a sample of coal depends not only on the ash content of the particles at each specific gravity, but also on the number of particles at each specific gravity. In other words, the cumulative ash content is a function of the quantity distribution and the instantaneous ash. It will readily be seen that the product of instantaneous ash and quantity distribution at a particular specific gravity is a measure of the ash contributed by the particles at that specific gravity. A summation of these products over the whole range of specific gravity would then be a measure of the ash content of the whole coal. The curve which is obtained when the products of instantaneous ash and quantity distribution are plotted against specific gravity will be termed the "ash distribution curve" and is of the general form shown in Figure 31, the area under this curve being a measure of the ash content.

The actual cumulative ash at specific gravity X is then given by the equation

$$\text{Cumulative ash at specific gravity X} = \frac{\int_{X'}^{X} A\,dX}{\int_{X'}^{X} D\,dX}$$

$$= \frac{\text{Area Q}}{\text{Area P}}$$

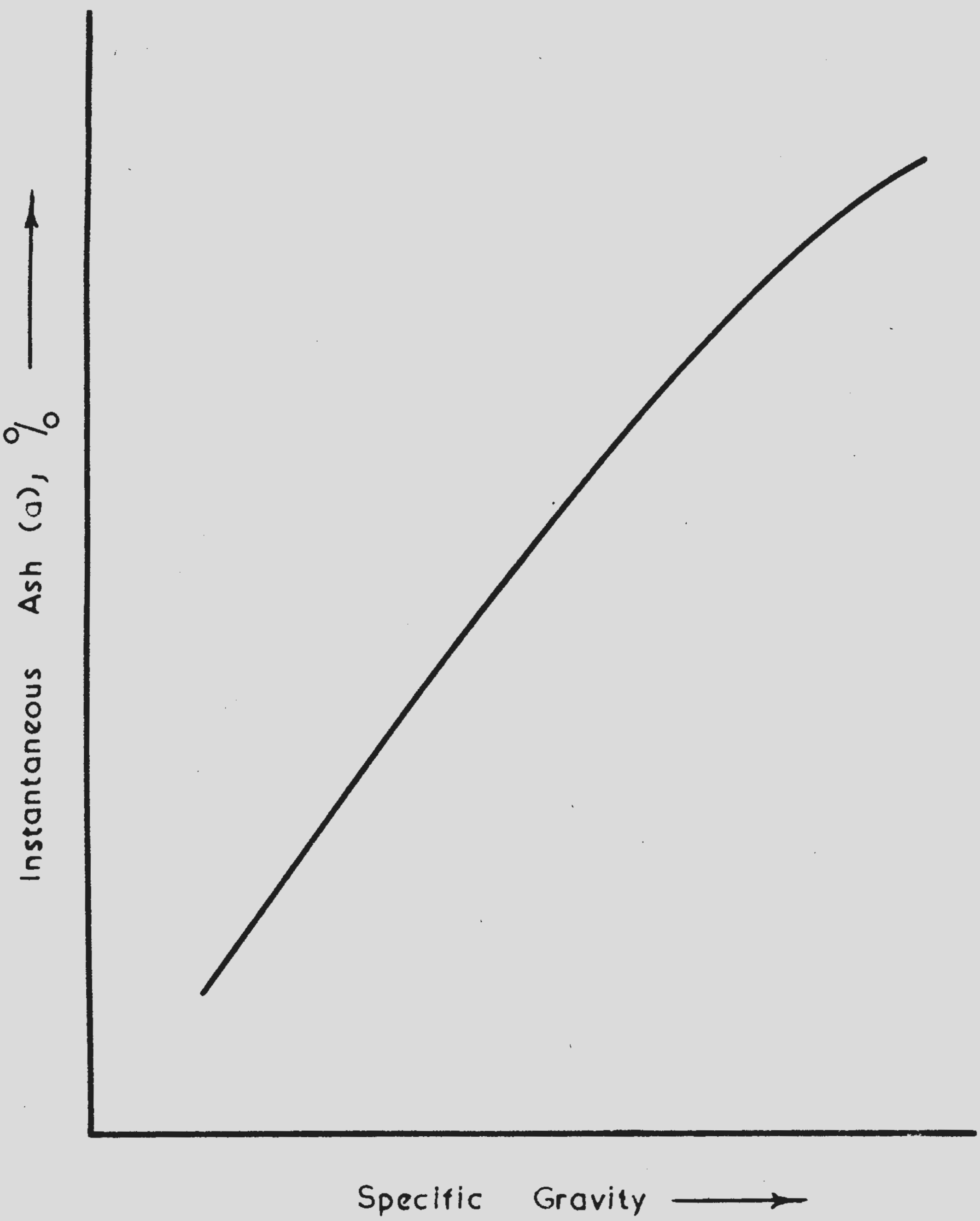

FIGURE 30 A typical Instantaneous Ash curve

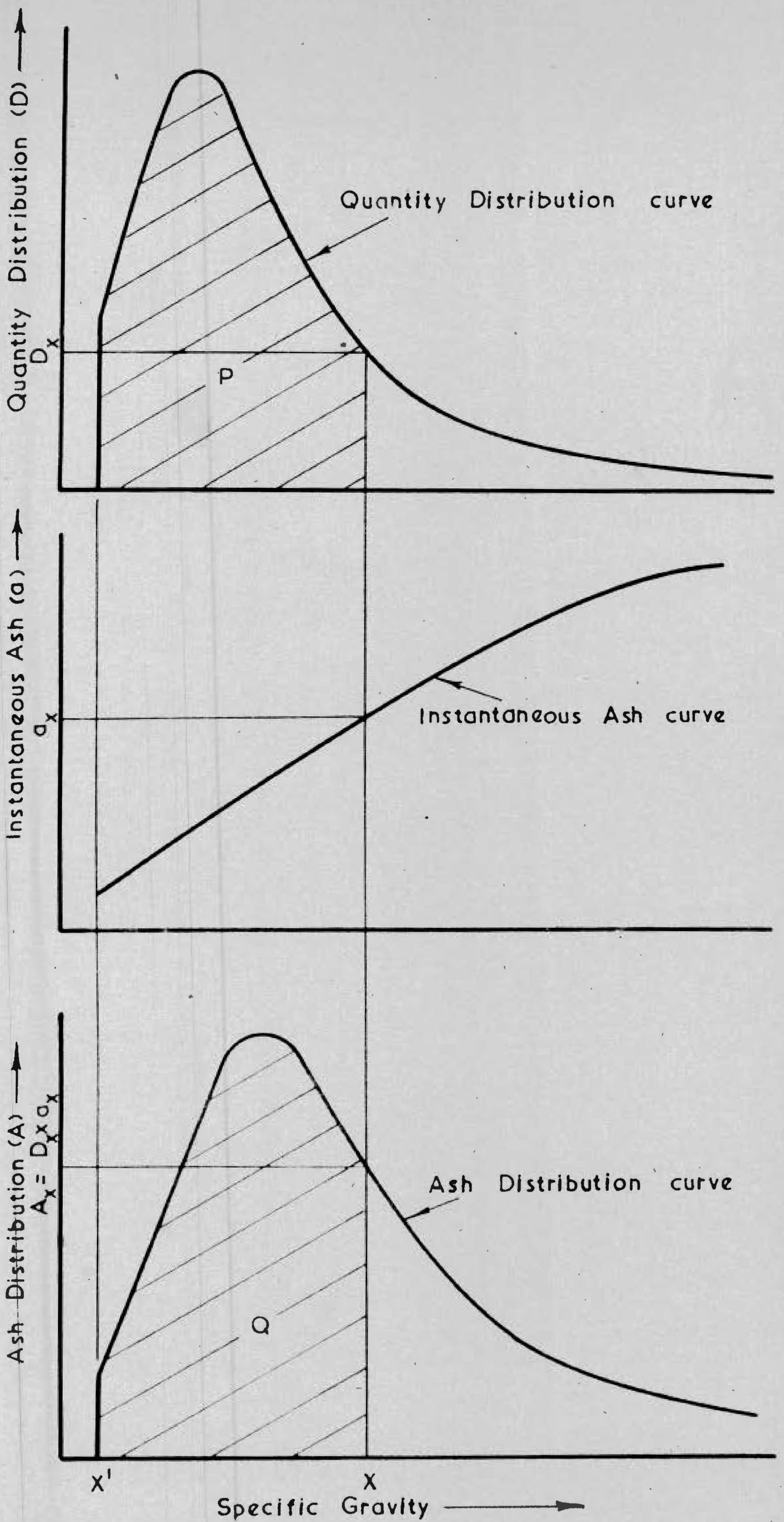

FIGURE 31 Derivation of the Ash Distribution curve

Estimation of the Yield and Ash Content under Practical Conditions.

As shown above, the quantity distribution and ash distribution curves may be used to enable one to express the theoretical yield and ash content of any specific gravity fraction in terms of areas.

By using the Tromp distribution factor curve of a particular washer in conjunction with these two curves it now becomes a comparatively easy matter to forecast the yield and ash content likely to be obtained in practice.

The quantity distribution curve shown in Figure 32 will be considered first and it will be assumed that it is desired to effect a separation at specific gravity $\bar{X}$.

If the separation were perfect, the "cutting line" would be $\bar{X}A$ and the yield would be given by the area $X'PA\bar{X}$. In practice, however, the separation will not be perfect. Let it be assumed that it will deviate from the ideal separation as indicated by the Tromp distribution factor curve shown in Figure 33. This curve shows that all material having a specific gravity lower than X_a will be recovered in the product, while all material having a specific gravity greater than X_c will be rejected in the tailing. The two extremeties of the "cutting line" will therefore be points B and X_c, Figure 32. For specific gravities between X_a and X_c, a varying percentage of the material present at any specific gravity is recovered in the product the remainder being discarded. Thus, Figure 33 shows that 80% of the material having specific gravity X_b will be recovered in the product. At this specific gravity, the quantity of feed present is represented by the line X_bC (of thickness $\triangle X$) (Figure 32). The quantity which would be recovered in the product will then be represented by the line $X_bD = 0.8\ X_bC$. Similarly, at the specific gravity of separation, $\bar{X}$, the quantity recovered will be represented by the line $\bar{X}E = 0.5\ \bar{X}A$. The actual "cutting line" will, therefore, be BX_c and the actual yield of washed product is given by the area $X'PBX_c$. Area BAE represents "clean" coal lost in the tailing and area $\bar{X}EX_c$ represents "tailing" recovered in the washed coal.

In the same way, the Tromp distribution factor curve may be used in conjunction with the Ash distribution curve to obtain the actual "cutting line", GX_c, in terms of ash units as shown in Figure 34. The area $X'GX_c$ will then represent the actual ash content of the washed coal, the value of which can be calculated from the following expression:-

$$\text{Actual ash content of washed coal} = \frac{\text{Area } X'GX_c \text{ (Fig. 34).}}{\text{Area } X'PBX_c \text{ (Fig. 32).}}$$

The reader/........52.

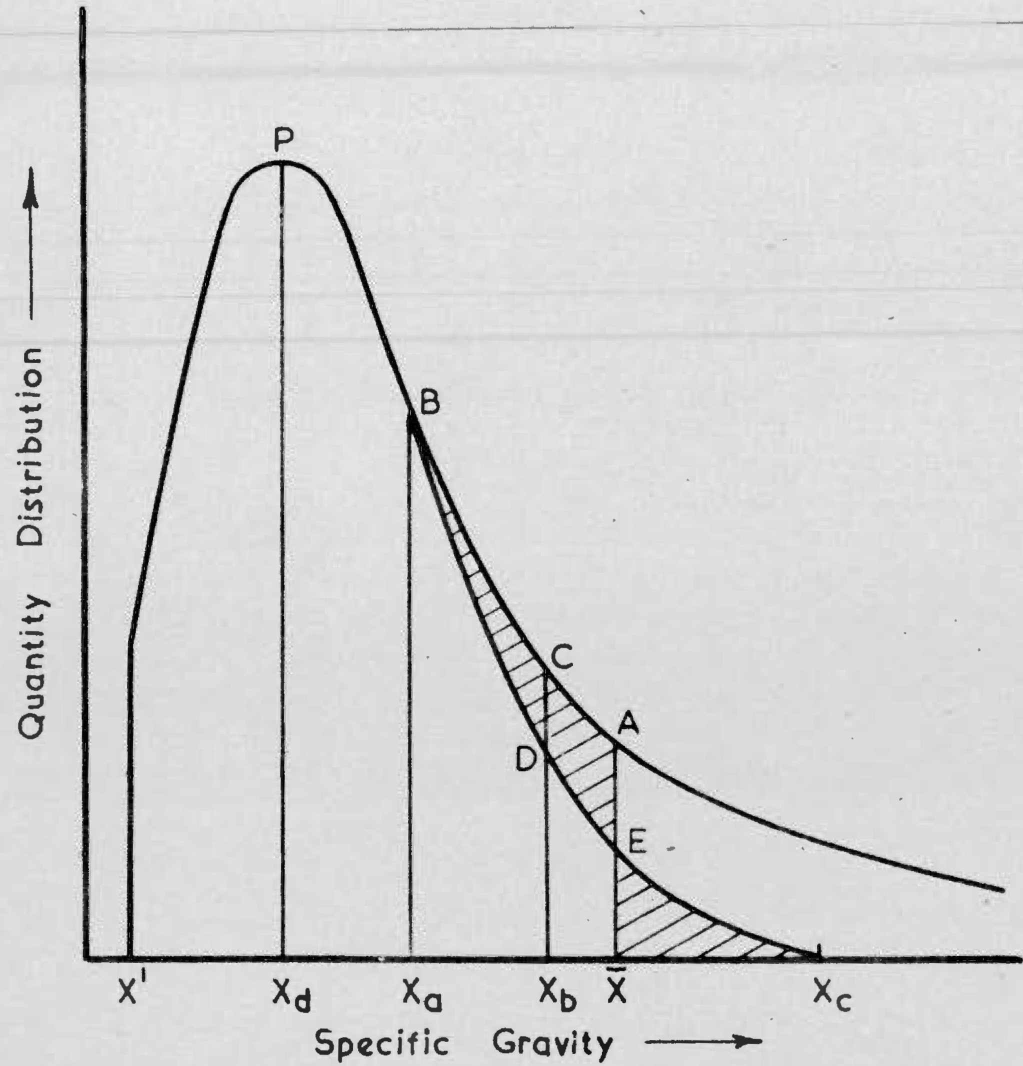

FIGURE 32 Quantity Distribution curve showing the actual separation

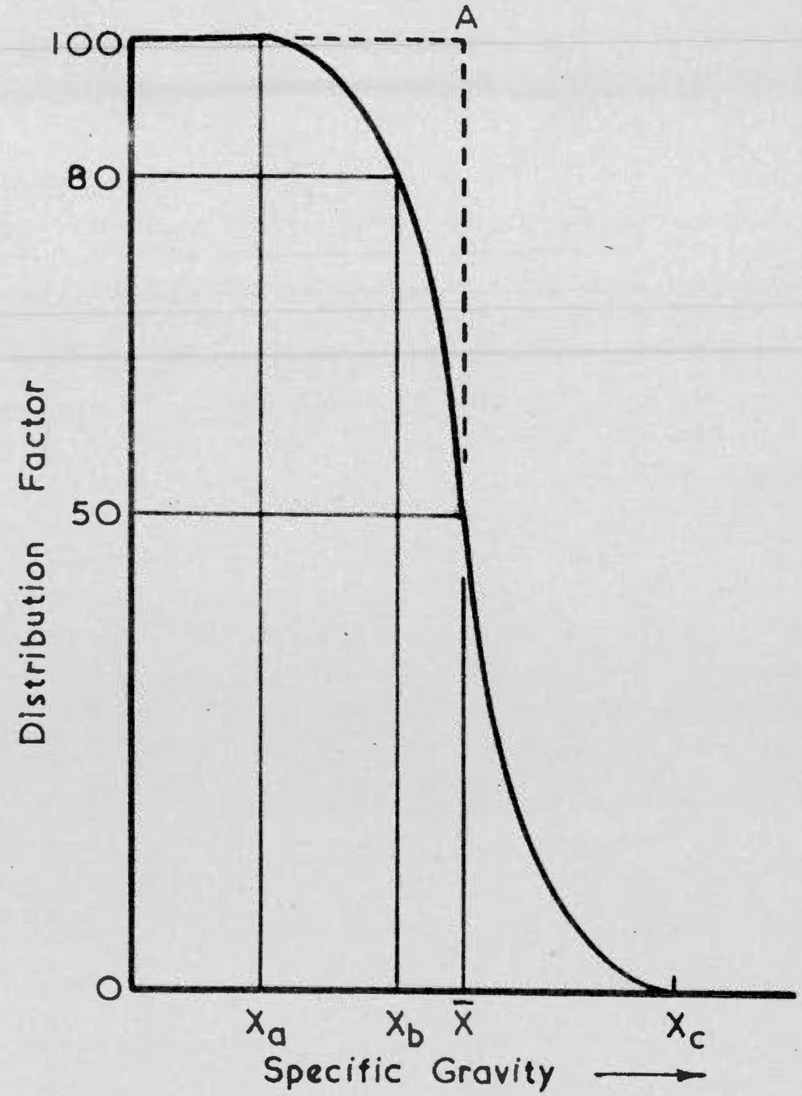

FIGURE 33 Distribution Factor curve

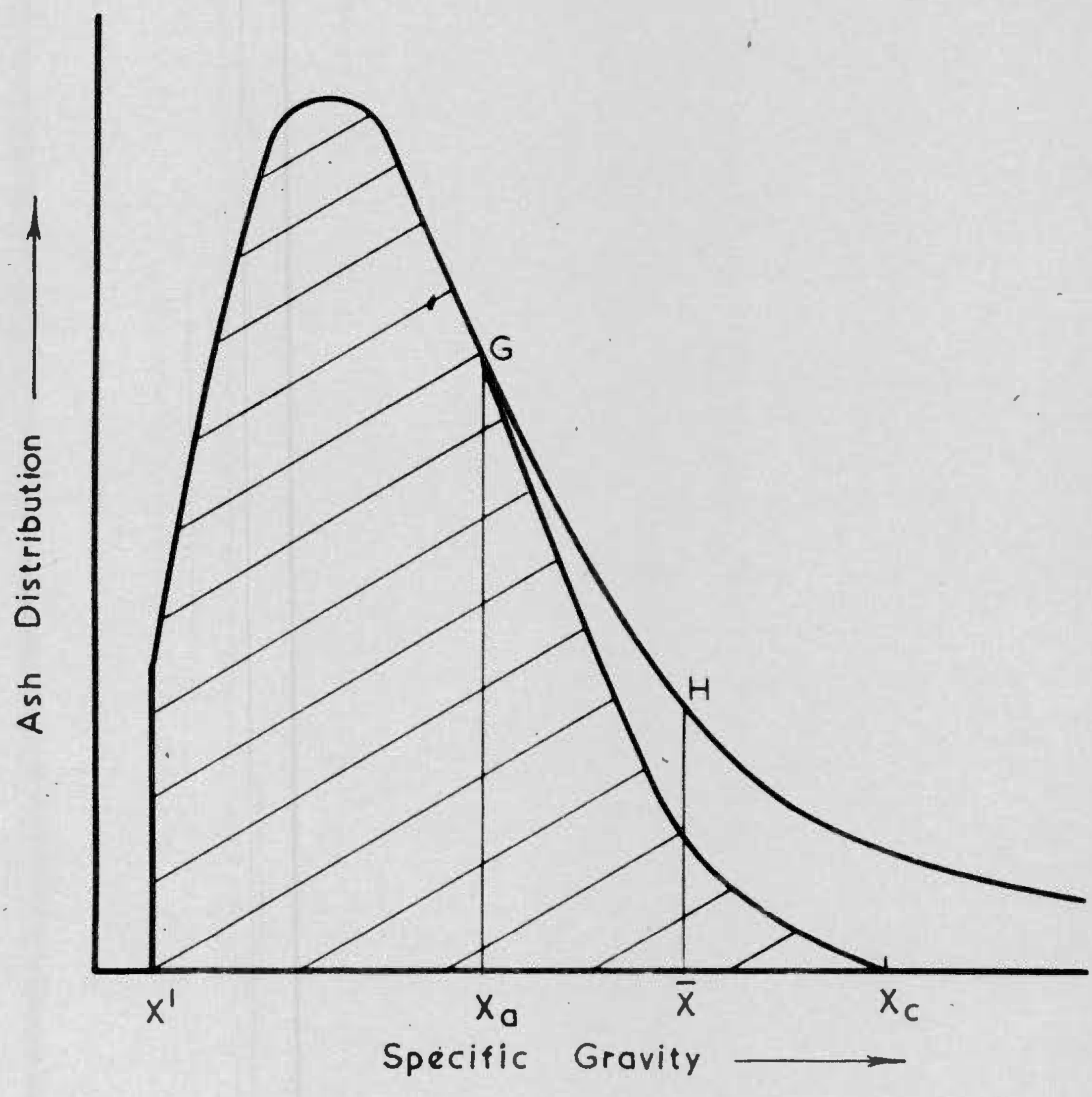

FIGURE 34 Ash Distribution curve showing the actual separation

The reader is referred to the original paper[49] in which this method was described for a detailed example of its application.

Although the method described may be used to estimate the yield and ash content of product likely to be obtained in practice, it is probably of greater value as a graphical representation of washery data. In other words, if the data actually obtained during a test on a washer is plotted in the manner suggested, a vivid picture of the washery performance will be obtained, clearly showing the relative magnitudes of all misplaced material. In this respect it is superior to the mirror diagram, for example, which would appear to require considerable practice before it could be interpreted at a glance.

The quantity distribution curve also provides a very clear conception regarding the difficulty of the washing problem at any specific gravity. For example, Figure 32 indicates that it would be extremely difficult to effect a separation with reasonably high overall efficiency at specific gravity Xd and that only a washer which is capable of very exact separation (i.e. having a small error area) would be suitable. In the vicinity of specific gravity $\bar{X}$, however, the quantity distribution is relatively lower, consequently the areas BAE and $\bar{X}EX_c$, representing the loss and gain, would be small compared with $X_1A\bar{X}$ (theoretical yield) even for a comparatively inefficient washer (large error area) and high overall efficiency could be expected. At this point, therefore, the use of a washer capable of effecting very sharp separation, would probably be of little advantage.

An Expression for the Overall Efficiency of a Process Derived from the Ash Distribution Curve.

It has already been pointed out that the majority of the formulae which have been proposed for determining the overall efficiency have little or no physical significance and one is consequently in some doubt as to whether they have any real value. While studying the graphical representation described above, it occurred to the author that an efficiency expression could be evolved from the various diagrams, which is capable of physical interpretation and may, therefore, be of interest.

Now both the quantity and quality of washed coal obtained from a washer are of economic importance and both these factors should, therefore, be taken into account when assessing the efficiency of a process. This could be achieved by basing the formula on the ash distribution curve, as the area under this curve represents the product of ash content and yield.

Since the quantity of "good" coal lost and the quantity of "poor" coal gained have an important bearing on the final product it was decided to deduce expressions which would represent the "recovery efficiency" and the "reject efficiency." These expressions are readily derived from the ash distribution curve shown in Figure 35. Thus theoretical recovery of "good" coal is represented by the area $X'AB\bar{X}$ while actual recovery is represented by the area $X'AC\bar{X}$ (area ABC represents "good" coal lost).

Hence,/......53.

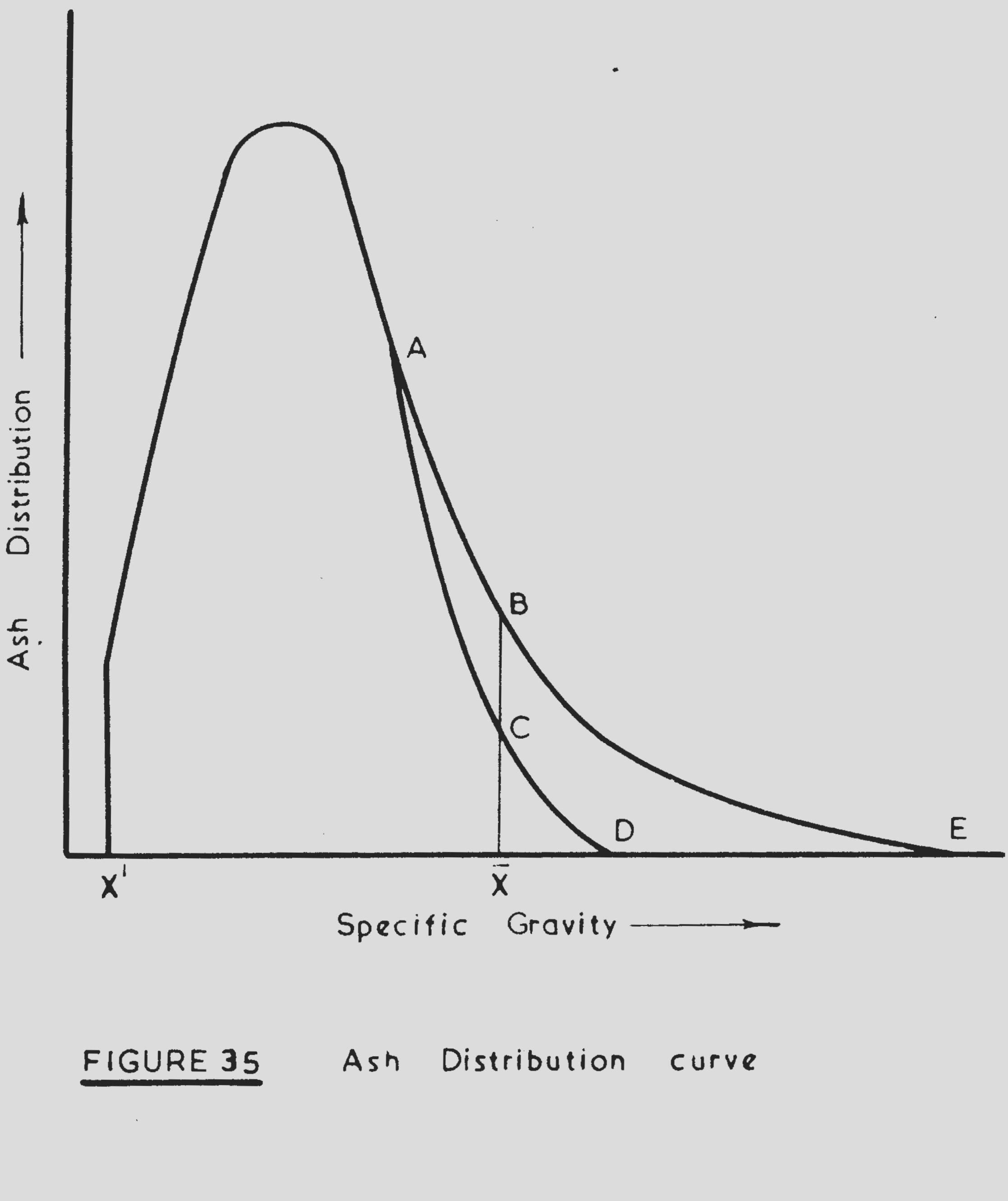

FIGURE 35 Ash Distribution curve

Hence, Recovery efficiency = $\frac{\text{Area } X' AC\bar{X}}{\text{Area } X' AB\bar{X}}$

Similarly, the theoretical reject is given by area $\bar{X}BE$ and actual reject by area DCBE, so that

$$\text{Reject efficiency} = \frac{\text{Area DCBE}}{\text{Area } \bar{X}BE}$$

(It will be clear that neither of these efficiencies can ever exceed unity, which is the case in several formulae previously described).

The overall efficiency may then be inferred as follows:-

Overall efficiency = Recovery efficiency x Reject efficiency.

These expressions are now capable of conversion into terms of float and sink data on the actual product and tailing obtained.

Suppose a float and sink analysis of the product and tailing was carried out at specific gravity $\bar{X}$ with the following results:-

Assume Product = Y% of feed and tailing = (100-Y)%

Float and Sink on Product.

Percentage of product floating at $\bar{X}$ = P_f %

Percentage of product sinking at $\bar{X}$ = $(100-P_f)$ %

Ash content of product floats = A_{pf} %

Ash content of product sinks = A_{ps} %

$\therefore$ Product floating as % of the feed = $\frac{P_f \; Y}{100}$ %

& Product sinking as % of the feed = $\frac{(100 - P_f) \; Y}{100}$ %

$\therefore$ Area $X' AC\bar{X} = \frac{(P_f \; Y) \; A_{pf}}{100}$

and Area $\bar{X}CD = \frac{((100 - P_f) \; Y) \; A_{ps}}{100}$

Float and Sink on Tailing:

Percentage of tailing floating at $\bar{X}$ = T_f %

Percentage of tailing sinking at $\bar{X}$ = $(100-T_f)$ %

Ash content of tailing floats = A_{tf} %

Ash content of tailing sinks = A_{ts} %

$\therefore$ Tailing floating as % of feed = $\dfrac{(T_f\ (100-Y))}{100}$ %

& " sinking " " " " = $\dfrac{((100-T_f)\ (100-Y))}{100}$ %.

$\therefore$ Area ABC = $\dfrac{(T_f\ (100-Y))\ A_{tf}}{100}$

& Area BCDE = $\dfrac{((100-T_f)\ (100-Y))\ A_{ts}}{100}$

Hence Area $X'AB\bar{X}$ = Area $X'AC\bar{X}$ + area ABC.

$$= \left\{ (P_f\ Y)A_{pf} + (T_f(100-Y))A_{tf} \right\} \div 100$$

and area $\bar{X}BE$ = area $\bar{X}CD$ + area BCDE.

$$= \left\{((100-P_f)\ Y)\ A_{ps} + ((100-T_f)(100-Y))A_{ts}.\right\} \div 100$$

$\therefore$ Recovery efficiency = $\dfrac{P_f\ Y\ A_{pf}}{(P_f\ Y\ A_{pf})+(T_f(100-Y)A_{tf})}$

$$= \frac{1}{1 + \dfrac{T_f(100-Y)A_{tf}}{P_f\ Y\ A_{pf}}} = \frac{1}{1 + \dfrac{\text{\% tailing floats x \% tailing x \% ash of tailing floats.}}{\text{\% product floats x \% product x \% ash of product floats.}}} \quad -\ (1)$$

& Reject efficiency = $\dfrac{(100-T_f)(100-Y)A_{ts}}{(100-T_f)(100-Y)A_{ts} + (100-P_f)YA_{ps}}$

$$= \frac{1}{1 + \dfrac{(100-P_f)Y\ A_{ps}}{(100-T_f)(100-Y)A_{ts}}} = \frac{1}{1 + \dfrac{\text{\% product sinks x \% product x \% ash of product sinks.}}{\text{\% tailing sinks x \% tailing x \% ash of tailing sinks.}}} \quad -\ (2)$$

/......55.

To facilitate reference to the new expression for the overall efficiency of a washing process it is proposed to refer to it as the "Fuel Research Institute Efficiency Formula," i.e. Fuel Resarch Institute efficiency

= Recovery efficiency X Reject efficiency

= expression (1) X expression (2).

Modified Fuel Research Institute Efficiency Formula.

As already pointed out, it is considered that the yield and ash content should both be taken into account in order to obtain an expression which will truly reflect the efficiency of a washing process. This was achieved above by basing the formula on the Ash Distribution Curve, the ash distribution representing the product of quantity and quality.

The resultant formulae are cumbersome, however, and require at least four ash determinations in addition to float and sink analysis of the product and tailing at the specific gravity of separation.

In order to simplify the expression for the recovery and reject efficiencies and to minimise analyses the possibility of using modified formulae based on the quantity distribution curve of the feed was investigated.

Consider the quantity distribution curve shown in Figure 36 and assume that FJ represents the actual cutting line in a washer effecting a separation at specific gravity $\bar{X}$. The Recovery and reject efficiencies may then be defined as follows:-

$$\text{Recovery Efficiency} = \frac{\text{Area } X' FH\bar{X}}{\text{Area } X' FG\bar{X}}$$

$$= \frac{\%\text{ product floats at SG}\bar{X} \times \%\text{ product}}{\%\text{ feed floats at S.G. } \bar{X}}$$

$$= \frac{\%\text{ product floats} \times \%\text{ product}}{(\%\text{ product floats} \times \%\text{ product}) + (\%\text{ tailing floats} \times \%\text{ tailing})}$$

$$= \frac{1}{1 + \dfrac{\%\text{ tailing floats} \times \%\text{ tailing}}{\%\text{ product floats} \times \%\text{ product}}} \quad - (3)$$

$$\text{Reject Efficiency} = \frac{\text{Area GKJH}}{\text{Area GK}\bar{X}}$$

$$= \frac{\%\text{ tailing sinks} \times \%\text{ tailing.}}{\%\text{ feed sinks.}}$$

$$= \frac{1}{1 + \dfrac{\%\text{ product sinks} \times \%\text{ product}}{\%\text{ tailing sinks} \times \%\text{ tailing}}} \quad - (4)$$

and modified Overall Efficiency = (3) x (4).

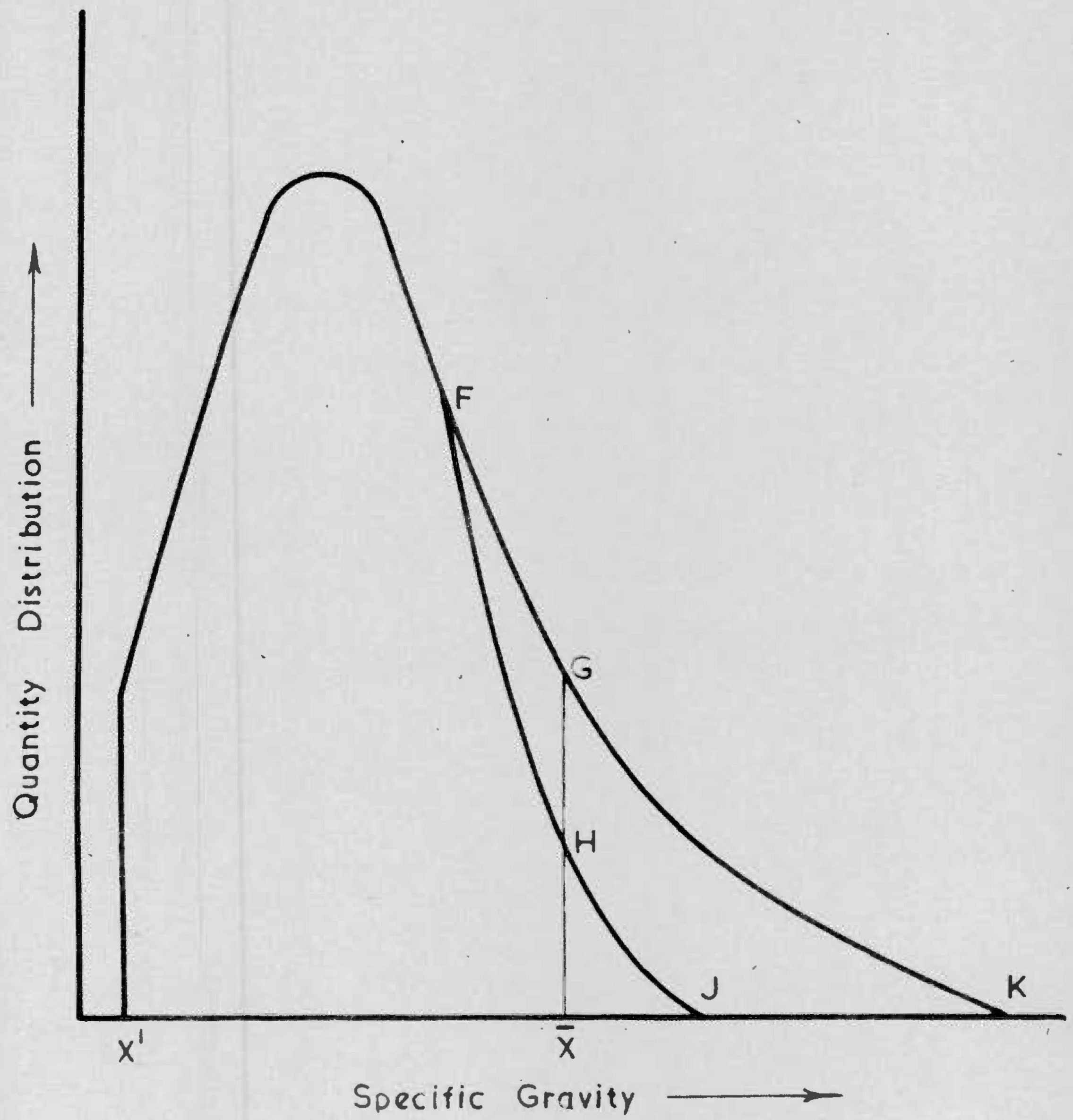

FIGURE 36 Quantity Distribution curve

Both the formulae proposed above will require to be tested thoroughly in practice, however, before their value can be assessed.

<u>Interpretation of the Drakeley Efficiency Formula with the Aid of the Quantity Distribution Curve.</u>

It may be of interest at this stage to interpret the physical significance of, say, the Drakeley efficiency formulae with the aid of the quantity distribution curve shown in Figure 36.

<u>Drakeley Formula.</u>

$$\text{Qualitative Factor} = \frac{100 \times \left[\dfrac{\text{area } X'FH\bar{X}}{\text{area } X'FHJ} - \dfrac{\text{area } X'FG\bar{X}}{\text{area } X'FGK}\right]}{\dfrac{\text{area } GK\bar{X}}{\text{area } X'FGK}}$$

It will be seen that this expression is not capable of any physical interpretation.

$$\text{Quantitative Factor} = \frac{100 \times \left[\dfrac{\text{area } X'FG\bar{X}}{\text{area } X'FGK} - \dfrac{\text{area } FGH}{\text{area } X\ FGK}\right]}{\dfrac{\text{area } X'FG\bar{X}}{\text{area } X'FGK}}$$

$$= \frac{100 \times \left[\text{area } X'FG\bar{X} - \text{area } FGH\right]}{\text{area } X'FG\bar{X}}$$

$$= 100 \times \frac{\text{area } X'FH\bar{X}}{\text{area } X'FG\bar{X}}$$

This is, ofcourse, the expression (3) for recovery efficiency deduced before.

Similarly for the modified Drakely expression it can be shown that the quantitative factor has no significance, while the <u>qualitative</u> factor reduces to the reject efficiency (expression 4).

This analysis suggests, then, that the Drakely quantitative factor and the <u>modified</u> Drakely qualitative factor should be used in combination to obtain the value of the overall efficiency.

THE EFFICIENCIES OF VARIOUS WASHING PROCESSES.

As explained before, in order to compare the efficiencies of different washing processes it is necessary to determine the "washer efficiency" in each case. This may be achieved by obtaining the Tromp curves of the washers concerned. Although available literature did not contain sufficient data to enable a detailed comparison of the different processes to be made, the Tromp curves shown in Figure 37 will enable the reader to obtain some idea of the potential efficiency of the various processes. The sources from which this information was obtained are indicated in Table 6.

TABLE 6.

Washer.	Reference Letter.	Size Grading of Feed.	Literature Reference.
Jig	A	3" to $1\frac{1}{2}$"	(50)
	B	$1\frac{1}{2}$" to $\frac{3}{4}$"	(50)
	C	$\frac{3}{8}$" to $\frac{3}{16}$"	(50)
	D	minus 20 mesh	(50)
	E	8 mm. to 0.5 mm.	(45)
Trough	F	1" to $\frac{5}{16}$"	(51)
	G	minus $\frac{5}{16}$"	(51)
	H	5 mm. to 0.5 mm.	(45)
Classifier	I	$\frac{13}{16}$" to 28 mesh.	(52)
Heavy medium	J	1" to $\frac{1}{8}$"	(53)
	K	3" to 1"	(53)
Froth Flotation	L	-10 mesh + 35 mesh	(54)
	M	-35 mesh + 100 mesh	(54)
	N	1 mm. to 0.05 mm.	(45)

It will be clear from Figure 37 that Jigs, Troughs and Classifiers are not capable of very sharp separations under normal operating conditions and that the efficiency tends to decrease with feed size. Heavy medium washers, on the other hand, are capable of effecting separations which do not deviate substantially from the ideal.

Although/........58.

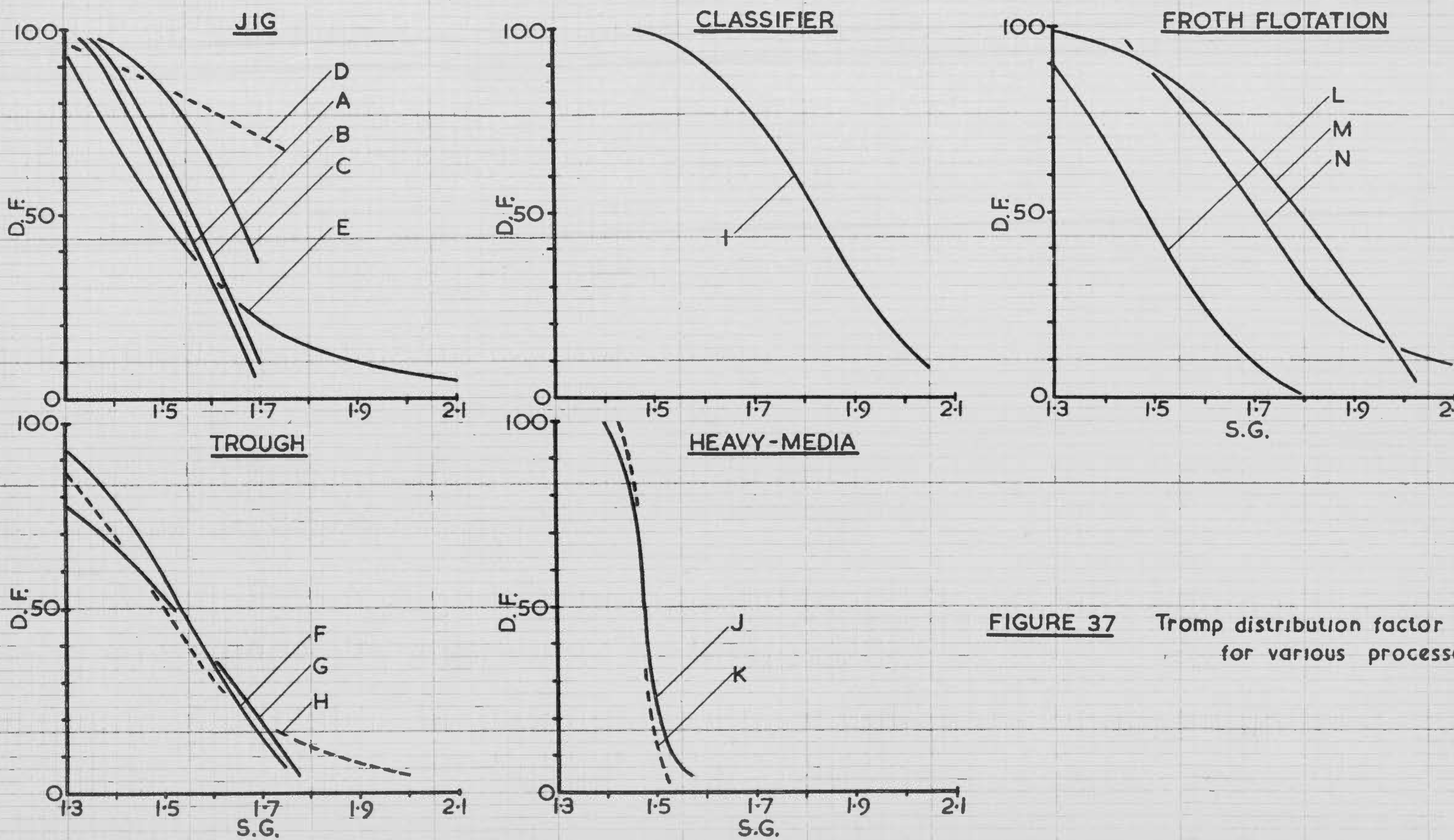

FIGURE 37 Tromp distribution factor curves for various processes

Although a froth flotation separation does not depend on the specific gravity of the particles, there is often a relationship between the surface properties and the specific gravity, so that the Tromp curves may be taken as a rough indication of the separation. While the cut does not appear to be particularly sharp, it compares favourably with that of jigs, troughs and classifiers when treating the coarser sizes.

Now, the suitability of a washer depends not only on its "washer efficiency" but also on the quantity of near gravity material in the feed to be washed. Thus, jigs, troughs and classifiers may be satisfactory for separations at specific gravities where the quantity distribution is low, but would be quite unsuitable if the quantity distribution exceeds a certain value, and in such cases, heavy medium washers would have to be used. It is for this reason that Bird's ± 0.10 specific gravity distribution figure (i.e. a practical value for quantity distribution in the vicinity of the specific gravity of separation) may be used as a measure of the difficulty of a washing problem. On this basis, washers may be roughly classified as follows in accordance with the percentage of near gravity material they may be expected to handle satisfactorily.

	± 0.10 S.G. distribution
Concentrating Tables	less than about 10%
Jigs, troughs, classifiers,	" " " 15%
Heavy medium washers	40% and even more

Needham[55] has set out the lower limits of size that can be treated effectively by various processes as shown below. (It is assumed that the conditions regarding ± 0.1 S.G. distribution must also be satisfied).

Process.	Limits.
Jigs and troughs.	Good down to 1 mm.; fair between 1 mm. and 0.5 mm.; negligible cleaning below 0.5 mm.
Concentrating Tables.	Good down to 0.5 mm.; fair between 0.5 mm. and 0.25 mm.; negligible cleaning below 0.25 mm.
Froth Flotation.	Good between 1 mm. and 0.15 mm.; poorer for sizes below 0.15 mm.; usually acceptable for whole range 1 mm. to 0.
Heavy medium. (conventional)	Unsuitable below 1.5 mm.
Sy-Vor	Not proved; suggested from 1.5 mm. to 0.15 mm.

In the/........59.

In the case of South African coals, the percentage of near gravity material at the required specific gravity of separation is generally materially higher than it is, say, for British coals. Washers which have proved satisfactory overseas are not of necessity suitable, therefore, for treating South African coal. Great care should consequently be exercised in the selection of a washer to ensure satisfactory operation.

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LIST OF DIAGRAMS EX LITERATURE.

PART 2.

A STUDY OF THE OPERATION OF THE CYCLONE WASHER AND ITS APPLICATION TO WITBANK FINE COAL.

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A STUDY OF THE OPERATION OF THE CYCLONE WASHER AND ITS APPLICATION TO WITBANK FINE COAL.

CONTENTS.

WASHING..../

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PART 2.

A STUDY OF THE OPERATION OF THE CYCLONE WASHER AND ITS APPLICATION TO WITBANK FINE COAL.

INTRODUCTION.

Although the Union of South Africa possesses reasonably large reserves of coal of various qualities, the reserves of coking coal are comparatively small and are being exhausted at an alarming rate. The proved reserves of coking coal are estimated at about 260 million tons, of which 160 million tons are in Natal and 100 million tons in the Transvaal. The Natal coking coal is high-quality straight coking coal, while that in the Transvaal is mainly blend quality.*

The present rate of production of coke is approximately 800,000 tons per annum,(1) of which about 75 per cent is consumed by the iron and steel industry. In order to produce this quantity of coke, about 1.3 million tons of coking coal are required per annum and it is estimated that this rate of consumption is likely to be doubled in the near future, mainly on account of the increasing requirements of the steel industry. Since it is estimated that some 3 million tons of coking coal and blend coal are also used for non-coking purposes per annum, it appears that the rate of consumption of this coal will be in the region of 5.5 million tons per annum within the next few years. On this basis, it will be clear that the life of the coking coal reserves is only about 45 years. By restricting the use of this coal to coke production as far as possible this figure can probably be doubled, but even 90 years is a

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* For the purposes of this Report, the coking coals of the Union are divided into two classes, as defined by the South African Coal Commission 1946-47, p.17.

(a) Straight coking coals, which contain well develcped coking properties and are capable after normal preparation of producing good quality metallurgical coke.
(b) Blend coking coals which contain poorly developed coking properties and are incapable of producing good quality metallurgical coke without the admixture of varying proportions of straight coking coal.

very short time in the life of a country. It is, therefore, of the utmost importance to locate, if possible, other sources of coking coal and to study the preparation necessary to render the coal suitable for metallurgical purposes.

The survey activities of the Mines Department has resulted in the discovery of one potential source of coking coal viz. the Waterberg coalfield. Tests carried out at the Fuel Research Institute suggests that this coal would have to be crushed to a fairly fine size and washed to a high discard to recover an acceptable coking coal.

Float and sink tests carried out at the Fuel Research Institute [2] [3] on natural fine coal produced in the Witbank coalfield, showed that, although these coals were normally non-coking, a coking fraction amounting to about 25 to 30 per cent of the raw coal could be produced by effecting a separation at 1.35 specific gravity. The float coal obtained had well developed swelling properties and appeared to be a straight coking coal suitable for metallurgical purposes. By effecting a separation at 1.4 specific gravity, the yield of float coal was about 50 per cent on the average, and seemed well suited as a blend coking coal. The coking fractions obtained in this way had the added advantage of low ash content, being of the order of 6 per cent. It was also found that a steam coal of acceptable quality could be produced from the sink material by re-washing it at about 1.58 specific gravity, the refuse only amounting to some 14 per cent of the raw coal. This work indicated, then, that Witbank fine coal is another potential source of coking coal and is more accessible at present than the Waterberg coal. Only a relatively small proportion of it would represent waste material if the coal were utilised in the manner suggested.

It should be explained at this stage, that the coking fraction in Witbank coals is only present in reasonable proportions in the fines naturally produced during mining. The coal

seems..../

seams found in the Witbank area, generally contain a relatively low proportion of the coking constituent, or bright coal. The bright coal is, however, more friable than the dull coal and consequently breaks more readily during mining and handling operations. As a result of this selective breakage, the bright coal becomes more concentrated in the smaller sizes.

It has been estimated that it is possible to recover nearly 1 million tons of coking coal per annum for the next 50 to 100 years by applying two stage washing to the fine coal produced in the Witbank and Eastern Witbank areas i.e. the proved coking coal reserves could be increased by about 30 per cent. At present, more than 700,000 tons of Witbank duff coal are being dumped per annum for the lack of a market. Since this represents an appreciable amount of irreplaceable coking coal going to waste, the importance and urgency of investigating the recovery of this material will be appreciated.

Before either of these sources of coking coal can be exploited, it is necessary to establish, both from the technical and economic points of view, whether the material can be treated on a commercial scale. In the present paper the technical aspect of the problem is examined.

THE PROBLEM.

In both cases a comparatively fine coal has to be treated and high discards have to be dealt with.

In order to appreciate the implications of the problem, some general remarks regarding washers and washing may be given.

According to Bird[4], the ± 0.10 specific gravity distribution at the desired specific gravity of separation is a measure of the difficulty of washing a coal. This value is readily obtained from the washability curves of the coal, and merely represents the fractional yield between specific gravities $(\bar{x} + 0.1)$ and $(\bar{x} - 0.1)$, where $\bar{x}$ is the specific gravity under consideration. On this basis, Bird gives the

following..../

following figures as a rough indication of the difficulty of a washing problem:

±0.10 S.G. Distribution per cent.	Degree of difficulty	Preparation.
0 to 7	Simple	Almost any process, high tonnages
7 to 10	Moderately difficult	Efficient process, high tonnages.
10 to 15	Difficult	Efficient process, medium tonnages, good operation.
15 to 20	Very difficult.	Efficient process, low tonnages, expert operation.
20 to 25	Exceedingly difficult.	Very efficient process, low tonnages, expert operation.
Above 25	Formidable	Limited to a few exceptionally efficient processes, expert operation.

Coal washers in general use may broadly be grouped into three classes, as follows :-

(a) Processes which make use of the influence of density and size of particles on the rate of settling when suspended in a fluid. This group includes jig washers, trough washers, upward current classifiers and concentrating tables. With the exception of concentrating tables, which are limited to coal smaller than about ¼ inch, these washers are capable of handling any practical size of coal, but are best used to effect separations at specific gravities where the ±0.10 specific gravity distribution is less than about 15 per cent. Separations may be effected at more difficult specific gravities but only at the expense of either efficiency or capacity. Generally speaking these washers are unsuitable for exceptionally "difficult" separations and the feed should also be fairly closely sized for best results.

(b).../

(b) Processes in which separation takes place in a fluid of high specific gravity such that the clean coal will float and the refuse will sink. The high specific gravity fluid may be either a true liquid or a suspension of solid particles in water. Since suitable heavy liquids are expensive, commercial processes of this type employ suspensions of various solids in water and are known as "Heavy Medium" or "Dense Medium" washers. These processes are extremely efficient and are consequently suitable for separations which would be classed as formidable according to Bird's scale. The conventional types of heavy medium washers, however, are usually limited in practice to coal coarser than about 1/8". The reason for this will be explained later.

(c) Processes which depend on properties other than specific gravity of the particles to effect a separation. Froth flotation is the most important in this group and employs differences in the surface wetting properties of coal and refuse to effect a separation. This process may be applied to coal finer than about 1/8", and is extensively used in Europe as an ancilliary to other washing operations. The capital and operating costs are very high compared with those of other processes in general use.

Aspects of the Problem of Washing Witbank Duff Coals :-

Washability curves of a typical Witbank duff coal are shown in Fig.1. The solid portion of the ±0.10 specific gravity distribution curve was obtained in the usual manner, while the dotted portion was determined by taking smaller specific gravity increments and then scaling these up to ±0.10 specific gravity. This procedure is probably not quite accurate but it gives one some idea of the equivalent distribution in the specific gravity range below 1.375. It is customary to apply a correction for

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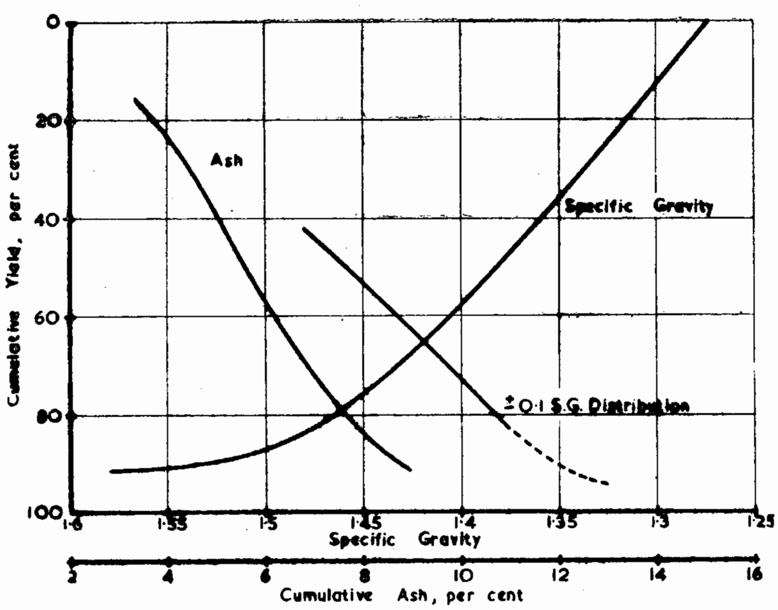

FIGURE I

Washability curves of a typical Witbank Duff.

material having specific gravity greater than 2.0, but this has been neglected in the present example.

It will be clear from the washability curves illustrated, that Witbank duff coals have extremely high ±0.10 specific gravity distributionsin the region of 1.35 to 1.4 specific gravity,(which is the gravity range where a cut will have to be made to recover the coking fraction). As will be seen, this value can easily be of the order of 80 per cent. The separations required to recover the coking fraction from these duffs, therefore, represent a formidable washing problem, and since jigs, trough washers, etc., are clearly unsuitable, it would appear necessary to turn ones attention to a heavy medium type of washer.

The washing of these coals is, however, rendered even more difficult by the fact that they contain about 50 per cent of material finer than 1/8", and this again excludes conventional heavy medium washers. It may be possible to effect the desired separation by means of froth flotation, but as this is an expensive process it could only be considered as a last resort.

As it was evident that all the established processes, with the exception of froth flotation, would be unsuitable for the separations in view, the necessity to study the various newly developed fine coal washers described in the literature was apparent. The cyclone washer was finally selected as holding out most promise, and it was decided to investigate the possible application of this process to Witbank fine coal.

DETAILS OF THE EXPERIMENTAL WORK AND METHOD OF EVALUATING RESULTS.

THE CYCLONE WASHER.

The cyclone washer was developed at the Dutch State Mines and was first described by Driessen,[5] in September 1945. This process is, in principle, a heavy medium washer, but differs from the conventional type in that large centrifugal

forces..../

forces are employed to effect the separation of small particles.

The advantage of employing centrifugal force lies in the fact that the separation of small particles may be accelerated in this manner. Thus, the time required to effect a separation depends on the separating velocities of the particles, and the latter is determined by the size of the particles, the viscosity of the bath and the magnitude of the force acting on the particles (see page 100). Assuming other variables to be constant, the separating velocity decreases with decreasing particle size and the time required for separation increases accordingly. Now, it will readily be accepted that the size of the coal cleaning plant required for any specified throughput, and consequently the cost of cleaning, will vary directly as the time required for separation and, therefore, inversely as the size of the coal to be treated. Since the time required for the separation of small particles is quite considerable under static conditions, it is uneconomical to wash fine coal in conventional heavy medium washers. However, it is possible to increase the separating velocities of small particles by increasing the force acting on them. This is achieved in the cyclone by rapidly rotating its contents, resulting in centrifugal forces equal to 1000 to 2000 times gravity acting on the small coal and shale particles.

<u>The Working Principle</u>[5]

The cyclone consists of a truncated cone with a flat cover having a central outlet, A, and a tangential feed pipe, B, just beneath the cover, as shown in Fig. 2. This arrangement will be recognised as being similar to that of a cyclone dust collector. A suspension of small particles (barytes, magnetite etc.) in water is supplied to the feed pipe under pressure and enters the cyclone with a high inlet velocity, thus giving the fluid a rapid rotational motion. A vortex is created by the rotating fluid, which is spread through the upper and lower openings to the outside, and a vertical air column is created inside the cone. Due to the centrifugal force acting on them,

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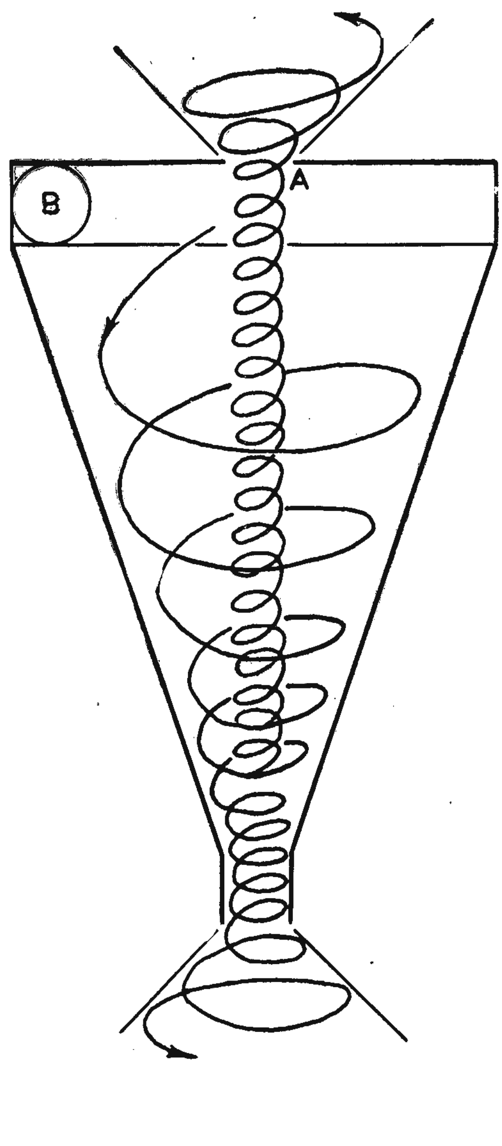

FIGURE 2

OPERATING PRINCIPLE OF A CYCLONE.

the heavy particles tend to move towards the cone wall and down towards the apex. Here, a proportion leaves the cyclone and the rest enters the vortex and either leaves the cyclone at A or is again forced towards the wall. Some of the suspended particles, therefore, are circulating in the cyclone and it is thus possible for the specific gravity of the suspension in the cone to be higher than the specific gravity of the suspension at the inlet. The motion of the heavy particles in the cyclone is influenced by two principal forces viz. the centrifugal force acting radially outwards and the drag due to water flowing radially inwards. The relative magnitudes of these forces at various points in the cyclone determine the path of the particles (i.e. whether the particle is forced towards the wall or leaves in the vortex) and consequently the effective specific gravity in the cone.

If the raw coal (a mixture of coal and shale, say) is now admitted with the suspension of heavy medium, the coal particles lighter than the effective specific gravity of the bed are unable to penetrate it and are swept away in the vortex to the outlet A. The shale particles, on the other hand, are able to penetrate the bed on account of their greater specific gravity and leave at the apex of the cone. Similar operation may be obtained by using a true liquid of appropriate specific gravity in place of a heavy medium suspension. However, as suitable liquids are expensive their use is not considered to be practical.

As far as is known, no exact theory of the motion of the fluid and suspended particles in a cyclone has yet been formulated. Driessen[(6)] has endeavoured to analyse the motion of the fluid in a cyclone mathematically and, although this work is of considerable interest, it has not yet been developed to the stage where it may be considered to be of practical value.

In..../

In the case of a non-viscous fluid, the relationship between the tangential velocity and the radius of rotation is given by the expression $VR = \text{constant}$

where V = tangential velocity

and R = radius of rotation.

However, according to Krijgsman[7], it has been found experimentally that the flow of water in a hydraulic cyclone is approximated more closely by the formula -

$VR^n = \text{constant}$, where $n = 0.5$. This formula may thus be used to estimate the tangential velocity at various points in the cyclone and hence the centrifugal force acting on a particle.

<u>LABORATORY CYCLONE PLANTS ERECTED AT THE FUEL RESEARCH INSTITUTE.</u>

<u>Three Inch Diameter Laboratory Cyclone Coal Washer.</u>

In Driessen's original paper[5], the results of several tests on a cyclone washer were given. From these tests it appeared that the cyclone was able to effect exceptionally sharp separation for particles down to about ½ m.m., the separation equalling the exactness obtained in conventional heavy medium washers treating coarse coal. Details of the design of the cyclone unit and its operation were not given however, and it was not possible, therefore, to confirm Driessen's claims and hence to determine the suitability of this process for treating Witbank duff coal. As the selection of a suitable washer was a matter of some urgency, it was decided to undertake the necessary experimental work in order to determine these data.

A report[8] on some tests carried out on a 3 inch diameter cyclone by <u>Geer</u> and <u>Yancey</u> at the U. S. A. Bureau of Mines was received before the experimental work was started. This work was of a preliminary nature and the paper consequently contained no conclusive data on the influence of the several variables. It formed, however, an invaluable basis

for.../

for the more detailed investigations reported in the present paper.

A 3 inch diameter cyclone, as used by the U. S. A. Bureau of Mines, was duly constructed and several tests were carried out on minus 1 mm plus 60 mesh coal. At first considerable difficulty was experienced in keeping the coal and heavy medium pulp in suspension and this caused blockage of the pipes, etc., and segregation of the heavier feed particles. As the operators gained more experience, these difficulties were overcome and a satisfactory set-up was developed.

Although the results obtained were most promising, it was considered that the unit was too small to be indicative of commercial scale operation. As fairly large quantities of washed Witbank duff coal were also required for coking tests, it was decided to build a larger cyclone plant capable of washing raw duff coal. It was considered that the use of the larger plant would also expedite the investigation.

Nine and a half inch diameter Laboratory Cyclone Coal Washer.

The first 9½ inch diameter cyclone unit which was constructed is shown in Fig. 3. This is a scale model of the 3 inch unit used by Geer and Yancey, except that the diameters of the feed pipe and of the opening in the apex of the cone were increased to minimise the risk of a stoppage. The conical portion and feed pipe were fabricated from brass and were accurately machined internally to the dimensions shown, while the cover plate and washed coal receiver were constructed of steel. A set of orifices and a set of nozzles having different apertures were also provided. The nozzles were made of brass and the orifice discs of stainless steel. The design of the cyclone was later modified when the importance of certain details became apparent. These alterations are fully discussed in the text.

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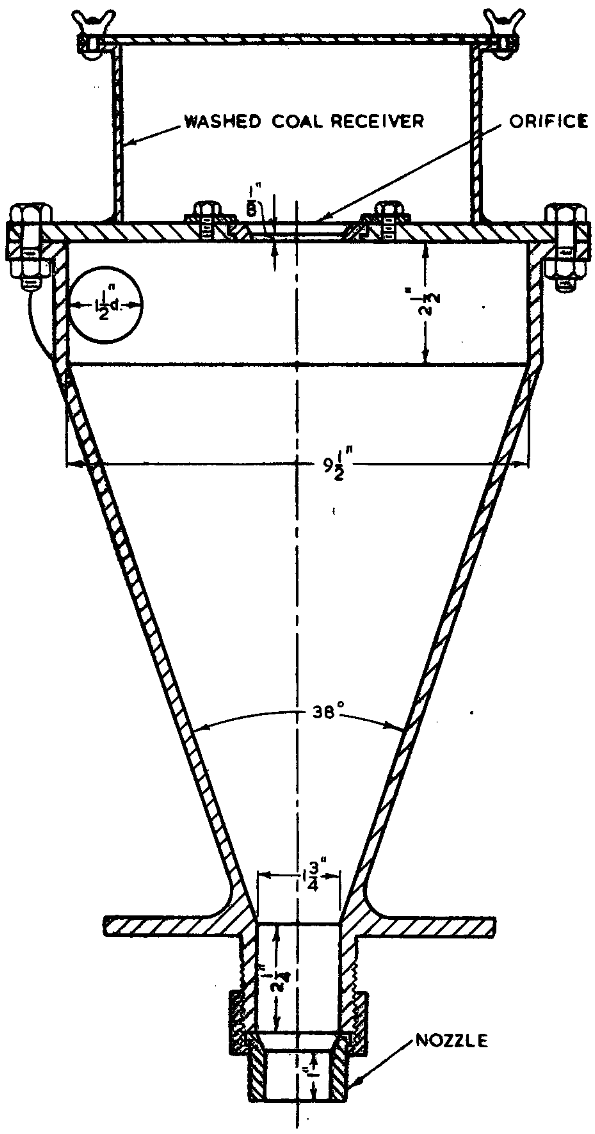

FIGURE 3

38° CYCLONE.

The general arrangement of the plant is shown diagrammatically in Fig 4, and in a photographic view in Fig. 5. A three inch sands pump circulates the pulp in the feed tank (which is of 600 gallons capacity), and is used to ensure that the coal remains in suspension. This pump is driven by an 8 H.P. motor. The pulp is agitated by four stirrers of the simple turbine type driven by a 5 H.P. motor. Each stirrer is 12 inches in diameter, and rotates at 200 R.P.M. The cyclone is supplied by a 2 inch sands pump driven by means of a 19 H.P. paraffin engine, the feed pressure being varied by altering the speed of the engine. It was found that any desired feed pressure up to about 40 lb. per sq. inch could be maintained by manually adjusting the engine throttle. The feed pressure is measured by means of a pressure gauge fitted to the feed pipe at a point approximately $8\frac{1}{2}$ inches from the cyclone inlet. When making any alterations to the cyclone's operating adjustments, the feed to the cyclone is by-passed to the feed tank by means of the valves indicated. Continous flow through the feed pump is, thus, ensured. The washed coal and tailing discharging from the cyclone are led to a sampling device from which they are returned to the feed tank; by operating a lever on this device, the two streams may be diverted to sample containers and thus be collected separately.

Testing Procedure.

A quantity of water is run into the feed tank by means of a hose and its volume is determined using a calibrated dipstick. The pumps and stirrers are then started and heavy medium is added to the water until the desired specific gravity of suspension is obtained. The specific gravity of the suspension is readily found by drawing off a sample of known volume from a cock fitted to the circulating pump discharge. From the volume of the water and the final specific gravity, the weight of suspension in the tank can

be.../

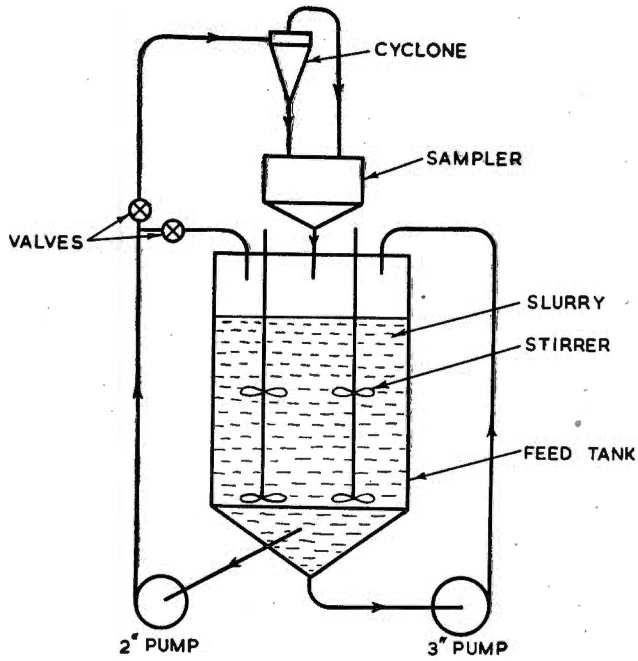

FIGURE 4

Diagrammatic Arrangement of $9\frac{1}{2}$"
diameter Cyclone plant.

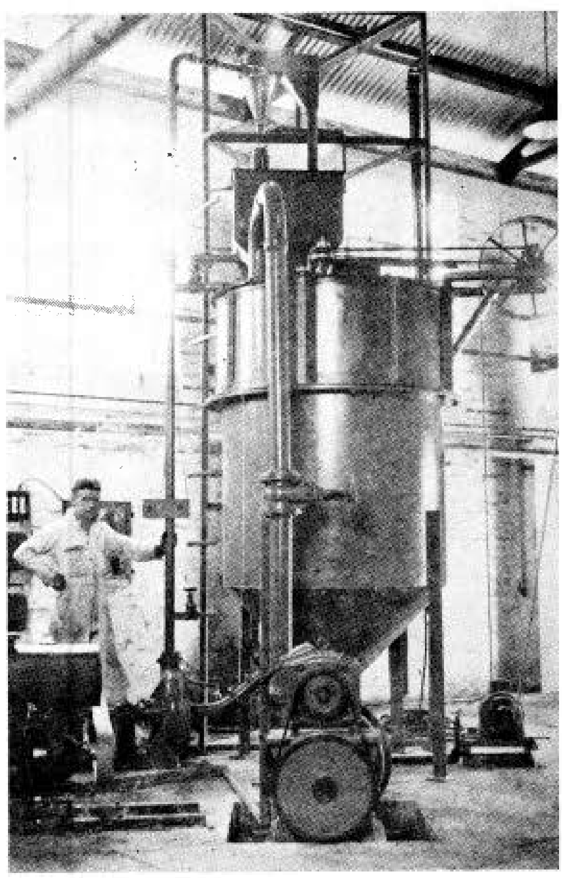

FIGURE 5.

Photographic view of $9\frac{1}{2}$" diameter
Cyclone Plant.

be calculated. If wet medium is used to prepare the suspension, it is more accurate to stop the pumps and stirrers momentarily in order to measure the volume of the suspension in the feed tank. Having estimated the weight of the suspension, the appropriate quantity of coal is added in accordance with the pulp ratio required.

When the orifice and nozzle to be tested have been fitted in place, the feed is admitted to the cyclone and the pressure is adjusted to the required value. The product and tailing discharging from the cyclone are returned to the feed tank for a few minutes before taking samples in order to ensure that conditions in the cyclone are stable. The product and tailing streams are then diverted simultaneously by means of the sampling device to suitable containers for a known period of time. In this way, the throughput of coal and the yield of product are readily determined.

Having obtained the required samples, the heavy medium particles are rinsed off on fine screens using a water spray manually applied. The wash water is collected in a large tank and the medium is allowed to settle. The clear water is finally drained off and the medium is treated in a froth flotation cell before re-use, in order to remove any coal slime with which it may have been contaminated.

After rinsing, the coal samples are drained and air-dried in the sun and are then weighed in order to determine the yield of product. The product and tailing are then sampled in accordance with standard practice for further analysis.

Limitations of the Plant.

In view of the fact that the feed tank is able to hold some 500 to 600 gallons of pulp, it appeared to be possible to carry out several tests from the same batch of feed. It was found, however, that the larger coal particles

degraded.../

degraded progressively if they remained in the apparatus for any length of time. As both the efficiency of separation and the specific gravity of separation are influenced to a certain extent by the size grading of the feed, excessive differences in the grading were not permissible. Although such size degradation was unavoidable, it was found that this effect on a series of tests on one batch became practically negligible if the actual test period did not exceed 15 minutes. In the majority of cases, the tests were completed in less than this time, so the size gradings should not differ appreciably. Coal finer than 1 mm showed negligible size degradation after half an hour or more in the apparatus, and was consequently used in several tests in which it was considered desirable to eliminate this factor altogether.

It was also found that the pulp ratio did not remain absolutely constant for all tests conducted on the same batch of pulp. The pulp flowing through the cyclone during the first test of a series, invariably contained a higher proportion of coal than that passing in later tests. The extent of the variation depended on the size grading of the feed, being more pronounced the coarser the feed. From this it was concluded that the agitation in the feed tank was insufficient to maintain large particles (ca $\frac{1}{4}$ inch) uniformly in suspension. Increase of the agitation would have entailed material alterations to the plant and, as it appeared that the majority of tests were not affected to any great extent by these variations in the pulp ratio, they were usually neglected. Some allowance should, however, be made when studying results of a series of tests.

GENERAL REMARKS ON THE EXPERIMENTAL WORK REPORTED.

Duff coal produced at Landau No. 3 colliery was selected as being typical of the Witbank duff coals and was used in the majority of the tests planned to investigate the influence of the several variables on the cyclone performance. In a few cases, fine coals from other sources were used while awaiting fresh supplies from Landau No. 3 colliery. However, the source from which the feed coal was obtained is clearly stated for each test.

Fine material was removed from the feed before washing, in order to facilitate rinsing of the samples and to minimise the percentage of coal slime contaminating the heavy medium after use. In view of the difficulty of dry-screening large quantities of coal on fine sieves, 1 mm was selected as the lower limit of size for the majority of tests and the samples were rinsed free of medium on 60 mesh screens. Where it appeared desirable to use finer feed, the duff was screened minus 1 mm plus 60 mesh and samples were recovered on 100 mesh screens. Since other feed gradings were used in a few tests, it should be stated that samples were recovered on 100 mesh screens in all cases where the feed contained material finer than 1 mm. In the case of the final washing tests reported, the raw duff was screened at half inch in order to remove abnormally large particles, but the fines were included in the cyclone feed.

Commercial barytes ($BaSO_4$) was used as heavy medium in all the tests reported in this paper. This material was selected as it was readily obtainable in a finely divided state at reasonable cost. The following is an average analysis of the barytes as supplied :-

<u>Chemical analysis.</u>

$BaSO_4$, 90 to 92 per cent
SiO_2, 6 to 8 per cent
Fe_2O_3, 1 per cent

Specific gravity, 4.17.

<u>Sieve analysis.</u>

+200	B.S.S.	nil
-200 +240	B.S.S.	0.76 per cent.
-240	B.S.S.	99.24 per cent.

Fine.../

Fine coal is readily removed from barytes by means of froth flotation, a mixture of 5 parts paraffin to 1 part by volume of pine oil being found to be a suitable reagent. (Reagent consumption is in the order of 2 lb. per ton of contaminated barytes).

As stated before, the samples of product and tailing were air-dried in the sun as other facilities were not available. The moisture content of some samples was determined and was found to be of the order of 2 per cent and has consequently been neglected except in the case of the final washing tests for which these values are recorded. All analyses are, however, expressed on an air-dried basis.

Mixtures of carbon tetrachloride and benzol were used for float and sink analysis of samples in all cases. These liquids evaporate readily from coal and their use thus simplifies drying of the samples after analyses.

EVALUATION OF THE WASHING TESTS.

An investigation of this nature requires one to determine the relative importance and effect of each operating variable by conducting a series of tests in each of which there should be, as far as possible, only one variable.

The relative importance of any variable can only be assessed if a reliable basis of comparison is used.

One may consider the determination of the overall efficiency of the process (using one of the recognised formulae developed for this purpose) as a suitable basis. However, it can be shown[9] that the overall efficiency is a function not only of the efficiency of the washer but also of the specific gravity distribution of the feed coal. The overall efficiency could, therefore, only be regarded as a suitable basis of comparison if all tests were conducted on the same coal and separations were effected at the same specific gravity

in..../

in all cases. As this would place an undue limitation on the scope of the investigation, the overall efficiency can not be regarded as a generally satisfactory basis.

In fact, two distinct efficiency values are generally involved, namely the overall efficiency of the process and the "mechanical" efficiency of the washer. For convenience these will be termed "washing efficiency" and "washer efficiency", respectively. In systematic work, as envisaged, the latter factor is of prime importance. Unfortunately, no expression for calculating the "washer efficiency" (as a percentage) could be found in available literature. The so-called "error area" that can be derived from the Tromp distribution factor curve[10] of the washer can, however, be regarded as a measure of the "washer efficiency".

Assessment of the Washer Efficiency.

Since the evaluation of the experimental work described in this paper is based largely on the Tromp distribution factor curves, the procedure for obtaining these may again be briefly described.

The distribution factor curve proposed by Tromp, indicates what percentage of each specific gravity fraction of the feed was recovered in the clean coal (or product) and what was rejected in the tailing. This curve is, therefore, independent of the actual quantity of material present in each specific gravity interval.

In order to determine these percentages, it is necessary, as a first step, to carry out separate float and sink analyses of the washed product and of the tailing. The distribution factor (or percentage recovered) for each specific gravity interval is then calculated from the float and sink data as illustrated by the numerical example shown in Table 1. In this case, the yield of washed product was 63 per cent of the feed coal.

TABLE 1.

Method of Determining the Tromp Distribution Factors.

Specific Gravity Stage	Product (63% of Feed) % of Product	Product (63% of Feed) % of Feed a x 0.63	Tailing (37% of Feed) % of Tailing	Tailing (37% of Feed) % of Feed c x 0.37	Feed % b + d	Distribution Factor % b ÷ e x 100
	a	b	c	d	e	f
To 1.3	13.5	8.51	0.3	0.11	8.62	98.7
1.3-1.35	47.9	30.18	1.8	0.67	30.85	97.8
1.35-1.4	26.9	16.95	5.3	1.96	18.91	89.6
1.4-1.45	8.3	5.23	20.1	7.44	12.67	41.3
1.45-1.5	1.9	1.20	18.0	6.66	7.86	15.3
1.5-1.58	0.8	0.50	20.3	7.51	8.01	6.2
> 1.58	0.8	0.50	34.2	12.65	13.15	3.8
Total	100.1	63.07	100.0	37.00	100.07	-

The distribution factor for each specific gravity interval is now plotted against the mean specific gravity of that interval; thus, the distribution factor for the interval 1.3 to 1.35 would be plotted at 1.325 specific gravity. In the case of most South African coals, it may be assumed that there is no coal having a specific gravity lower than about 1.28, so that the mean of the first stage in the example is taken as 1.29. The distribution factor curve shown in Fig. 6 has been plotted in this manner from the data in Table 1.

Tromp defines the specific gravity of separation as the specific gravity at which half of the material present goes to the product and half goes to the tailing i.e. the distribution factor is 50 per cent at the specific gravity of separation and the latter may thus be read off the curve (1.425 in Fig.6).

The shape of the Tromp distribution factor curve is virtually independent of the specific gravity distribution of the feed and the specific gravity of separation. It is influenced,

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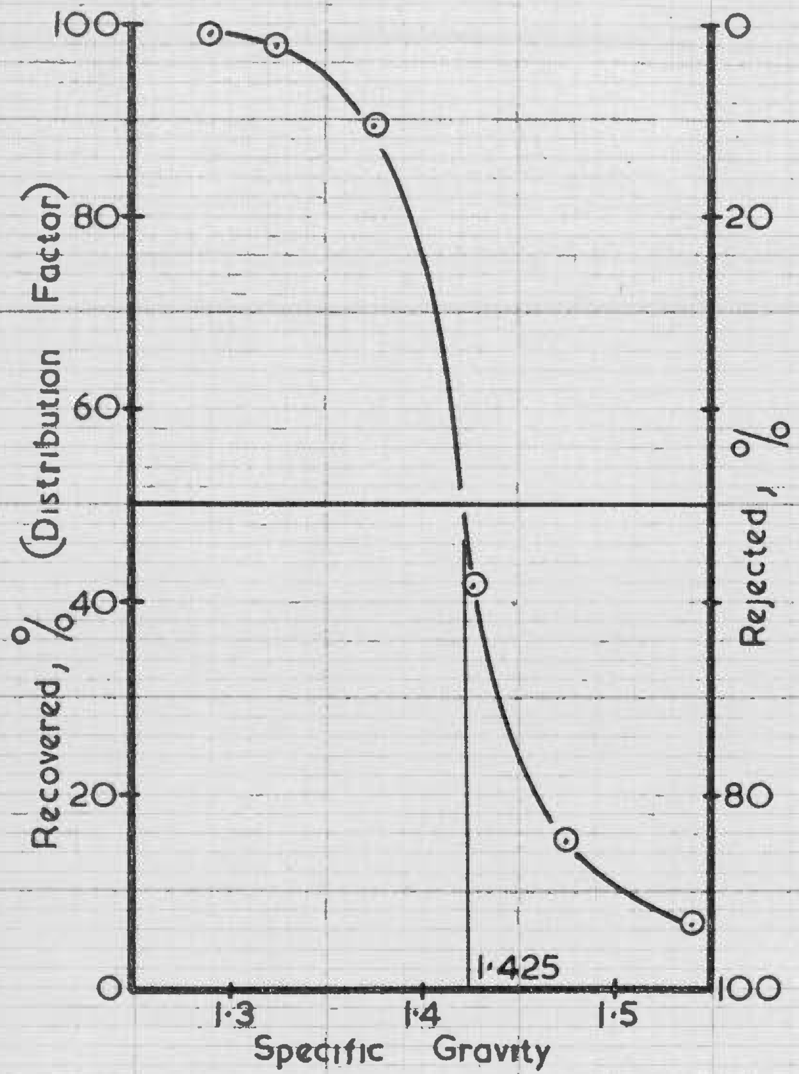

FIGURE 6 Typical Tromp curve

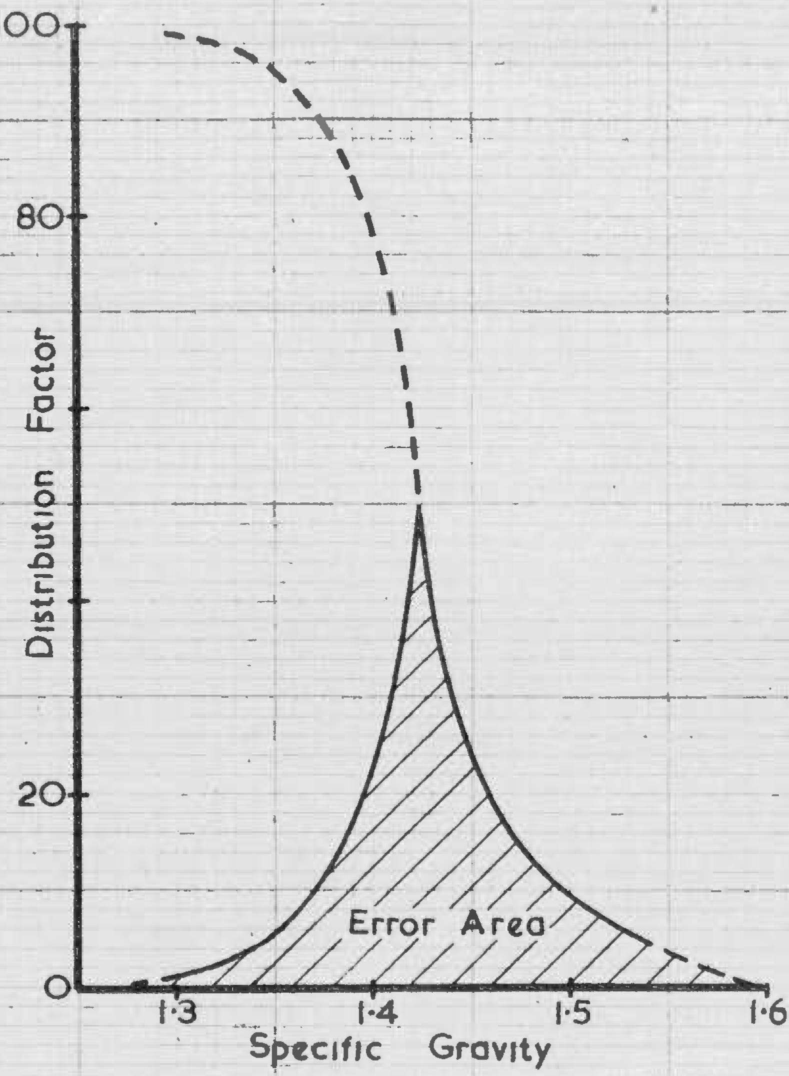

FIGURE 7 Tromp curve showing the Error Area

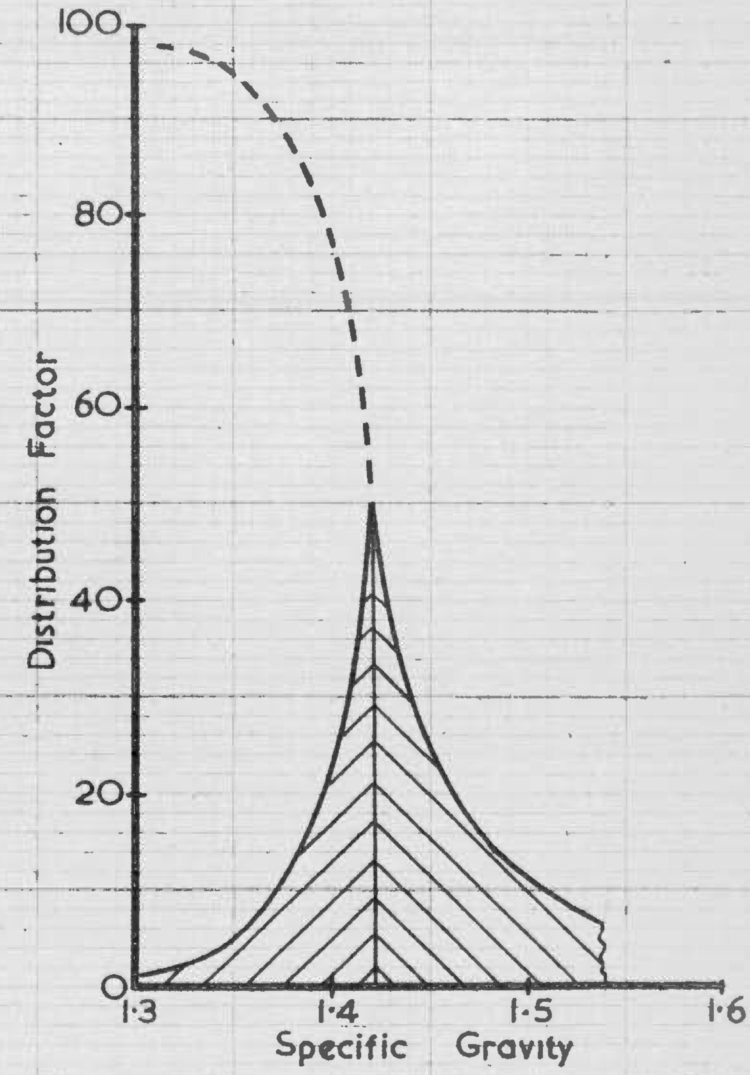

FIGURE 8 Tromp curve for disc orifice

however, by the size grading of the feed and, in the case of some washers, by the load. The load does not affect the distribution factor curve of a heavy medium washer to any extent, provided that the washer is not overloaded. This curve may, therefore, be regarded as a characteristic of a washer for a feed of specified size grading.

Since the distribution factor curve, in effect, shows graphically the deviation from the ideal separation (i.e. a vertical line through the specific gravity of separation), the shape of the curve will clearly be a measure of the washer efficiency. A numerical value for the washer efficiency may be obtained by redrawing the curve as shown in Fig. 7, i.e. the upper portion of the curve in Fig. 6 has merely been rotated through 180 degrees around the 50 percent distribution factor line. The shaded area, usually termed the "error area", will then be a numerical measure of the washer efficiency, i.e. the smaller the error area the greater is the washer efficiency.

In order to obtain the true error area, the complete Tromp curve should be drawn as indicated by the dotted lines in Fig. 7. Theoretically, it is possible to obtain the complete curve mathematically when three points on the curve have been determined by float and sink analysis. In practice it was found that it was not advisable to adopt this principle because float and sink data are not sufficiently accurate, and a truer picture is obtained by determining as many points as possible experimentally. In the case of the tests reported in the present paper, float and sink analyses were carried out at six specific gravities between the limits of 1.3 and 1.58, and a mean curve was drawn through the distribution factor values obtained. The greater part of the Tromp curve could be drawn in this way as the cyclone separations were mostly effected between 1.35 and 1.4 specific gravity and were usually reasonably exact. Float and sink analysis was not continued beyond 1.58 specific gravity as this is the maximum which can be

attained.../

attained with carbon tetrachloride at room temperature. Higher specific gravities would have required the use of expensive liquids, such as bromoform, and this would have severely restricted the number of samples which could be analysed. Since the complete distribution factor curves were not obtained, it was not possible to determine the error areas and their use as a basis for comparison was, therefore, not possible. The curves could have been extrapolated beyond the limits of the float and sink analysis but this was not considered to be justifiable. It was found, however, that significant differences in the washer efficiencies could readily be detected by comparing the shape of the curves. Comparison of the curves is facilitated by transferring one or more of the curves to tracing paper and then superimposing one on the other.

Determination of the Washing Efficiency.

It has been pointed out that the washing efficiency alone is of little value when the influence of alterations to the operating adjustments of a washer are to be assessed. However, as the same coal was used in the majority of tests carried out during the investigation, it was found that the determination of the washing efficiency was of definite value. Thus it could serve as a guide to indicate errors in the analyses. For example, a high washing efficiency would not be consistent with a large error area.

The Fraser and Yancey expression[11] for determining washing efficiency has been used throughout, i.e.

$$\text{Washing efficiency} = \frac{Ya}{Yt} \times \frac{Af - Aa}{Af - At} \times 100\%$$

where Ya = actual yield of product, %

Yt = theoretical yield at the specific gravity of separation, %

Af = ash content of the feed, %

Aa = actual ash content of the product, %

At = ash content of the theoretical product at the specific gravity of separation, %

Definition ..../

DEFINITIONS OF TERMS.

In order to avoid confusion, the terms used in reporting the experimental work have been standardised as follows :-

Diameter of a cyclone : The internal diameter of the parallel portion of the cyclone.

Orifice : The opening in the cover plate of the cyclone.

Nozzle : The opening at the apex of the cone.

Feed Pressure : The pressure measured in the feed pipe at a point approximately $8\frac{1}{2}$ inches from the cyclone inlet.

Specific gravity of suspension : The specific gravity of the heavy medium suspension as prepared in the feed tank.

Pulp Ratio: The ratio between the weight of suspension and the weight of coal in the pulp.

Specific gravity of separation : The specific gravity at which the Tromp distribution factor is 50 per cent. (In a few tests, another method was used, but this is clearly stated.)

Washing efficiency : The value obtained for the overall efficiency using the Fraser and Yancey expression.

Washer efficiency : The efficiency of separation as denoted by the shape of the Tromp distribution factor curve.

Nozzle loading : The quantity of refuse passing through the nozzle in unit time, expressed in tons per hour per inch nozzle diameter.

Product : The washed coal discharged from the orifice.

Tailing : The refuse discharged from the nozzle.

Ton.: 2000 lb.

Capacity or throughput : The quantity of raw coal passing through the cyclone in unit time.

Ash content of feed : The ash content of the feed as calculated from those of the product and tailing obtained.

THE MAIN OPERATING CHARACTERISTICS OF A $9\frac{1}{2}$ INCH DIAMETER CYCLONE WASHER.

SCOPE OF THE INVESTIGATION.

The application of the cyclone principle to the beneficiation of coal is a comparatively recent development and much fundamental and practical research remains to be done. The work is of such magnitude that it is essential at the outset to limit the scope of one's investigation in order to reach some stage of finality within a reasonable period of time.

As explained, the prime object of undertaking investigational work at the Fuel Research Institute was to determine whether the cyclone washer would be a suitable means for recovering the coking fraction from Witbank duff coal. It was consequently decided to give preference to a practical experimental investigation, and to abandon any attempt at a protracted study and mathematical analysis of the motion of solid particles and of the flow of water in a cyclone, although it was realised that such an analysis would be invaluable to a clear understanding of the operation of the washer.

Although all possible avenues have not been fully explored, the final separations which were obtained are considered to be fair indications of the cyclone's potentialities. While no attempt is made to formulate theories regarding the mechanics of the separation in a cyclone, tentative explanations of observed phenomena are offered.

It is hoped that the present paper may constitute a useful basis for further study of the operation of the cyclone washer.

Characteristics .../

CHARACTERISTICS INVESTIGATED.

Apart from economic considerations, the suitability of a washer for any specified separation is determined principally by its capacity and efficiency. The experimental work was, therefore, aimed at obtaining the optimum efficiency and capacity when washing Witbank duff coal in the region of 1.35 to 1.4 specific gravity in the 9½ inch diameter cyclone. It was decided to concentrate on separations in this low specific gravity range, as such separations probably represent the most formidable washing problem likely to be encountered in practice and would, therefore, be the most stringent test for the cyclone washer.

The influence of the following factors on cyclone performance were investigated :-

(1) Design of the orifice.

(2) Relationship between orifice diameter and nozzle diameter.

(3) Specific gravity of suspension.

(4) Pulp ratio.

(5) Orifice diameter.

(6) Relationship between the diameter of the nozzle and the grading of the feed.

(7) Feed pressure.

(8) Feed pipe diameter.

(9) Apex angle of cyclone.

(10) Size grading of feed.

Although some data had been obtained from the tests conducted on the 3 inch diameter cyclone, there was very little conclusive information on the influence of the several variables at the time when the investigation of the 9½ inch diameter cyclone commenced. The investigation was consequently in progress for some time before the relative importance of the variables was fully appreciated and it became possible

to..../

to plan the experimental work in more logical sequence. An outline on all the tests carried out and the conclusions drawn from each will not be attempted, as it is considered that the operating characteristics of the cyclone can be depicted more clearly by dealing with each factor separately in the light of the author's ultimate knowledge of the subject as a whole.

Since the variables are largely interdependent, it is not possible to discuss the importance of each without cross reference to several others. The author has consequently endeavoured to arrange the paper in the most convenient sequence in this respect.

Sufficient experimental data will be found in the body of the paper to indicate general trends. The interested reader will find a full compilation of analytical results in the appendices.

THE INFLUENCE OF ORIFICE DESIGN.

In preliminary tests,[12] a simple disc orifice was used of the type illustrated in Fig.3. When studying the Tromp distribution factor curves obtained for these tests, it was observed that the deviation from the ideal separation was greater for specific gravities in the refuse zone, than it was for specific gravities lower than that of the cutting point.

This is illustrated by the Tromp distribution factor curve for one of these tests (No.11B) shown in Fig 8. It will be quite clear, simply by inspection, that the shaded area to the right of the specific gravity of separation is greater than the area to the left. If the complete Tromp curve could be drawn, the difference in these areas would be quite substantial. This diagram indicates, then that the

deviation..../

deviation is greater for specific gravities higher than the specific gravity of separation. In other words, the percentage of "sink" material recovered in the product would be greater than the "float" lost to the tailing if the specific gravity distribution of the feed were uniform.

This effect could be explained by assuming that some of the feed coal was actually leaving the cyclone through the orifice before a specific gravity separation had been effected. Similar by-passing of material has been observed in cyclone dust collectors and a vortex tube is used to prevent this happening. It was, therefore, decided to determine whether the replacement of the disc-orifice by a tube projecting beneath the cover plate, as shown in Fig. 9, would yield better results. If beneficial, it would then be necessary to determine the optimum length of tube to be used.

A series of tests was accordingly conducted, in which tubes of various lengths were used instead of a disc-type orifice. The results of these tests are reported in Table 2, and the corresponding Tromp distribution factor curves are shown in Fig. 10.

TABLE 2.

The Influence of Orifice Tubes of Various Lengths.

The following conditions were constant for all tests :-

(1) $9\frac{1}{2}$" diameter cyclone, 25° apex angle, $1\frac{1}{2}$" diameter feed pipe.

(2) Orifice tube diameter, d, $1\frac{7}{8}$ inches.

(3) Nozzle diameter $1\frac{5}{8}$ inches.

(4) Feed coal:- Landau No.3, $-\frac{1}{4}$" + 1 mm.

(5) Specific gravity of suspension, 1.28

(6) Pulp ratio, 6 to 1.

(7) Feed pressure $7\frac{1}{2}$ lb. per sq. inch.

Test No..../

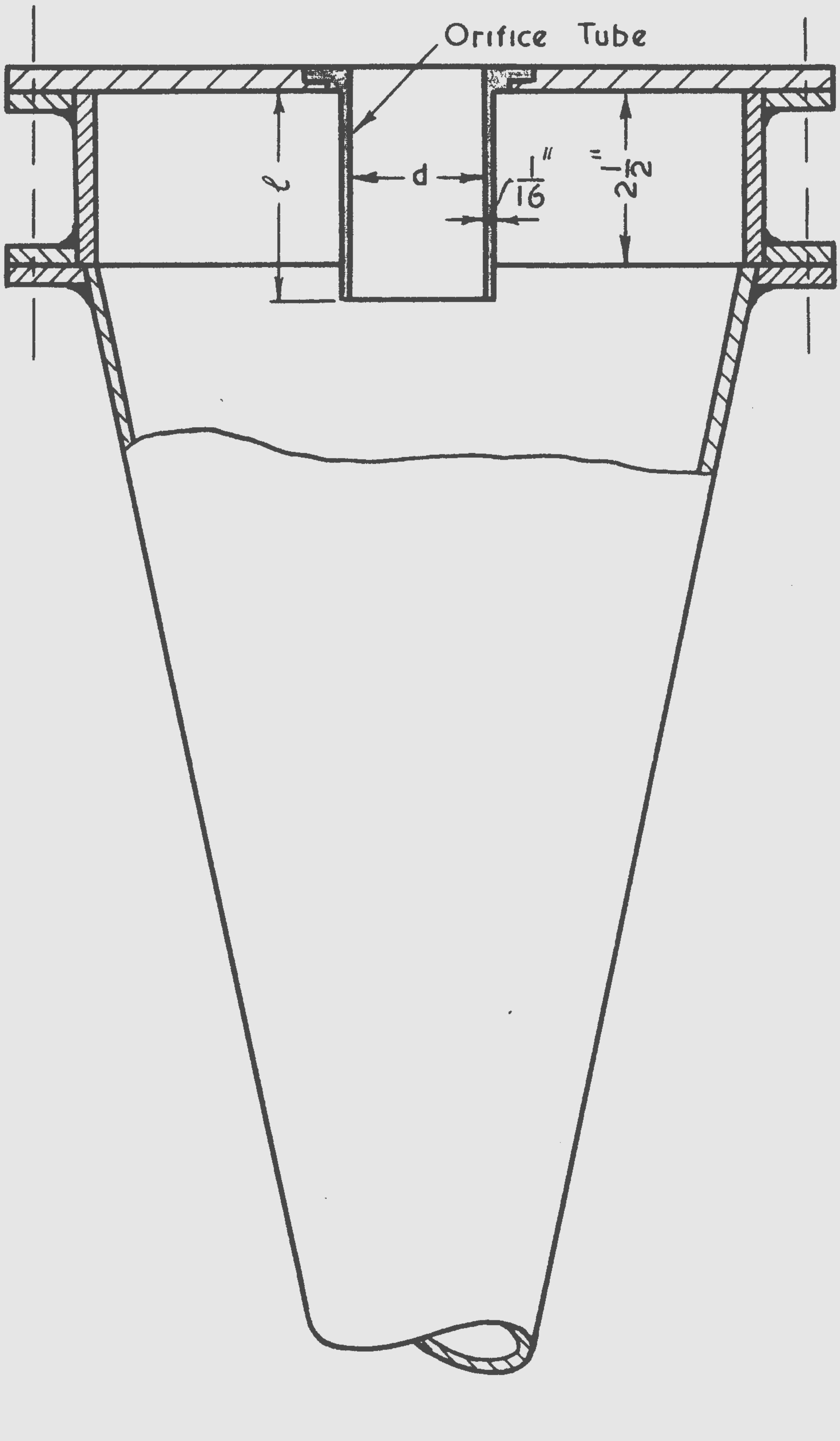

FIGURE 9 The Orifice Tube

Test No.	18 C	19 C	20 D	20 E	20 F	20 G	20 H
Tube length, ℓ, inches.	Disc Orifice	5/8	1 1/4	1 7/8	2 1/2	3 1/8	3 3/4
Yield of product, %	30.8	29.8	34.3	38.3	38.6	41.8	41.9
Ash content of product, %	5.8	5.4	5.6	5.7	5.7	5.8	5.8
Ash content of feed, %	14.4	14.3	14.1	13.4	13.8	13.5	13.6
Specific Gravity of separation.	1.348	1.348	1.358	-	-	1.378	-
Washing efficiency, %	88	91	96	-	-	97	-

Comparing the yield, the ash content and the washing efficiency figures in Table 2, it appears that the use of the tube improved the performance materially. However, the Tromp distribution factor curves do not show as marked a decrease in the deviation for specific gravities above that of the cutting point as was anticipated. Variation in tube length above 5/8 inch did not appear to influence the efficiency of separation to any appreciable extent, but a tendency towards increasing specific gravity of separation with increase of tube length is evident. Tests 20D to 20H were carried out on the same batch of feed coal in the order indicated and, as the various tubes had to be fitted in place after each test, the total time taken for the series was greater than usual. The feed probably degraded to a considerable degree during this period and any improvement in the efficiency may thus have been obscured. Degradation of the feed may also account for the increase in the specific gravity of separation observed in Table 2.

In order to confirm that the use of an orifice tube does actually improve the efficiency by decreasing the percentage of heavy material recovered in the product, the tests shown in Table 3 and Fig.11 were carried out. Tests 21A and 21B were conducted on the same batch of feed coal, but since the samples were taken within a few minutes of one another, the difference in feed size is probably negligible. This also applies to tests 21G and 21H, for which tests a fresh batch of pulp was used.

TABLE 3..../

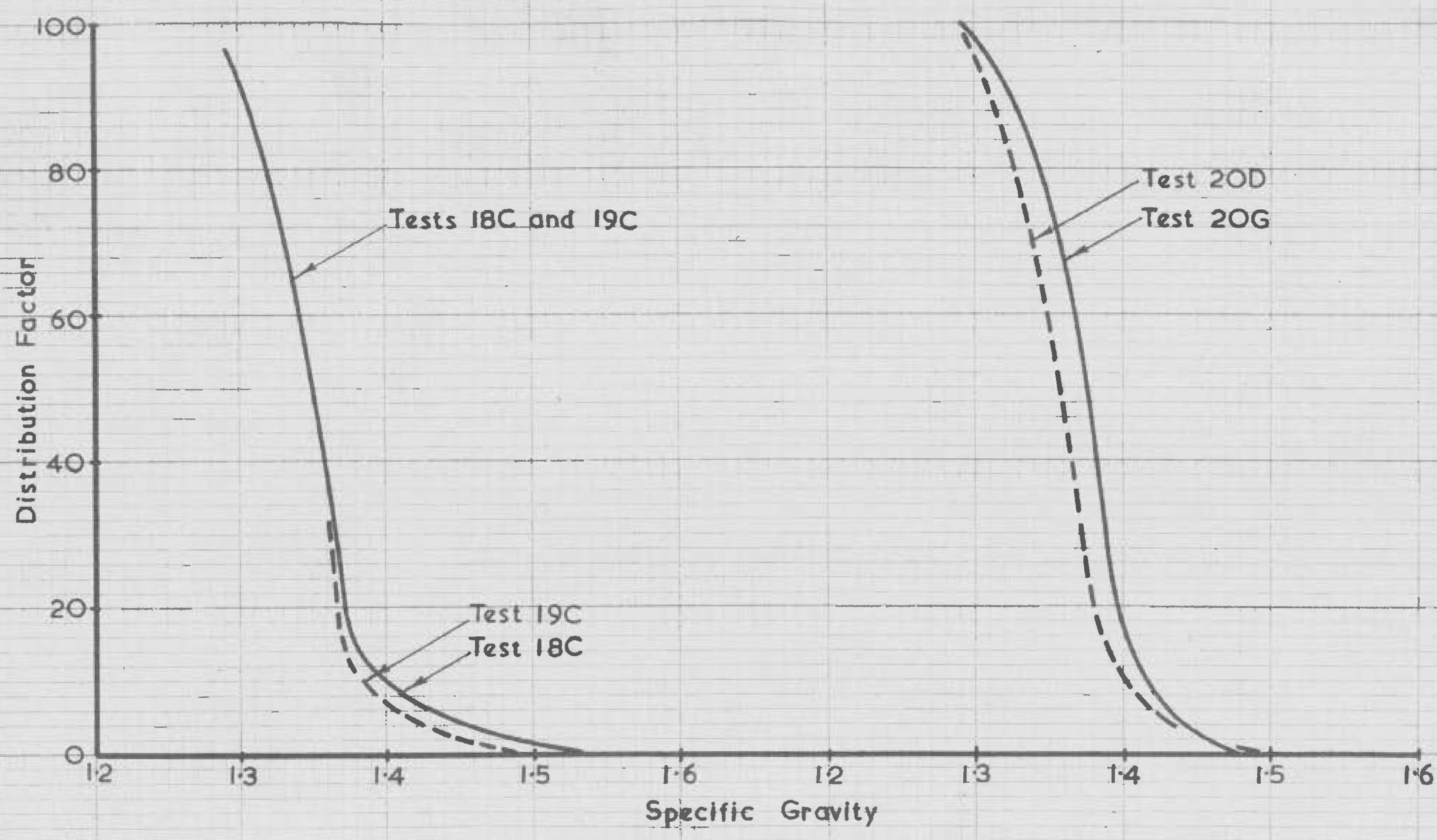

FIGURE 10 Tromp curves relating to Table 2

TABLE 3.

The Influence of an Orifice Tube on the Efficiency of Separation.

The following conditions were constant for all tests.

(1) $9\frac{1}{2}$ inches diameter cyclone, $1\frac{1}{2}$ inches diameter feed pipe, apex angle as indicated.

(2) Orifice tube diameter, d, $1\frac{7}{8}$ inches.

(3) Feed:- Landau No.3 duff, $-\frac{1}{4}$ inch plus zero, plus 100 mesh recovered.

(4) Specific gravity of suspension, 1.28.

(5) Pulp ratio 6 to 1.

(6) Feed pressure, $7\frac{1}{2}$ lb. per sq. inch.

Test No.	21 A	21 B	21 G	21 H
Apex angle, degrees.	25	25	38	38
Orifice tube length, ℓ, inches.	disc Orifice $\ell = 0$	$1\frac{1}{4}$	disc Orifice $\ell = 0$	$1\frac{1}{4}$
Nozzle diameter, inches.	$1\frac{5}{8}$	$1\frac{5}{8}$	$1\frac{3}{4}$	$1\frac{3}{4}$
Yield of product, %	38.3	38.5	34.2	30.0
Ash content of product, %	7.3	6.0	7.2	6.2
Ash content of feed, %	15.2	14.7	15.0	14.7
Specific Gravity of separation	1.365	1.37	1.355	1.348

The results of the tests reported in Table 3 clearly indicate that the efficiency of separation is substantially improved by fitting an orifice tube in lieu of a disc-type orifice. Thus, the yields of product obtained in Tests 21A and 21B are practically the same, but the ash content is considerably lower in test 21B in which the orifice tube had been used. Similar results will be observed in the case of tests 21G and 21H.

If the Tromp distribution factor curves for these tests are compared (Fig.11), it will be noted that the shape of the upper portion of the curves virtually remains unchanged, while the deviation of the lower portions is decreased for the tests in which the orifice tube was fitted. This shows that the recovery of "light" coal is not affected by the tube, but that

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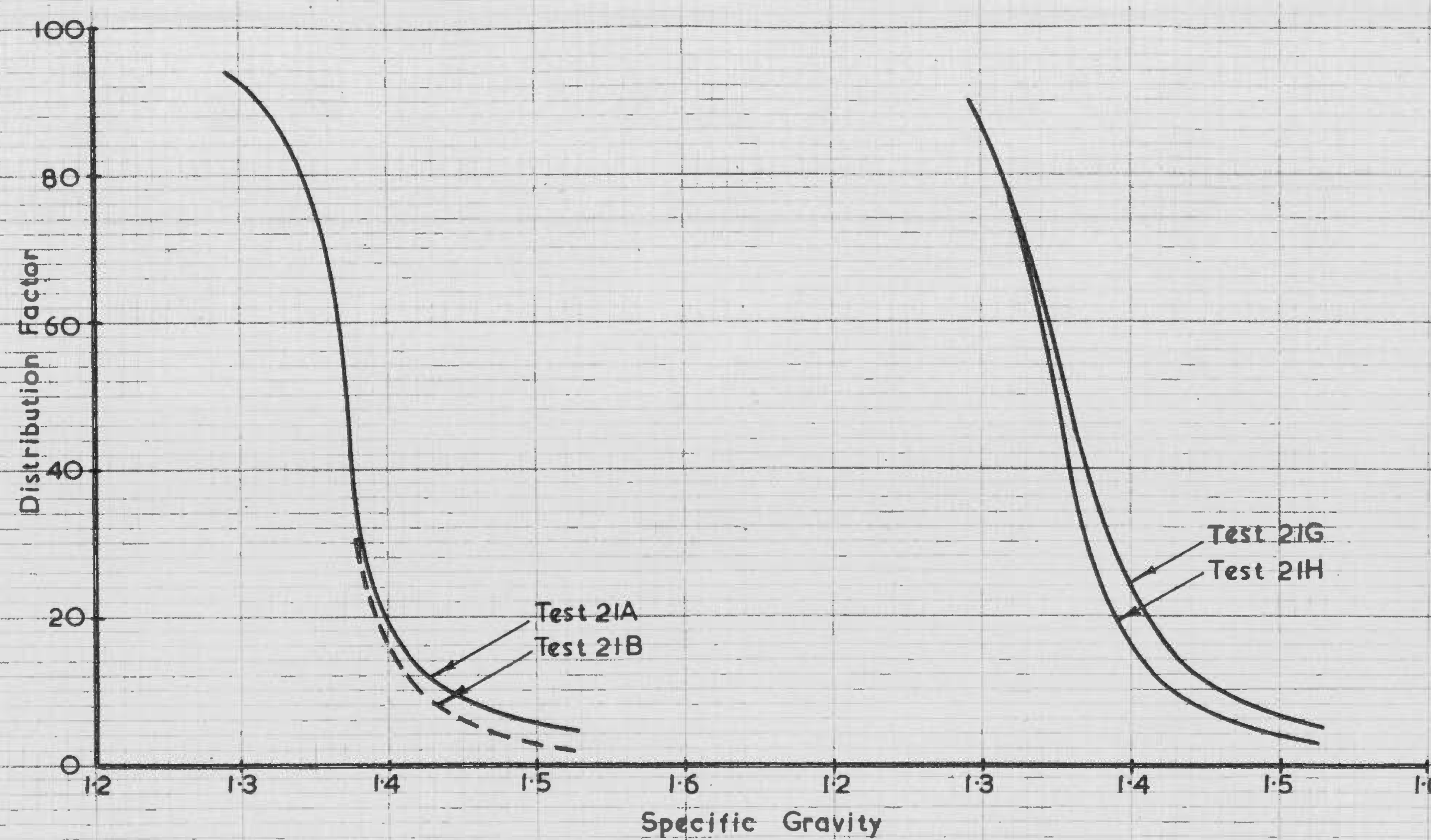

FIGURE 11 Tromp curves relating to Table 3

the recovery of "heavy" material is decreased.

At that stage, it had been proved that the tube had the desired effect but it still remained to determine the optimum length and to investigate the influence of the tube length on the specific gravity of separation. Fine feed coal was selected for these tests in order to minimise the degradation effect during the experiment. Details of the tests will be found in Table 4 and the Tromp distribution factor curves are shown in Fig.12.

TABLE 4.

The Influence of Orifice Tube Length on Cyclone Performance.

The following conditions were constant for all tests:-

(1) $9\frac{1}{2}$ inches diameter cyclone, 25 degrees apex angle, $1\frac{1}{2}$ inches diameter feed pipe.

(2) Orifice tube diameter, d, $1\frac{7}{8}$ inches.

(3) Nozzle diameter, $1\frac{5}{8}$ inches.

(4) Feed coal :- Landau No.3 duff, -1 mm +60 mesh.

(5) Specific Gravity of suspension, 1.3

(6) Pulp ratio, 6 to 1.

(7) Feed pressure, $7\frac{1}{2}$ lb. per sq. inch.

Test No.	31 T	31 U	31 V	31 W	31 X	31 Y	31Z	31AA
Orifice tube length inches.	0	$\frac{5}{8}$	$1\frac{1}{4}$	$1\frac{7}{8}$	$2\frac{1}{2}$	$3\frac{1}{8}$	$9\frac{1}{2}$	$16\frac{1}{2}$
Yield of product,%	41.4	37.1	44.5	46.1	45.4	47.6	48.0	31.3
Ash content of product,%	6.5	5.9	5.6	5.9	5.7	5.7	5.7	8.8
Ash of feed, %	14.9	15.0	13.8	14.3	14.7	14.3	-	15.0
S.G.of separation	1.375	1.365	-	1.395	1.39	1.395	1.4	1.32

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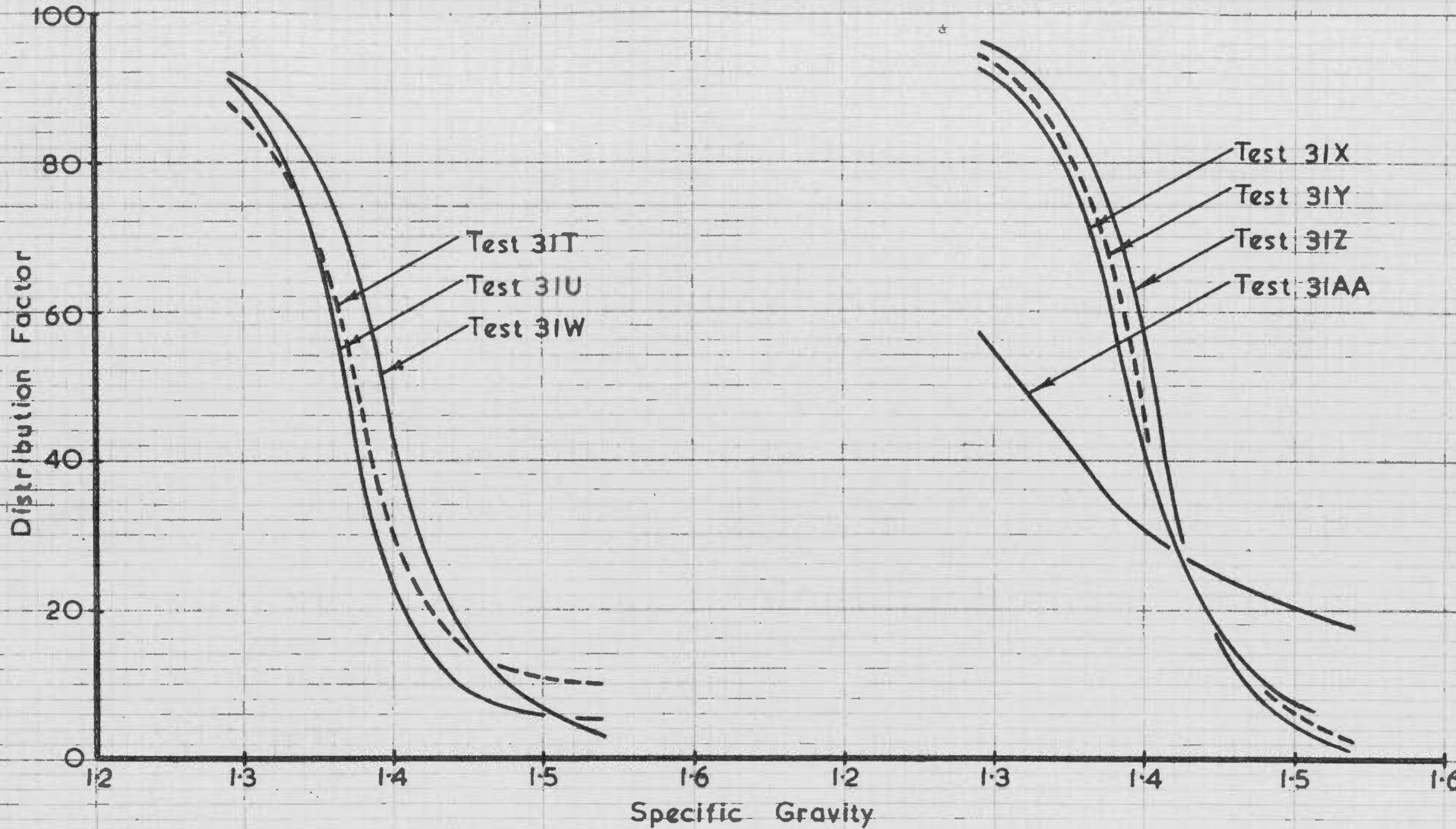

FIGURE 12 Tromp curves relating to Table 4

Since the washability curves for this series of tests (see appendix 3) show that the composition of the feed did not vary materially, the yields and ash contents of the products are a measure of the cyclone performance. These figures indicate that the efficiency of separation increased as the tube length was increased from zero to about 1¼ inches and then remained substantially constant for further increase in length to 9½ inches. A tube 16½ inches in length evidently gave rise to disturbed conditions as will be seen from the poor results obtained.

The Tromp distribution factor curves for these tests (Fig.12) show the same tendency. The deviation at the higher specific gravities has been decreased by increasing the tube length from zero to 1⅞ inches and the shape of the curves then remains practically constant for tubes up to 9½ inches in length. Unfortunately the samples of test 31V were spoiled during float and sink analysis, but the yield and ash figures suggest that the Tromp curve would not differ appreciably from that of, say, test 31X.

Slight variation in the specific gravity of separation will be noted for tube lengths up to about 1¼ inches (Tests 31T, 31U and 31V). This is ascribed to experimental errors since the specific gravity of separation appears to be reasonably constant for greater tube lengths, test 31AA being excluded. The tendency towards increase of specific gravity of separation with increasing tube length, which was observed in Table 2, is probably due to the effects of degradation.

From this work it appears that the tube length is not critical provided that it exceeds a minimum length of about 1¼ inches and is not abnormally long (since a tube of 16½ inches in length appears to give rise to disturbed conditions). The fact that a tube only 5/8 inch in length improves the efficiency noticeably, suggests that material which tends to by-pass the cyclone does so just beneath the cover plate. Since the length

of..../

of the parallel portion of the cyclone was $2\frac{1}{2}$ inches, a practical rule would be to make the tube length about equal to the length of the parallel portion, thus satisfying the terminal conditions.

In a patent application[13] by the Dutch State Mines, a sketch of a cyclone is shown in which the feed pipe is inclined downwards at an angle of approximately 5 degrees to the plane of the cover plate. It has not been considered worth while to determine the influence of this modification and its purpose is not stated, but it is presumably intended to decrease the tendency of the feed to by-pass the cyclone. Inclination of the feed pipe may also improve the flow characteristics of the cyclone by minimising shock losses, etc.

THE INFLUENCE OF THE NOZZLE AND ORIFICE COMBINATION.

The influence of the relationship between the diameter of the orifice and the diameter of the nozzle on the performance of the cyclone is illustrated by the series of tests reported in Table 5. For convenience, the diameter of the nozzle was varied in these tests while all other operating adjustments were kept constant. To facilitate interpretation of the results, specific gravity of separation has been plotted against nozzle diameter in Fig. 13.

TABLE 5.

The influence of the Nozzle diameter when all other variables are constant.

The following conditions were constant for all tests :-

(1) $9\frac{1}{2}$ inches diameter cyclone, 25 degrees apex angle, $1\frac{1}{2}$ inches diameter feed pipe.

(2) $1\frac{7}{8}$ inches diameter orifice tube projecting $1\frac{1}{4}$ inches beneath cover plate.

(3) Feed pressure, $7\frac{1}{2}$ lb. per sq. inch.

(4) Specific gravity of suspension, 1.3.

(5) Feed coal; Landau No. 3 duff, $-\frac{1}{4}$" +1 mm.

(6) Pulp ratio, 6 to 1.

Test No..../

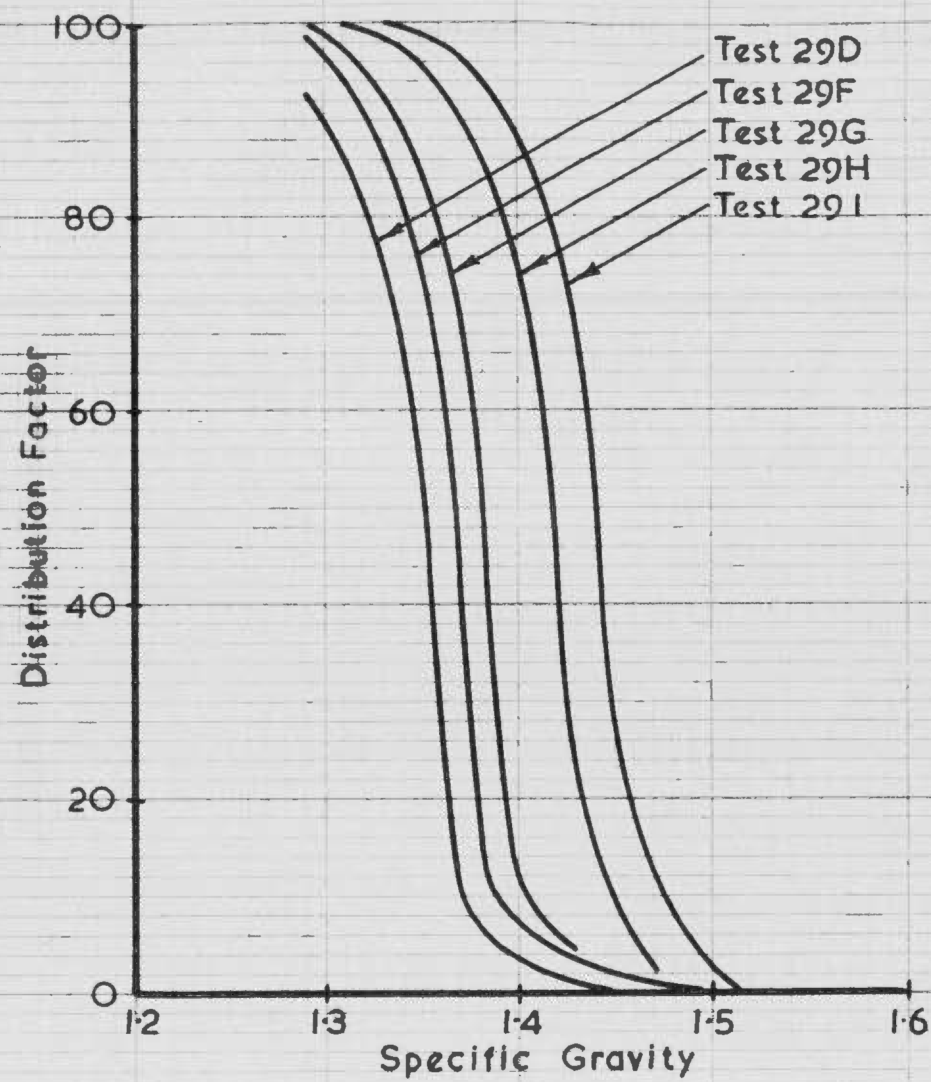

FIGURE 14 Tromp curves relating to Table 5

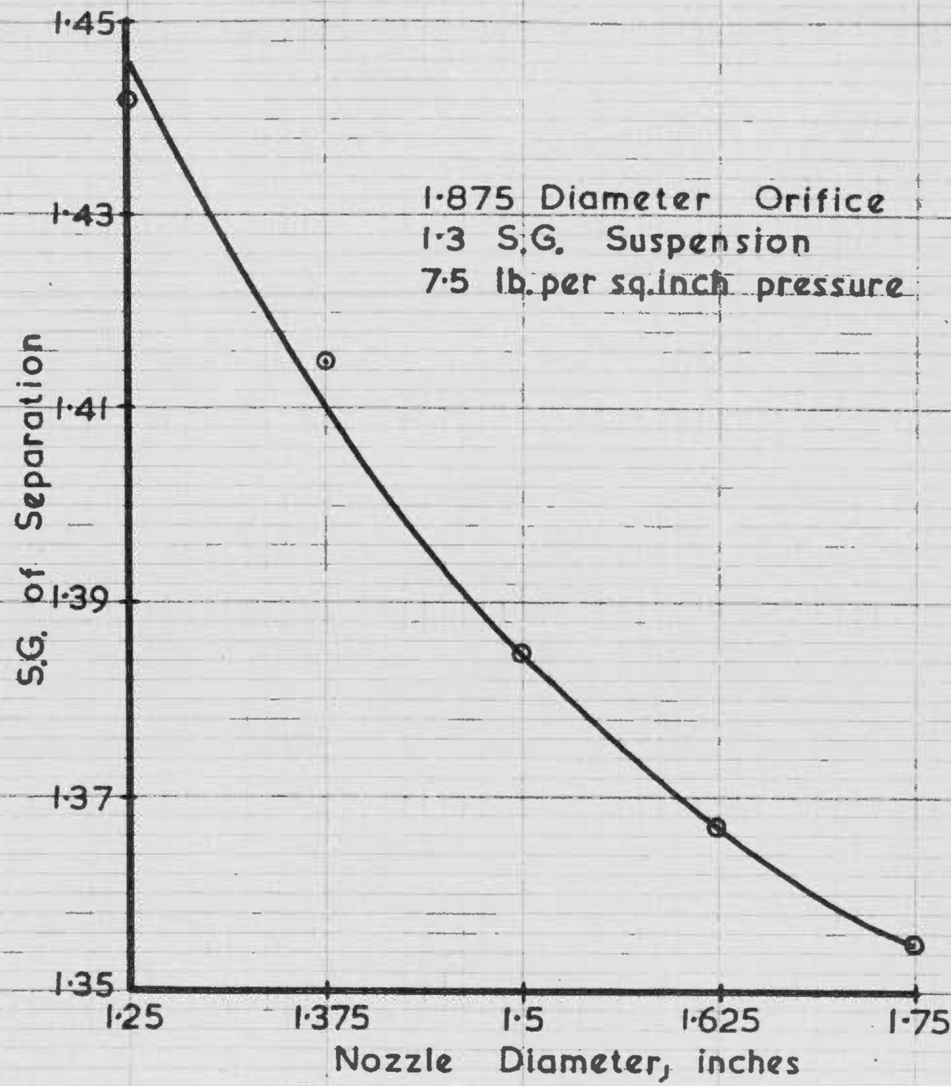

FIGURE 13 Relation between Nozzle diameter and S.G. of Separation

Test No.	29D	29F	29G	29H	29I
Nozzle diameter, inches	$1\frac{3}{4}$	$1\frac{5}{8}$	$1\frac{1}{2}$	$1\frac{3}{8}$	$1\frac{1}{4}$
Yield of product, %	22.3	32.6	40.2	56.6	66.1
Ash content of product, %	4.9	5.4	5.8	6.4	7.1
Ash content of feed, %	14.2	13.4	13.3	12.2	12.1
Specific gravity of separation.	1.355	1.367	1.385	1.415	1.442
Capacity, tons per hour	2.12	1.83	1.66	1.65	-

It will be seen from these tests that the diameter of the nozzle determines the specific gravity at which the separation is effected, all other variables being constant, and Fig.13 indicates that there is a definite relationship between nozzle diameter and specific gravity of separation under these conditions. It is important to note that the specific gravity of separation increases with decreasing nozzle diameter. Other tests have shown that a similar effect is obtained if the nozzle diameter is kept constant and the orifice diameter is varied. In this case, the specific gravity of separation increases with increasing orifice diameter. Thus, a separation at 1.355 specific gravity (Test 29D) could be changed to a separation at 1.442 by keeping the orifice diameter constant at $1\frac{7}{8}$ inches and reducing the nozzle diameter to $1\frac{1}{4}$ inches as in Test 29I. Alternatively, the nozzle diameter could be kept constant at $1\frac{3}{4}$ inches and the orifice diameter could be increased from $1\frac{7}{8}$ inches to the appropriate size. It will be clear, then, that within practical limits any orifice (or nozzle) whatsoever may be used for a given separation provided that the appropriate nozzle (or orifice) is also used. In other words, the relationship between the orifice diameter and nozzle diameter determines the specific gravity of separation and, in general, the greater the ratio between the orifice and nozzle diameters, the higher is the specific gravity of separation, other variables being constant. Since the specific gravity of suspension largely

determines...../

determines the relationship between the orifice and nozzle required for a specified separation, it will be appreciated that the number of combinations of orifice and nozzle which could be used is almost infinite. However, as will be shown later, the actual diameters of the orifice and nozzle (and their relationship) have an important bearing on the capacity, efficiency and size grading of feed which can be treated. Therefore, the final selection is largely governed by these factors.

Apart from the specific gravity of the suspension, the relationship between the orifice and nozzle diameters affords the most important means of controlling the separation and has the advantage that an alteration to either the orifice or nozzle diameter varies the cut almost instantaneously. In practice it will probably be most convenient to vary the nozzle as it is usually more accessable than the orifice. The range of variation of specific gravity of separation which may be obtained by nozzle control is limited, however, by the size grading and specific gravity distribution of the feed and the capacity required. The reason for this is discussed in detail in a later chapter.

The Tromp distribution factor curves relating to this series of tests are shown in Fig. 14. Comparison of these curves suggests that the efficiency of separation is not affected to any appreciable extent by variation of the nozzle diameter between the limits investigated. It must not be assumed, however, that such a wide variation of nozzle diameter would be permissible under all conditions. A nozzle (or orifice) of any specified diameter is only able to pass a certain maximum quantity of coal in unit time, feed pressure and other variables being constant. If the maximum permissible rate of discharge of solid is exceeded, the efficiency will be impaired. The smallest nozzle which can be used for any desired separation will thus depend on the capacity required and on the washability

characteristics../

characteristics of the feed (i.e. on the percentage discard).

In the series of tests under discussion, the percentage tailing as well as the capacity decreased progressively as the nozzle diameter was reduced and this resulted in a substantial decrease in the rate of discharge of refuse. Since the efficiency of separation appears to have remained constant, it may be concluded that the nozzle was not overloaded in any of these tests.

The progressive decrease of capacity which will be observed in Table 5, is not a feature of the cyclone, but is due to settling of coal which takes place in the feed tank during a series of experiments conducted on the same batch of feed (i.e. the pulp flowing through the cyclone contains more coal for the first test of a series than it does for the last test from the same batch of pulp). Since the pulp ratio of the slurry actually passing through the cyclone is variable, small changes in the capacity due to a change in the rate of flow of pulp are likely to be obscured. In order to obtain some idea of the influence of the nozzle diameter on the rate of flow of pulp, several tests were carried out using water as feed, the effluents from the orifice and nozzle being collected and weighed. The time taken to collect these quantities was also noted. The results of these tests are reported in Table 6.

TABLE 6.

Influence of Nozzle diameter on the flow of water through a Cyclone.

The following conditions were constant for all tests :-

(1) $9\frac{1}{2}$ inches diameter cyclone, 25 degrees apex angle, $1\frac{1}{2}$ inches diameter feed pipe.

(2) $1\frac{7}{8}$ inches diameter orifice tube projecting $1\frac{1}{4}$ inches beneath cover plate.

(3) Feed pressure, 10 lb. per sq. inch.

Nozzle diam. (inches)	Rate of flow of feed. (lb.per sec.)	Flow through Orifice percentage of total
$1\frac{3}{4}$	8.1	50.6
$1\frac{5}{8}$	7.3	60.3
$1\frac{1}{2}$	7.3	67.0
$1\frac{3}{8}$	7.2	74.9
$1\frac{1}{4}$	7.2	82.1

These tests indicate that the total rate of flow through the cyclone is not affected appreciably by variation of the nozzle diameter between the limits investigated.

It is interesting to note that the proportion of water flowing through the orifice increases as the nozzle diameter is decreased, the relationship being approximately linear. Comparing Tables 5 and 6, it will be seen that an increase in the proportion of water flowing through the orifice corresponds to an increase in the specific gravity of separation.

Referring again to Table 5, it was stated that all variables, with the exception of the nozzle diameter, had been kept constant. It was subsequently realised, however, that this was not strictly correct as one variable had been neglected, viz. the ratio between the nozzle diameter and the mean size of the tailing. Disregarding possible degradation of the feed, this ratio decreased when the smaller nozzles were tested. Nevertheless, it does not appear that the results obtained were influenced by this oversight, as later tests proved that the separation of minus $\frac{1}{4}$ inch plus 1 mm feed is only affected by this factor when the nozzle diameter is less than about $1\frac{1}{4}$ inches.

As stated previously, any orifice (or nozzle) may be used to effect a specified separation provided that the appropriate nozzle (or orifice) is also used. This is illustrated in Table 7, in which various suitable combinations of orifice and nozzle are shown for a separation at approximately 1.35

specific .../

specific gravity. The specific gravity of the suspension was kept constant at 1.28 for all these tests and various orifices were fitted. The appropriate nozzle for each orifice was then determined by experiment. An approximate method[14] was used to determine the specific gravity of separation in each case.

TABLE 7.

Some combinations of the Orifice and Nozzle which could be used to effect a separation at approximately 1.35 S.G.

The following conditions were constant for all tests :-

(1) $9\frac{1}{2}$ inches diameter cyclone, 25 degrees apex angle, $1\frac{1}{2}$ inches diameter feed pipe.
(2) Specific gravity of suspension, 1.28.
(3) Feed coal:- Landau No. 3, $-\frac{1}{4}$" +1 mm.
(4) Pulp ratio, 6 to 1
(5) Feed pressure, $7\frac{1}{2}$ lb. per sq. inch.
(6) Disc-type orifices used in all tests.

Test No.	22A	18C	22D	22G
Orifice diameter, inches.	$2\frac{1}{8}$	$1\frac{7}{8}$	$1\frac{1}{2}$	1
Nozzle diameter, inches.	$1\frac{3}{4}$	$1\frac{5}{8}$	$1\frac{1}{4}$	1
Yield of product, %	29.2	30.8	34.3	34.3
Ash of product, %	5.8	5.8	6.0	5.8
Ash of feed, %	14.4	14.4	14.8	15.3
S.G. of separation.	1.35	1.35	1.36	1.36
Washing efficiency, %	83	88	86	86
Capacity, tons of feed per hour.	2.4	2.1	2.0	1.4
Diameter of largest permissible particle, inches.	0.35	0.325	0.25	0.2
Total flow of water, lb.per sec.	8.1	7.5	5.9	4.1
Feed pipe velocity, ft.per sec.	10.6	9.8	7.7	5.4
Centrifugal acceleration at the radius of the orifice, ft.per sec.^{2}	4800	5200	5100	5600

These tests are not of considerable value as the Tromp distribution factor curves were not determined, hence no reliable conclusion can be drawn as to the influence of the various orifice and nozzle combinations on efficiency. However, it is important to note that the capacity shows a tendency to

decrease..../

decrease when the smaller orifices are used. This was later confirmed by tests using water as feed. The results of these tests are also shown in Table 7. It will be noted that the rate of flow decreases with decreasing orifice diameter.

Later tests indicated that the largest particles present in the feed should not exceed about one fifth of the nozzle diameter. The largest permissible particles were, therefore, estimated on this basis as shown in Table 7.

Since the Witbank duff coals which were to be washed finally in the $9\frac{1}{2}$ inches diameter cyclone contain particles of half to quarter inch, it was clear that the smaller sizes of orifice and nozzle would not be suitable for these separations. As small orifice and nozzle combinations also decrease the maximum capacity attainable, it was decided to limit further work to large orifices, with the object of determining the maximum permissible diameter of this aperture. The tests conducted in this connection are described in a later chapter.

Although the present investigation is not concerned with the separation of material which consists entirely of very small particles, an analysis of the probable influence of the orifice and nozzle combination on the efficiency of separation of such feed may be of interest.

Assuming that the relationship between the tangential velocity and the radius of rotation is given by the expression, $VR^{\frac{1}{2}}$ = constant,[(7)] it is clear that the centrifugal force acting on a particle becomes greater as the radius is decreased. Since the centre of the vortex in the cyclone is occupied by air, it follows that the radius of rotation of a particle can not be zero but is limited by the diameter of the air column. As a first approximation, the minimum radius of rotation may be taken as being equal to the radius of the orifice (this is a reasonable approximation as observations have shown that the rotating ring of fluid issuing from the orifice is comparatively thin).

The..../

a constant specific gravity of separation. In other words, two variables must be altered in order to investigate the influence of the specific gravity of suspension on cyclone performance for a fixed separation. Since the investigation was primarily concerned with separations at predetermined specific gravities (1.35 to 1.4 S.G. for Witbank duff) this procedure had to be adopted.

Four suspension specific gravities were selected for the first series of tests which were conducted to investigate the influence of this variable on cyclone performance. The orifice diameter was kept constant and the nozzle diameter was varied in order to obtain a separation in the region of 1.35 specific gravity with each suspension tested. The results of tests in which the separation was affected at approximately the required point, are reported in Table 8, and the corresponding Tromp curves are shown in Fig. 15.

TABLE 8.

Influence of the Specific Gravity of the Barytes suspension on Cyclone performance.

The following conditions were constant for all these tests:-

(1) $9\frac{1}{2}$ inches diameter cyclone, 25 degrees apex angle, $1\frac{1}{2}$ inches diameter feed pipe.

(2) $1\frac{7}{8}$ inches diameter orifice tube projecting $1\frac{1}{4}$ inches beneath cover plate.

(3) Feed coal :- Landau No. 3 duff, $-\frac{1}{4}$" +1 mm.

(4) Pulp ratio, 6 to 1.

(5) Feed pressure, $7\frac{1}{2}$ lb. per sq. inch.

Test No..../

Test No.	29A	28A	28B	20D	29D	29F
Specific gravity of suspension	1.20	1.24	1.24	1.28	1.30	1.30
Nozzle diameter, inches.	7/8	1 1/8	1 1/4	1 5/8	1 3/4	1 5/8
Yield of product, %	39.5	31.0	25.9	34.3	22.3	32.6
Ash of product, %	7.4	6.1	5.3	5.6	4.9	5.4
Ash of feed, %	13.4	15.3	14.3	14.1	14.2	13.4
Specific Gravity of separation.	1.363	1.375	1.36	1.358	1.355	1.367
Washing efficiency, %	84	87	-	96	89	96
Capacity, tons per hour	3.25	2.72	2.44	2.13	2.12	1.83
Refuse, tons per hour, per inch nozzle diameter	2.25	1.67	1.45	0.86	0.94	0.76

Although the results are somewhat variable, one gains the general impression from Table 8 and the Tromp distribution factor curves shown in Fig. 15, that there is a tendency for the efficiency of separation to decrease as the suspension specific gravity is reduced. However, it will be noted that the loss of efficiency (if any) is less noticeable over the specific gravity range 1.3 to 1.24 than in the range 1.24 to 1.20. This fact suggested that the poor efficiency obtained in test 29A was not entirely due to the relatively low specific gravity of suspension used. Furthermore, it was observed that in both the tests (29A and 28A) in which the efficiency was impaired to a noticeable degree, comparatively small nozzles had been fitted. Now, since the feed coal was relatively large compared with the diameter of these nozzles, it was concluded that the variation in the relationship between the diameter of the nozzle and the size of the coal passing through it, was at least partly responsible for the loss of efficiency noted. (The significance of the quantities passing through the nozzle was not appreciated until later).

Since it was impractical to maintain a constant relationship between the diameter of the nozzle and the size of the material passing by altering the grading of the feed for each nozzle, and since all sizes of coal are not cut with

equal..../

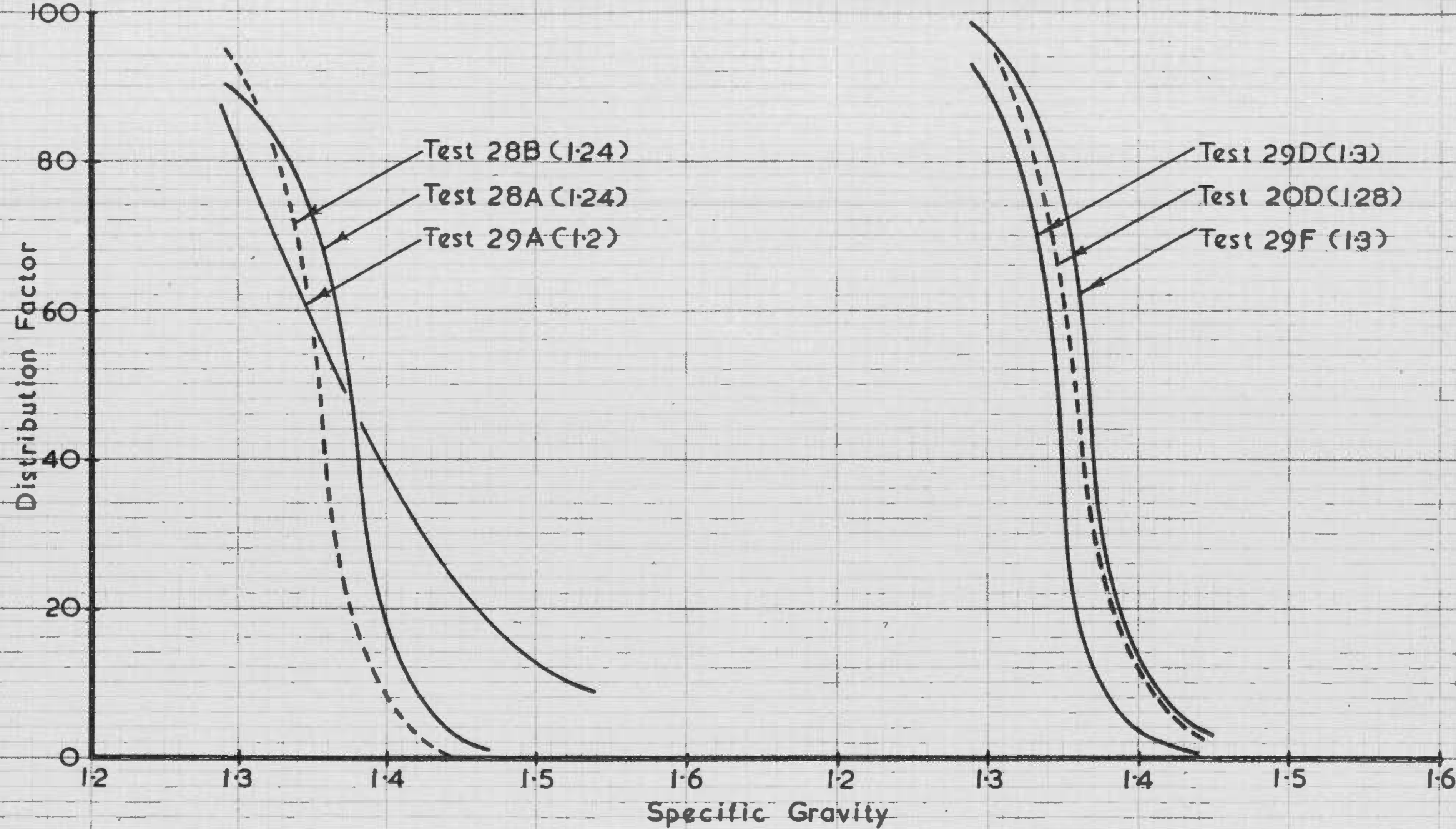

FIGURE 15 Tromp curves relating to Table 8

equal efficiency, it was decided to repeat the tests referred to above using fine feed coal. Even the smallest nozzle used would then be large compared with size of the coal particles, and the influence of the relationship between the nozzle diameter and the size of the material would be minimised, if not entirely eliminated. The results of these tests are reported in Table 9 and the relevant Tromp distribution factor curves are shown in Fig.16.

TABLE 9.

The influence of the Specific Gravity of the Bartyes suspension on Cyclone performance when treating fine feed coal.

The following conditions were constant for all tests:-

(1) $9\frac{1}{2}$ inches diameter cyclone, 25 degrees apex angle, $1\frac{1}{2}$ inches diameter feed pipe.

(2) $1\frac{7}{8}$ inches diameter orifice tube projecting $1\frac{1}{4}$ inches beneath cover plate.

(3) Feed coal :- Landau No. 3 duff, - 1 mm +60 mesh.

(4) Pulp ratio, 6 to 1.

(5) Pressure, 10 lb. per sq. inch.

Test No.	76E	76A	76C	48I	47B	47F	26C	31C
S.G.of suspension	1.1	1.15	1.15	1.15	1.2	1.2	1.28	1.3
Nozzle dia. inches.	$\frac{5}{8}$	$\frac{7}{8}$	1	$\frac{3}{4}$	1	$1\frac{1}{8}$	$1\frac{5}{8}$	$1\frac{5}{8}$
Product yield, %	53.0	53.0	25.8	54.8	50.1	44.4	41.6	39.4
Product Ash, %	8.8	7.6	4.8	8.0	7.0	5.8	5.9	5.6
Feed Ash, %	14.9	14.2	13.8	13.7	14.4	12.6	13.6	14.6
S.G.of separation.	1.42	1.42	1.32	1.43	1.41	1.38	1.37	1.368
Washing efficiency, %	62.0	70.0	85.0	71.0	82.2	86.6	86	87
Capacity, tons per hour	3.3	2.4	2.0	3.5	3.0	2.3	2.2	2.0
Refuse, tons per hour	1.55	1.13	1.49	1.58	1.50	1.28	1.29	1.22
Tons of refuse per hour per inch nozzle diameter.	2.48	1.29	1.49	2.11	1.50	1.14	0.79	0.75

Although the results of the tests shown in Table 9 and Fig.16 again convey the impression that there is a tendency for the efficiency of separation to decrease with decreasing suspension specific gravity, particularly for suspensions below 1.2 S.G.,

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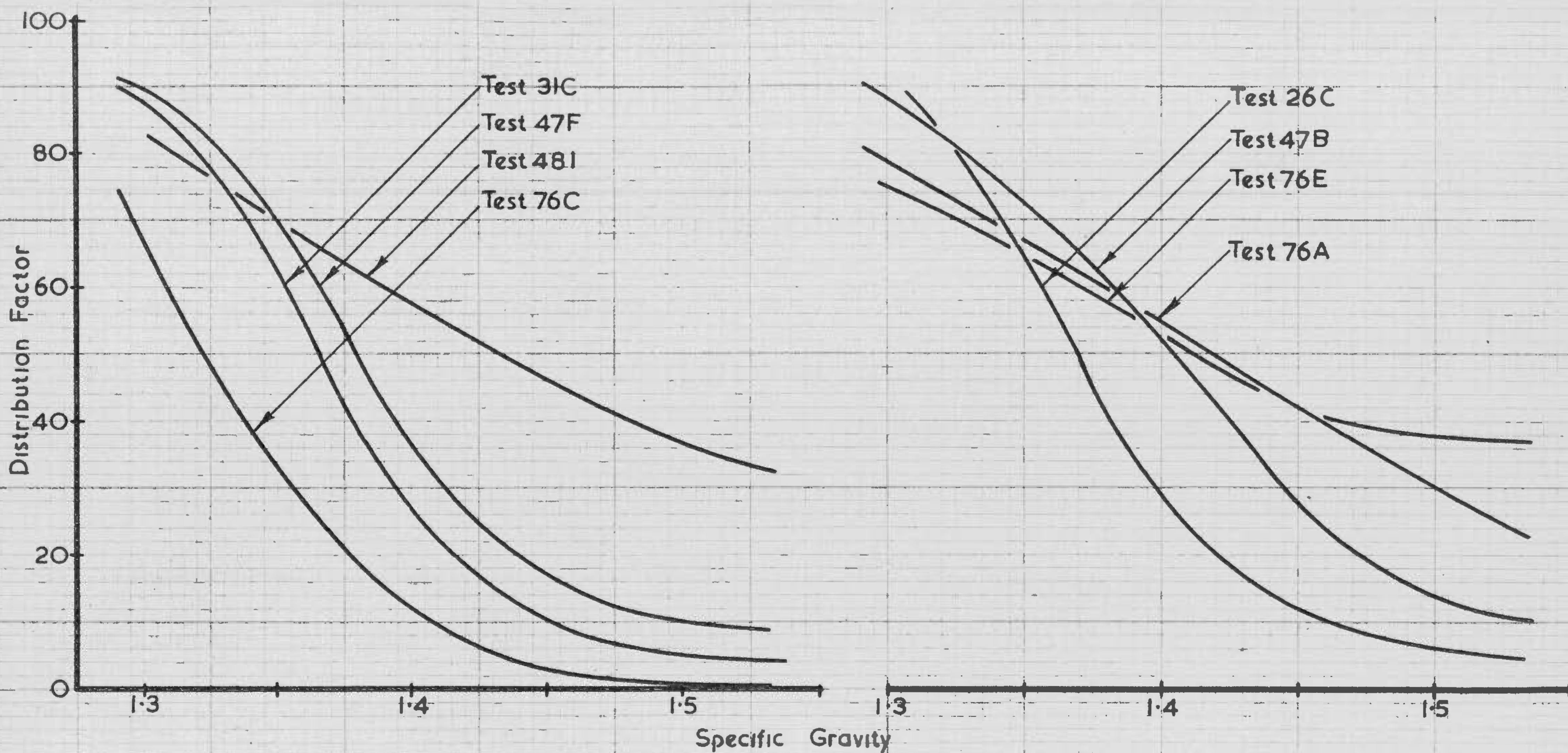

FIGURE 16 Tromp curves relating to Table 9

the fact that this tendency is not consistent indicated that some other factor was probably also involved. For example, comparatively poor separations were obtained in Tests 76A and 48 I using a 1.15 specific gravity suspension, while the same suspension gave a reasonably sharp "cut" in Test 76C. Since the influence of the relationship between the diameter of the nozzle and the size of the feed had largely been eliminated, it was clear that some other explanation of these phenomena had to be sought.

Upon further study, it was observed that the capacity (or throughput) of the cyclone had varied and that loss of efficiency was generally associated with an increase in the capacity and a decrease in the nozzle diameter. Thus, comparing tests 47B and 47F, it will be noted that the Tromp distribution factor curve for the former is distinctly inferior to that for the latter. Now, the suspension specific gravity was constant at 1.2 for both tests but the nozzle diameter was smaller in test 47B while the throughput was greater. Similarly, in tests 76A and 76C the suspension specific gravity was constant at 1.15 and the nozzle diameters were 7/8 inch and one inch respectively. In this case, the difference in the efficiencies of separation is particularly marked, higher efficiency being obtained with the larger nozzle and slightly lower throughput.

Since the efficiency of separation is apparently influenced by both the nozzle diameter and the throughput, it was concluded that the quantity of material passing through the nozzle in unit time was an important factor. Assuming constant throughput of feed and specific gravity of separation, it follows that a constant quantity of refuse must pass through the nozzle in unit time. Since the nozzle diameter has to be reduced as the suspension specific gravity is decreased, (in order to maintain a constant specific gravity of separation) the area through which the refuse passes becomes progressively smaller. It seems reasonable to suppose, therefore, that at some stage insufficient area becomes available for the discharge of refuse and disturbed con-

ditions of..../

of flow will result, with consequent loss of efficiency. In other words, it is suggested that the cyclone would become overloaded if the nozzle diameter is decreased below a certain minimum size, the pulp ratio, rate of flow and specific gravity of separation remaining constant.

The rates of discharge of refuse were consequently calculated for the tests under discussion and are shown in Table 9. It is now necessary to relate these values to the diameter of the nozzle in order to verify the theory outlined above. The total cross sectional area of the nozzle is not a suitable basis, as the pulp does not fill the nozzle but merely occupies a rotating annular ring, an air space being present in the centre of the nozzle. Unfortunately, the relative proportions of these spaces under various operating conditions is not known and it is beyond the scope of the present investigation to determine these data. It may be mentioned, however, that a few preliminary experiments were conducted on a small celluloid cyclone in order to obtain a clearer conception of its operation. It was observed, during these tests, that the annular ring of fluid flowing through the orifice and nozzle was very thin compared with the respective diameters of these apertures. One is therefore inclined to assume, as a first approximation, that the annular area of flow is proportional to the circumference of the nozzle (or orifice) and, hence, to its diameter. The rate of flow of refuse through a nozzle, may, if this is true, be represented approximately by the rate of discharge of refuse expressed in tons per hour per inch of nozzle diameter. These values were calculated for the various tests and are also shown in Table 9. This figure should naturally be based on the total solids passing through the nozzle, but since the distribution of the heavy medium between the orifice and nozzle was not known, the rate of flow of the medium was temporarily neglected.

Comparing..../

Comparing the rates of discharge of refuse per inch of nozzle diameter shown in Table 9, it will be seen that, with the exception of test 76A, a reasonably sharp separation was obtained in all tests in which this value was less than about 1.49. A similar effect will be observed for the tests reported in Table 8. It appears, then, that under these experimental conditions the maximum permissible rate of discharge of refuse is in the order of 1.49 tons per hour per inch nozzle diameter when the feed pressure is about $7\frac{1}{2}$ to 10 lb. per sq. inch. (This value may be increased, by increasing the pressure, as will be shown later). It will be appreciated that the figure deduced above should only be regarded as a rough guide at this stage. The factors which govern the maximum permissible rate of discharge of refuse will be discussed in greater detail in a later chapter.

Assuming for the present that the maximum quantity of refuse which can be discharged from the cyclone in unit time is proportional to the nozzle diameter, it follows that the diameter of this aperture would have an important bearing on capacity and on the choice of suspension specific gravity. Clearly, its relative importance would depend on the particular washing problem, and two alternatives may be discussed by way of illustration.

Consider, for example, the case where Witbank duff is to be cut at about 1.35 specific gravity. The discard, which has to be eliminated through the nozzle, is about 70% of the feed. A comparatively large nozzle will consequently be required in order to ensure satisfactory capacity and efficiency. If it is desired to use a low specific gravity of suspension to effect this separation, it follows that a very large orifice would be required in order to provide the necessary relationship between orifice diameter and nozzle diameter. This possibility was investigated, as will be described later, and it was found that the efficiency was impaired when the orifice diameter exceeded about $1\frac{7}{8}$ inches, hence there is not much scope in this direction. It follows, therefore, that in order to obtain high

capacity.../

capacity, it is necessary to use a relatively high suspension specific gravity for this separation to ensure that the nozzle may be as large as possible. It will be shown later that this conclusion is independent of the actual diameter of orifice used.

A separation at a high specific gravity, on the other hand, represents the opposite situation. Consider a hypothetical case in which the yield of product is, say, 90%. Here, a small value for the ratio between the orifice and nozzle diameters would not be an advantage, because the nozzle would be more than adequate for the quantity of refuse to be passed and the orifice would determine the optimum capacity at any given pressure. Clearly, a large orifice would be required, while the diameter of the nozzle would be of little consequence provided that it was of sufficient diameter to pass the size of coal to be treated without disturbance. Relatively lower suspension specific gravities could, therefore, be used to advantage. -

Referring again to the tests shown in Table 9, and comparing the Tromp distribution factor curves for tests 31C, 26C, 47F and 76C, it will be seen that, although the suspension specific gravity was varied between the limits of 1.30 and 1.15 in these tests, the efficiencies of separation do not differ appreciably. Since the rate of discharge of refuse was comparatively low in all these tests and did not, therefore, influence the operation of the cyclone, it appears that variation of the suspension specific gravity between these limits does not affect the "potential" separation materially. Thus, a relatively low specific gravity of suspension should yield a satisfactory separation provided that the percentage of coal in the pulp is reduced to a suitable value (i.e. by increasing the pulp ratio).

In order to test this deduction, tests 76A and 48 I were repeated using a pulp ratio of 12 to 1. In both cases, an extremely low yield of product was obtained when using the

higher.../

higher pulp ratio, and it was consequently not possible to assess the washer efficiency. However, these tests did show that the specific gravity of separation is influenced by the pulp ratio when the cyclone is operated under overloaded conditions i.e. discharge of refuse at a rate in excess of the maximum permissible for a given nozzle causes an additional "nozzling effect" and raises the specific gravity of separation.

The influence of the specific gravity of suspension on the specific gravity of separation, when all other variables are kept constant, would be of importance if this factor is used for washery control. Although no tests were specifically carried out to determine this relationship, various tests in which the specific gravity of suspension happened to be a common variable have been selected at random from the records and are shown in Table 10.

TABLE 10.

The influence of the specific gravity of suspension on the specific gravity of separation when the other variables are constant.

The following conditions were constant for all tests :-

(1) 9½ inches diameter cyclone, 25 degrees apex angle, 1½ inches diameter feed pipe.

(2) 1⅞ inches diameter orifice tube projecting 1¼ inches beneath cover plate.

(3) Pulp ratio, 6 to 1.

Test No.	Size grading of feed	Feed pressure lb/sq.in.	Nozzle dia. inches.	S.G. of suspension.	S.G. of separation	Thickening.
97A	-⅜" +100 #	10	1½	1.5	1.585	0.085
33A	-3/16" +40 #	7½	1½	1.3	1.378	0.078
95D	-¼" +1 mm	10	1½	1.3	1.365	0.065
38A	-½" +40 #	10	1¾	1.52	1.593	0.073
29D	-¼" + 1 mm	7½	1¾	1.3	1.355	0.055
29 I	-¼" + 1 mm	7½	1¼	1.3	1.442	0.142
28B	-¼" + 1 mm	7½	1¼	1.24	1.355	0.115

Before..../

Before discussing the above table it is necessary to point out that all sizes of coal are not washed at the same specific gravity under otherwise constant operating conditions. It has been found that the specific gravity of separation increases as the particle size is decreased below about 0.08 inch in the case of the barytes suspension used. The percentage fines in the feed, therefore, influences the overall separation to a certain extent. Variation in the feed pressure also influences the specific gravity of separation, but this effect is not very great for changes in feed pressure between $7\frac{1}{2}$ and 10 lb. per sq. inch.

Taking these factors into consideration, it will be seen from Table 10 that the "thickening effect" is reasonably constant as the suspension specific gravity is varied, other variables being constant. From this it is concluded that there is a linear relationship between specific gravity of suspension and specific gravity of separation, between the limits investigated.

THE INFLUENCE OF PULP RATIO ON CYCLONE PERFORMANCE.

It has already been shown that the pulp ratio affects the specific gravity of separation when the cyclone is operated under overloaded conditions, and that the maximum capacity of the cyclone is determined largely by the diameter of the nozzle in the case of separations in which the percentage refuse exceeds the percentage product. It was also explained that the nozzle should be as large as possible and, since the orifice diameter can not exceed $1\frac{7}{8}$ inches, a relatively high suspension specific gravity would be required to effect such a separation.

The influence of pulp ratio on cyclone performance, when the nozzle is not overloaded, is illustrated by the tests reported in Table 11. It will be observed that the specific gravity of suspension was relatively high and that the nozzle was consequently reasonably large.

The Tromp distribution factor curves obtained for these tests are shown in Fig 17.

FIGURE 11.../

TABLE 11.

The influence of Pulp Ratio on Cyclone performance

The following conditions were constant for these tests:-

(1) $9\frac{1}{2}$ inches diameter cyclone, 25 degrees apex angle, $1\frac{1}{2}$ inches diameter feed pipe.

(2) $1\frac{7}{8}$ inches diameter orifice tube projecting $1\frac{1}{4}$ inches beneath cover plate.

(3) $1\frac{1}{2}$ inches diameter nozzle.

(4) Suspension specific gravity 1.30.

(5) Feed coal :- Waterberg; seam samples as indicated; crushed $-\frac{3}{16}$" + 36 mesh.

(6) Feed pressure, $7\frac{1}{2}$ lb.per sq.inch.

Test No	34A	33A	32A	35A
Pulp ratio	6	5	4	3
Seam	6D	6E	6F	6A
Yield of product, %	38.8	33.0	39.0	29.4
Ash of product, %	9.5	8.8	9.5	9.7
Ash of feed, %	23.0	26.5	22.1	34.6
Specific gravity of separation	1.381	1.378	1.373	1.378
Washing efficiency, %	92.0	95.0	91.0	95.0
Capacity, tons per hour	2.48	3.17	3.94	4.1
Refuse, tons per hour per inch nozzle diameter	1.01	1.42	1.6	1.93

It will be clear that the capacity of the cyclone is determined by two factors viz: (a) the rate of flow of pulp and (b) the proportion of coal in the pulp. Now, the pulp ratio merely defines the proportion of coal in the pulp and, therefore, determines the capacity when all other variables are constant. Thus, as will be seen in Table 11, the capacity increases as the pulp ratio is decreased. It follows, then, that the pulp ratio should be as low as possible in order to obtain maximum capacity under any given operating conditions.

If the Tromp distribution factor curves for these tests are compared (Fig.17), it will be noted that variation of pulp ratio, between the limits specified, had negligible influence on the efficiency of separation, and consequently it may be concluded that the maximum capacity at $7\frac{1}{2}$ lb. per sq. inch was not reached.

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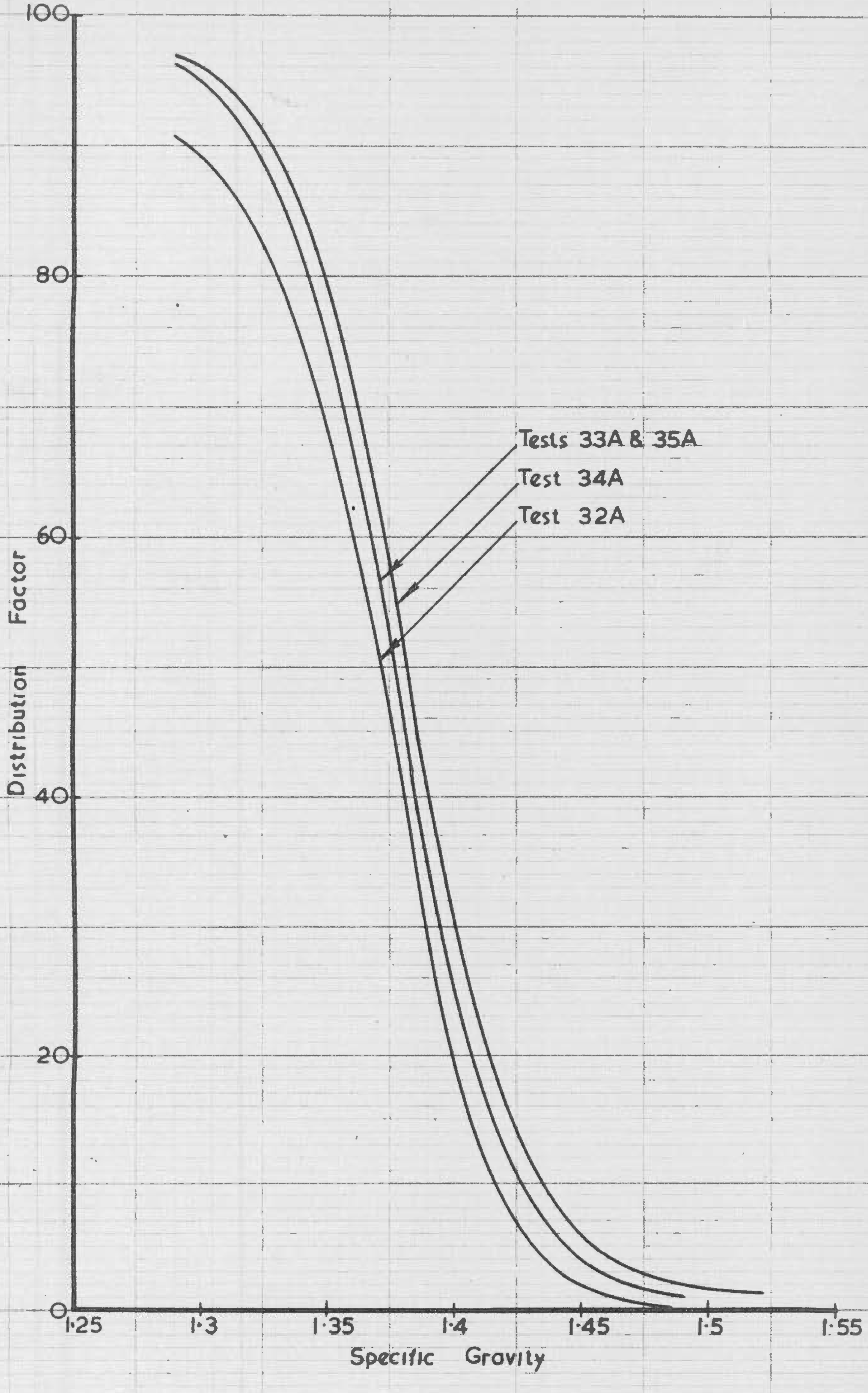

FIGURE 17 Tromp curves relating to Table 11

Since the pulp tends to lose fluidity when the pulp ratio is decreased below about 3 to 1, it was not considered prudent to conduct tests with lower pulp ratios with the object of determining the maximum permissible nozzle loading (and hence the l west permissible pulp ratio for the specified operating conditions).

It will also be observed that the specific gravity of separation was substantially constant despite the comparatively large variations in the pulp ratio and the specific gravity composition of the feed coal (see appendix 3). From this it may be concluded that the specific gravity distribution does not influence the performance of the cyclone (except in cases where the nozzle or orifice would become overloaded by such variation).

The fact that the pulp ratio does not influence the efficiency or specific gravity of separation suggests that a cyclone plant would be extremely flexible in practice from the point of view of capacity, provided that the washer is not overloaded. In other words, fluctuations in the rate of feed would not necessitate frequent adjustments to the operating conditions i.e. the pulp ratio may be allowed to vary (within reasonable limits) without adverse effects.

Although samples from different seams of the Waterberg coalfield were used for these tests, they were all crushed in the same mill and it was found that the size distribution of the cyclone feed did not vary appreciably. These conclusions should not, therefore, be affected to any great extent, by this factor.

Table 11 indicates that the rate of discharge of refuse may be varied between 1.01 and 1.93 tons per hour per inch nozzle diameter without influencing the efficiency. This does not agree with the conclusions drawn from Table 9 viz: that the discharge rate should not exceed 1.49 tons of refuse per hour per inch nozzle diameter. This discrepancy may be accounted for in two ways :-

(a) that the maximum permissible rate of discharge depends on the size of the particles (-1 mm +60 mesh feed was used for the tests reported in Table 9 and 3/16" +40 mesh.../

mesh for the tests in Table 11).

(b) that the maximum permissible nozzle loading expressed in tons per hour per inch nozzle diameter is not constant for all nozzle diameters (i.e. higher loadings are permissible for large nozzles than for small nozzles).

Of these two explanations the latter is favoured. In support of this statement the following evidence may be offered :-

(1) The value, 1.49, was deduced from the results of tests in which the nozzle diameter was about 1 inch, while the value 1.93 was obtained during tests where a $1\frac{1}{2}$ inches diameter nozzle was used.

(2) In test 76A (Table 9) the nozzle loading was only 1.29 tons per hour per inch nozzle diameter, yet the efficiency was poor. In this case the nozzle diameter was $7/8$ inch, and a lower "specific loading" would explain the facts.

(3) In test 28A (Table 8) the feed was relatively coarse, yet slight loss of efficiency was evident although the nozzle loading was only 1.67 (as compared with 1.93 obtained later). The diameter of the nozzle was $1\frac{1}{8}$" in this test and the phenomena could be explained by assuming that the "specific loading" decreases with nozzle diameter.

In order to determine whether this conclusion is reasonable, tests 76A, 76C, 28A and 35A (Tables 8, 9 and 11) were repeated using the appropriate suspension specific gravities but no coal was added to the pulp. The specific gravities of the effluents from the orifice and nozzle were measured and the rate of flow of each stream was determined. From these data, the volume of the pulp issuing from the nozzle and orifice in unit time, the weight of barytes passing through the nozzle in unit time, and the percentage of the feed pulp flowing through the nozzle were calculated. Relevant values are shown in Table 12.

TABLE 12..../

TABLE 12.

The Influence of Nozzle Diameter on the Distribution of Pulp under Various Operating Conditions.

The following conditions were constant for these tests :-

(1) $9\frac{1}{2}$ inches diameter cyclone, 25 degrees apex angle, $1\frac{1}{2}$ inches diameter feed pipe.

(2) $1\frac{7}{8}$ inches diameter orifice tube projecting $1\frac{1}{4}$ inches beneath cover plate.

(3) Feed pressure 10 lb. per sq. inch.

Nozzle diam. ins.	Feed suspension S.G.	Flow of feed through cyclone		Flow of suspension through nozzle				Flow of tailing through nozzle			
		lb/sec	cu. ft/sec.	% by volume	S.G.	flow of pulp cu.ft/sec.	flow of Barytes lb/sec.	Test No.	lb/sec.	cu. ft/sec	Nozzle loading tn/hr/in
$\frac{7}{8}$	1.15	8.1	0.113	5.3	1.81	0.006	0.38	76A	0.63	0.007	1.29
1	1.15	8.2	0.114	6.7	1.72	0.0076	0.43	76C	0.83	0.009	1.49
$1\frac{1}{8}$	1.24	8.8	0.115	10.4	1.67	0.0115	0.64	28A	1.04	0.011	1.67
$1\frac{1}{2}$	1.3	9.2	0.113	31.5	1.48	0.0356	1.41	35A	1.61	0.017	1.93

Tests 28A and 35A were originally carried out at $7\frac{1}{2}$ lb. per sq. inch pressure, while the tests using suspension only were all carried out at 10 lb. per sq. inch. Since the difference in the rates of flow at these two pressures is only of the order of 10 per cent, this factor is neglected for the purpose of the present analysis.

The results of the tests in Table 12 indicate that the total volume of pulp passing through the cyclone in unit time is not influenced by the diameter of the nozzle or the specific gravity of the suspension, under the operating conditions specified. However, the distribution of the pulp between the orifice and nozzle is affected appreciably by the diameter of the nozzle. Thus, the percentage of the feed pulp which passes through the nozzle decreases rapidly as the nozzle diameter is decreased below $1\frac{1}{2}$ inches. It should also be noted that the specific gravity of the pulp issuing from the nozzle increases as its

diameter.../

diameter is decreased, nothwithstanding the fact that the specific gravity of the feed suspension was decreased as the nozzle diameter was reduced. In other words, the rate of flow of pulp through the nozzle is reduced substantially as its diameter is decreased and, in addition, the concentration of heavy medium in this effluent increases materially.

Now, other tests have shown that the maximum permissible nozzle loading may be increased by raising the pressure (see Tables 17 and 18) i.e. by increasing the rate of flow through the nozzle. From this it may be concluded that the maximum nozzle loading is a function of the rate of flow. As will be seen from Table 12, the volume issuing from the nozzle in unit time decreases very rapidly as its diameter is reduced. Thus, the rate of flow through the one inch diameter nozzle is approximately one fifth of the corresponding value for the $1\frac{1}{2}$ inches diameter nozzle, which is a much greater decrease than would be the case if the flow were directly proportional to the diameter of the nozzle as was suggested before. In view of the fact that the concentration of barytes in the pulp increases with decreasing nozzle diameter, an even lower rate of discharge of refuse will probably be permissible than would be suggested by the rate of flow alone (since the percentage of total solids in the pulp is probably the governing factor).

It appears, then, that the "maximum permissible nozzle loading" (expressed in tons per hour per inch nozzle diameter) is not applicable to any nozzle diameter other than the one for which it was determined. As will be shown later, the relationship between the orifice and nozzle diameter also influences the maximum permissible nozzle loading for a given nozzle diameter.

It is appreciated that the exact relationship between the permissible rate of discharge of refuse and the nozzle diameter has not yet been determined, but it is clear that such a relationship exists and has an important bearing on cyclone capacity. Detailed work in this connection is beyond the scope

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of the investigation as a whole, and has consequently not been attempted. For the present, it is convenient to determine the nozzle loading in the manner suggested and to use this value as a practical guide when comparing the performance of the washer under various operating conditions. However, its limitations must be borne in mind.

It may be of interest, before concluding the discussion on this subject, to compare the rate of flow of suspension through the nozzle with the rate of discharge of refuse in actual washing tests. The rates of discharge of refuse in tests 76A, 76C, 28A and 35A are accordingly shown in Table 12. The volume of refuse passing through the nozzle per second was calculated assuming the specific gravity of the refuse to be 1.5.

It will be noted that the rate of discharge of refuse in tests 76A, 76C and 28A was very high compared with the rate of discharge of barytes in the case of the tests in which no coal was present. Since barytes was a component of the pulp containing these tailings, it may be concluded that the solid content (or specific gravity) of the discharge was very high. However, comparison of the relevant volumes is more instructive. Thus, it will be seen that the volume of the tailing discharged in unit time during tests 76A, 76C and 28A is equal to if not greater than the volume of barytes suspension discharged under the same operating conditions. Since barytes and water were also present in the pulps containing the refuse, it follows that much greater volumes were discharged in unit time than would be the case if no coal were present. This fact suggests that the nozzle was overloaded in these three tests. The Tromp distribution factor curves for tests 76A and 28A substantiate this conclusion. Since the volume of pulp passing in unit time was greater than the volume of suspension flowing through the nozzle under normal conditions, the effective area of the nozzle was probably decreased by the passage of refuse and an additional nozzling effect could be expected. As stated before, this was

actually..../

actually found to be the case when test 76A was repeated using a 12 to 1 pulp ratio. In test 35A, on the other hand, the volume of refuse discharged in unit time is only about half the normal value and consequently overloading of the nozzle is less probable. This is in agreement with the results actually obtained in this test i.e. neither the efficiency nor the specific gravity of the separation appear to have been affected.

THE INFLUENCE OF LARGE DIAMETER ORIFICES ON THE EFFICIENCY OF SEPARATION.

An analysis of the probable influence of the specific gravity of suspension on washing cost, indicated that the lowest possible suspension specific gravity should be used to effect any desired separation, provided that the efficiency is not impaired. Now, it has been shown that the ratio between the orifice diameter and the nozzle diameter must be increased as the suspension specific gravity is decreased in order to maintain a constant separation. Since the diameter of the nozzle (i.e. the smallest aperture) determines the size of feed which can be treated and the rate of discharge of refuse, (hence the maximum capacity) the largest possible nozzle should be used in the case of the separation of Witbank duff at 1.35 to 1.4 S.G.. It follows, then, that very large orifices would be required when low suspension specific gravities are used. In order to determine the influence of large diameter orifices on cyclone efficiency, the series of tests reported in Table 13 was carried out.

As the opening at the apex of the existing cyclone was only $1\frac{3}{4}$ inches, it was necessary to select a comparatively low suspension specific gravity (i.e. 1.2) in order to obtain a large value of the ratio between orifice diameter and nozzle diameter for a separation in the region of 1.35 to 1.4 specific gravity. In this way, fairly large orifices could be tested without reconstructing the cyclone in order to increase the opening at the apex and, in any event, this would have been an additional variable as the influence of nozzles larger than $1\frac{3}{4}$ inches was

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not known. In carrying out the tests, the required orifice was fitted in the cyclone and the nozzle diameter was varied in order to obtain a suitable specific gravity of separation, all other variables being kept constant. The specific gravities of separation actually obtained for the various orifices occupy a wider range than would have been desirable, but it is not considered that the final conclusions will be affected appreciably.

TABLE 13.

The influence of large diameter orifices on the efficiency.

The following conditions were constant for all tests :-

(1) $9\frac{1}{2}$ inches diameter cyclone, 25 degrees apex angle, $1\frac{1}{2}$ inches diameter feed pipe.

(2) Suspension specific gravity, 1.2.

(3) Feed coal :- Landau No. 3 duff, $-\frac{1}{4}$" +1 mm.

(4) Pulp ratio, 6 to 1.

(5) Feed pressure, 10 lb. per sq. inch.

(6) Orifice tubes projecting $2\frac{1}{2}$ inches beneath cover plate were used in all tests except 71C in which the tube length was $1\frac{1}{4}$ inches (as shown previously, this should not affect the efficiency).

Test No.	71C	73H	72G	73E	75F	73A	72C	75D
Orifice dia. inches	$1\frac{7}{8}$	$1\frac{7}{8}$	$2\frac{17}{32}$	$2\frac{17}{32}$	$2\frac{17}{32}$	3	$3\frac{3}{8}$	$3\frac{3}{8}$
Nozzle dia. inches	1	$\frac{7}{8}$	$1\frac{3}{8}$	$1\frac{3}{8}$	$1\frac{1}{2}$	$1\frac{1}{2}$	$1\frac{5}{8}$	$1\frac{5}{8}$
Yield of product, %	26.2	27.3	50.3	51.9	44.6	52.9	30.7	33.0
Ash of product, %	5.5	5.3	8.0	8.5	7.3	9.4	7.4	7.4
Ash of feed, %	13.1	12.8	12.8	14.2	12.8	14.4	14.0	13.8
S.G. of separation	1.34	1.32	1.39	1.42	1.375	1.43	1.335	1.335
Capacity, tons/hour	3.5	2.7	3.3	4.7	3.5	6.5	5.9	4.8
Rate of discharge of refuse, tons/hr/in nozzle diameter	2.58	2.24	1.19	1.64	1.29	2.04	2.52	1.98
Total refuse tons/hr.	2.58	1.96	1.63	2.26	1.94	3.06	4.09	3.22
Rate of flow of water lb. per sec.	7.82	-	-	-	10.38	12.37	13.33	-
Feed pipe velocity ft. per sec.	10.2	-	-	-	13.5	16.1	17.4	-
Centrifugal acceleration at the radius of the orifice, ft/sec²	5700	-	-	-	5400	5500	5100	-
Flow of water through nozzle cu.ft/sec.	0.0074	-	-	-	0.0138	0.0107	0.0095	-

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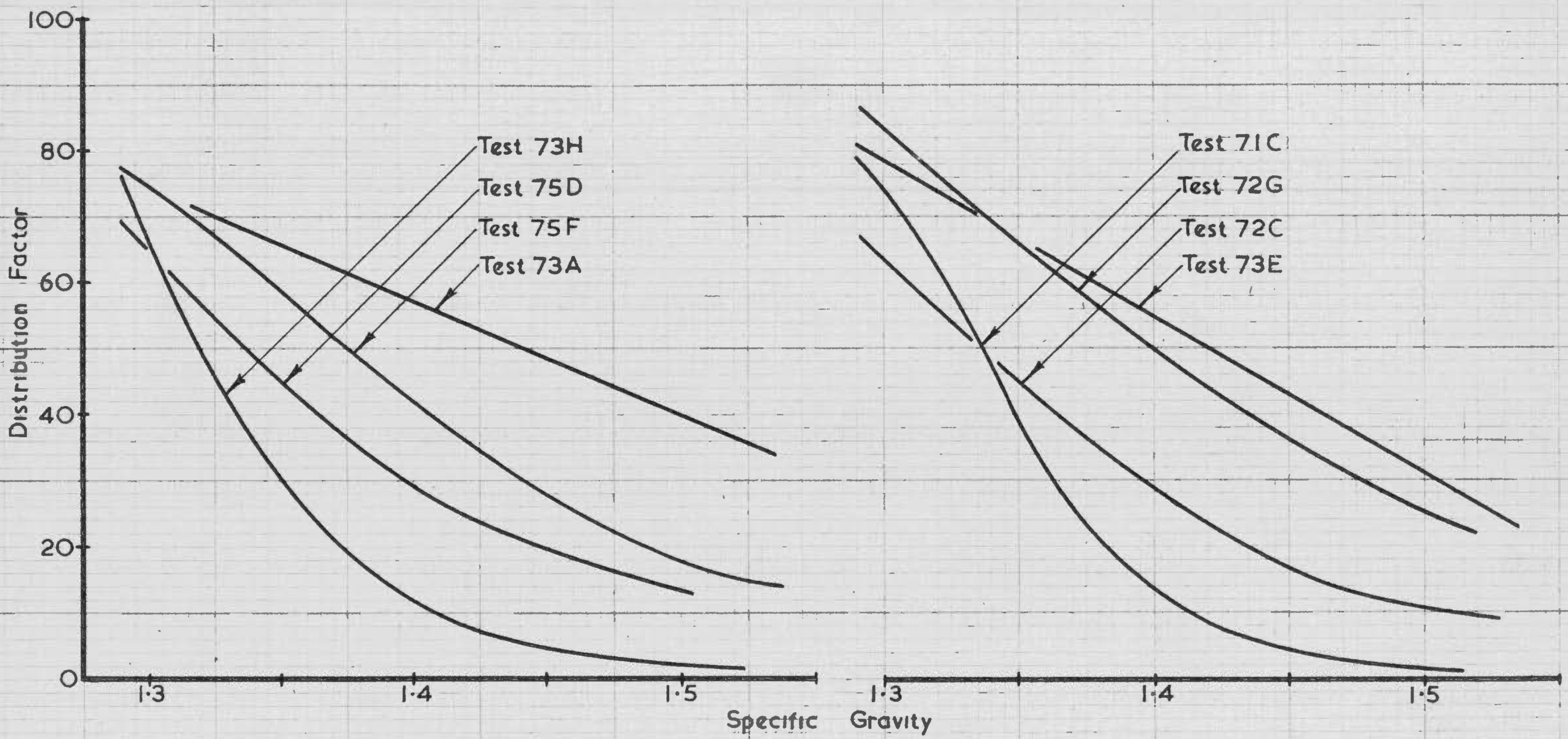

FIGURE 18 Tromp curves relating to Table 13

It can be proved that the washing efficiency is a function of both the washer efficiency and the specific gravity of separation for any particular coal. Since the specific gravity of separation varies considerably in the present series of tests, the values for washing efficiency would be misleading and have consequently been omitted. Comparison of the efficiencies of separation obtained in the various tests will, therefore, be confined to a consideration of the Tromp distribution factor curves shown in Fig. 18.

Although the results of the tests shown in Table 13 and Fig 18 convey the impression that the efficiency of separation decreases substantially as the orifice diameter is increased, it is necessary to analyse these tests carefully before it can be decided in what measure this loss of efficiency may be attributed to the diameter of the orifice.

It will be observed that there is a general tendency for the capacity of the cyclone to increase as the orifice and nozzle diameters are made larger. Since the capacity values are affected by variations in the pulp ratio, four of the tests under discussion were repeated using water as feed, in order to establish the relationship between the rate of flow and the orifice diameter. The results of these tests are also shown in Table 13. It will be seen that the rate of flow increases appreciably as the orifice diameter is increased. Thus, increase in the orifice diameter from $1\frac{7}{8}$ to $3\frac{3}{8}$ inches increases the rate of flow by about 70 per cent. In other words, increase of the orifice diameter decreases the resistance to flow, hence the quantity passing in unit time increases, the applied pressure being constant. It will be recalled that a similar effect has been observed before.

Assuming the period of pulp retention to be inversely proportional to the rate of flow, it follows that the time available for the separation to be completed decreases with increasing orifice diameter and this factor should be taken into account

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when comparing the efficiencies obtained for the various orifices indicated in Table 13. Now the influence of the period of pulp retention could be eliminated by operating a small orifice at a higher pressure than a larger orifice, the pressures being selected so that the rate of flow of feed is the same in both cases. However, as will be shown later, the centrifugal acceleration increases with increasing rate of flow, all other variables being constant. This fact would render the smaller orifice comparatively even more efficient than the larger orifice when both are used at the same feed rate, particularly when small particles are present in the feed.

Returning to the tests in Table 13, consider the values obtained for the nozzle loading. It is known from previous experience that the nozzle was undoubtedly overloaded in tests 71C and 73H (see Tests 76A and 47B, Table 9) and hence it may be concluded that even sharper separations would have been obtained had this not been the case. The nozzle loadings for the remaining tests can not, however, be interpreted without considering the effect of the orifice diameter on the rate of flow through the nozzle.

Test 75F may be considered as an example. In an earlier test (35A Table 11) a nozzle of $1\frac{1}{2}$ inches diameter was used with an orifice of $1\frac{7}{8}$ inches diameter and a nozzle loading of 1.93 tons per hour per inch did not cause loss of efficiency. The question now arises whether the $1\frac{1}{2}$ inches diameter nozzle used in test 75F was overloaded or not, the nozzle loading being only 1.29 tons per hour per inch. Now, it will be seen that the rate of flow of water through the nozzle was 0.0138 cubic feet per sec. in this test. From Table 6, it is found that the rate of flow of water through a $1\frac{1}{2}$ inches diameter nozzle when associated with a $1\frac{7}{8}$ inches diameter orifice, is 0.0387 cubic feet per second for the same pressure. In other words, the rate of flow through a $1\frac{1}{2}$ inches diameter nozzle is increased almost three times by reducing the orifice diameter from $2\frac{17}{32}$ to $1\frac{7}{8}$ inches.

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If the maximum permissible nozzle loading is assumed to be proportional to the rate of flow through the nozzle, its value would vary accordingly. Since insufficient data is available regarding the maximum rate of discharge of refuse for various rates of flow, it is not possible to determine whether the nozzle was overloaded in Test 75F. However, since the normal rate of flow through the nozzle in Test 75F is approximately twice that of the corresponding value for Test 71C (0.0138 cubic feet per second compared with 0.0074 cubic feet per second) and since the rate of discharge of refuse was lower (1.94 tons per hour compared with 2.51 tons per hour), it may be concluded that the nozzle was overloaded to a greater degree in Test 71C than it was in Test 75F.

This implies that the difference in the efficiencies likely to be obtained under normal operating conditions, when using the $1\frac{7}{8}$ and $2\frac{17}{32}$ inches diameter orifices, would be even more marked than is indicated by Fig.18.

It is clear from these tests that, although the feed consisted of relatively coarse particles, the efficiency of separation is substantially impaired when the orifice diameter is $2\frac{17}{32}$ inches or larger. This loss of efficiency may be attributed partly to the fact that both the period of pulp retention and the centrifugal acceleration at the radius of the orifice decrease with increasing orifice diameter. Both these factors would become progressively more important as the feed decreases in size. In order to obtain the optimum efficiency for all size grades, it follows that the orifice should be as small as possible. A large capacity, on the other hand, requires that the orifice diameter should be as large as possible. The final selection will, therefore, represent a compromise between these opposing requirements and will mainly be influenced by the size grading of the feed. As it was found that reasonably sharp separations could be obtained for particle sizes down to about 100 mesh when using a $1\frac{7}{8}$ inches diameter orifice, this

orifice..../

orifice was selected as being the largest suitable for washing Witbank duff. It was evident that as the orifice diameter increases above $1\frac{7}{8}$ inches, the efficiency of separation of the smaller sizes would first be impaired and eventually that of the coarser particles, as was the case when an orifice of $2\frac{17}{32}$ inches diameter was used.

Comparing the values of the rate of flow of water through the nozzle shown in Table 13, it will be observed that this value is not increased materially when both the orifice and nozzle diameters are made larger in order to maintain a specified separation. For example, in Test 72C the nozzle diameter was $1\frac{5}{8}$ inches as compared with one inch in Test 71C, and although the nozzle diameter had been increased 62.5 per cent the rate of flow through the nozzle was only increased by about 28 per cent. On the other hand, in the case of the tests reported in Table 6, the orifice diameter was constant at $1\frac{7}{8}$ inches and an increase in the nozzle diameter from $1\frac{1}{4}$ to $1\frac{5}{8}$ inches (an increase of 30 per cent in the nozzle diameter and a decrease of 23 per cent in the ratio between the orifice and nozzle) increased the rate of flow through the nozzle by 125 per cent. In view of the fact that the ratio between the diameter of the orifice and the diameter of the nozzle was almost the same in tests 72C and 71C, it may be concluded that the value of this ratio is a more important factor determining the rate of flow through the nozzle than the actual diameter of this aperture. Thus, assuming the maximum nozzle loading to be proportional to the rate of flow through the nozzle, it follows that the ratio between the orifice and nozzle diameters must be small, when effecting a separation which yields a large percentage tailing, in order to obtain a large capacity i.e. a relatively high suspension specific gravity must be used.

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THE INFLUENCE OF THE RELATIONSHIP BETWEEN THE DIAMETER OF THE NOZZLE AND THE GRADING OF THE FEED COAL.

It was realised early in the investigation that the nozzle, being the smallest aperture in most cases, would determine the maximum size of coal which could be treated under any given conditions. It also seemed reasonable to assume that the flow through the nozzle would be disturbed if the coal particles were large compared with the diameter of the nozzle and that such conditions may have an adverse effect on the efficiency. In order to investigate the influence of the relationship between the nozzle diameter and the size of the feed on cyclone performance, several tests were carried out in which all cyclone operating adjustments were kept constant with the exception of the size grading of the feed. The results of these tests are reported in Table 14, and the Tromp distribution factor curves relating to this table are shown in Fig.19. The size distribution of each tailing obtained was determined by screen analysis and is plotted in Fig.20, (actual values obtained by analysis are shown in Appendix 5).

A suspension specific gravity of 1.2 was selected so that the orifice would be large compared with the nozzle. The separations would not, therefore, be affected by the relationship between the diameter of the orifice and the size of the feed. The size of the material actually flowing through the nozzle would then be the only variable. It is for this reason that the ensuing discussion is based on the size distribution of the tailing and not on that of the feed.

The mean size of the tailing was determined by means of the following formula (15) :-

$$\text{mean diameter} = \frac{\sum wd}{\sum w}$$

where w = fractional yield as a percentage by weight between any two screen sizes.

d = arithmetic mean aperture of the two screens between which w was determined.

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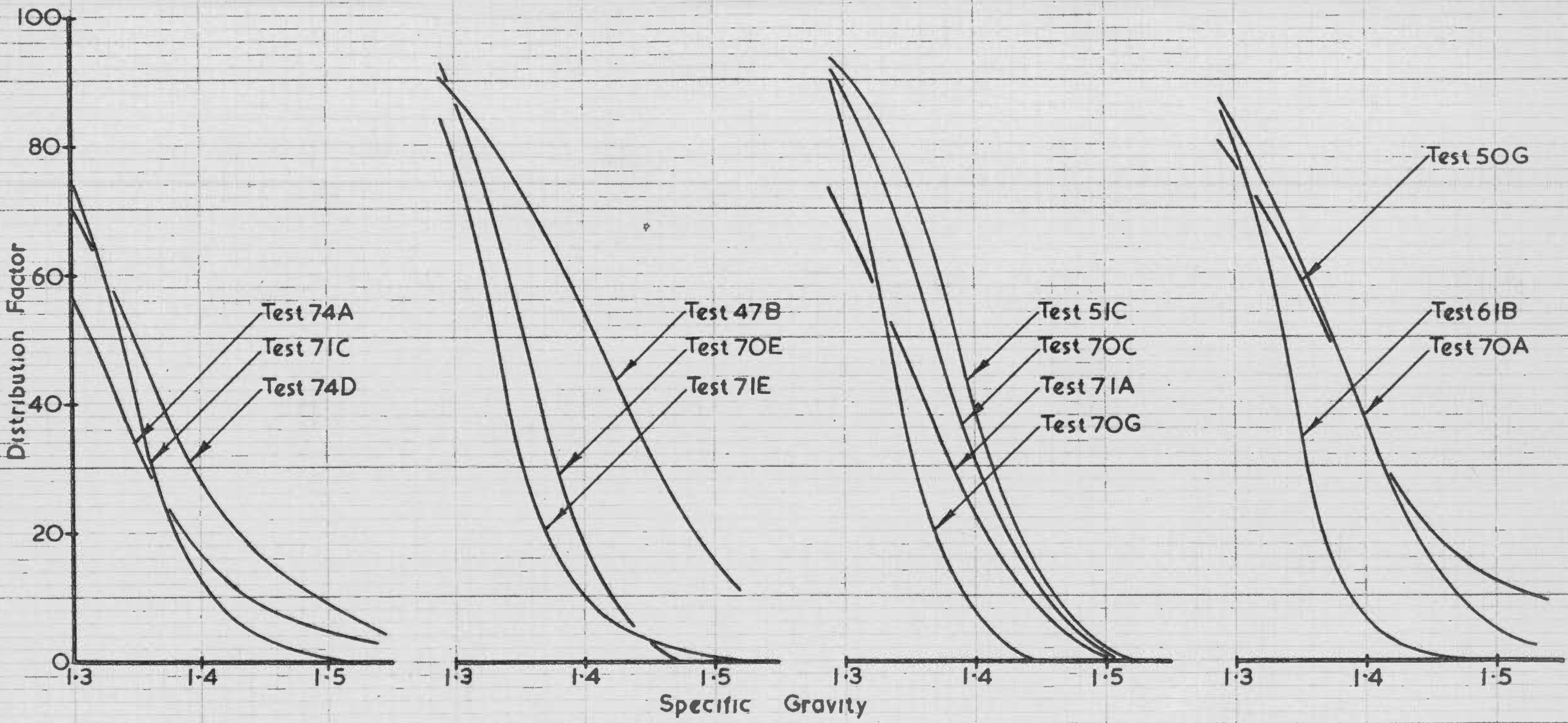

FIGURE 19 Tromp curves relating to Table 14

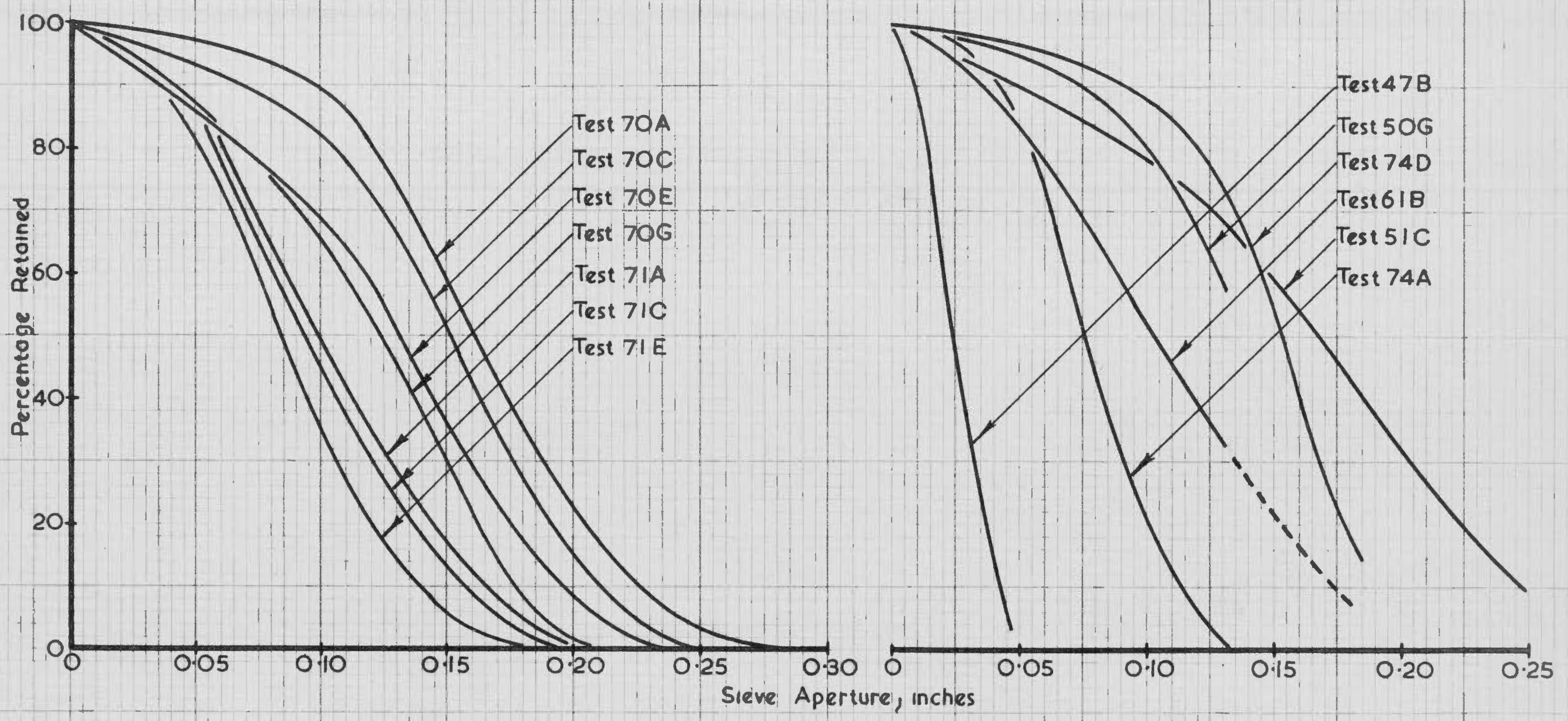

FIGURE 20 Size distributions of the tailings

There are several more elaborate (and possibly more accurate) methods of determining the mean diameter of a sample of coal, but the additional work involved was not considered justifiable for the purpose of the present investigation.

It is unfortunate that a pulp ratio of 6 to 1 was used for these tests as the nozzle was consequently overloaded in the majority of cases. Several of the tests had already been carried out, however, before the importance of nozzle loading was fully appreciated. As the results obtained indicated that the size gradings of the Witbank duff coals, which it was proposed to wash finally, would not have appreciable influence on the separation provided that a reasonably large nozzle was used (the condition also for large capacity), it was decided not to repeat the series using a higher pulp ratio.

Comparing the results of the tests shown in Table 14, (see page 59a) it will be seen that there is a general tendency for the specific gravity of separation to increase with increase of mean tailing size above about 0.12 inches. In order to illustrate this effect more clearly, specific gravity of separation was plotted against the mean tailing size in Fig. 21. Neglecting test 74A, it will be seen that there appears to be a trend of the form indicated viz:-

(1) increasing specific gravity of separation with <u>decreasing</u> mean tailing diameter to the left of A.

(2) constant specific gravity of separation between A and B.

(3) increasing specific gravity of separation with increasing mean tailing diameter to the right of B.

Now, as will be explained later, it is usual in heavy medium processes for the specific gravity at which particles are separated to increase as the size of the particles become smaller than a certain minimum value, depending on the size of heavy medium particles in the suspension. Material larger than this lower limit of size is cut at approximately constant specific gravity irrespective of particle size.

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TABLE 14.

THE INFLUENCE OF FEED SIZE ON CYCLONE PERFORMANCE

The following conditions were constant for all tests :-

(1) $9\frac{1}{2}$" diameter cyclone.
(2) 25 degrees apex angle.
(3) $1\frac{1}{2}$ inches diameter feed pipe.
(4) $1\frac{7}{8}$ inches diameter orifice tube projecting $1\frac{1}{4}$" beneath cover plate.
(5) 1" diameter nozzle.
(6) 1.2 Specific gravity suspension.
(7) 6:1 pulp ratio.
(8) Feed pressure, 10 lb. per sq. inch.
(9) Feed coal: Landau No.3 duff, screened as indicated.

Test No.	47B	74A	71E	71C	71A	61B	70G	70E	50G	70C	74D	70A	51C	52A
Nominal grading of feed.	-1 mm +60 #	-1/8" +1 mm	-1/4" +1 mm	-1/4" +1 mm	-1/4" +1 mm	-1/4" +1 mm	-3/8" +1/8"	-3/8" +1/8"	-1/4" +1/8"	-3/8" +1/8"	-1/4" +1/8"	-3/8" +1/8"	-3/8" +1/4"	-1/2" +1/4"
Product yield, %	50.1	24.1	25.3	26.2	26.0	21.5	28.0	33.8	37.0	39.6	34.1	36.9	42.7	
Product ash, %	7.0	6.4	5.0	5.5	5.9	5.7	-	6.6	7.6	6.7	7.4	-	6.8	Bridging of nozzle
Feed ash, %	14.4	12.7	13.1	13.1	13.9	13.4	-	13.7	-	12.6	12.8	-	12.4	
Specific gravity of separation	1.41	1.315	1.335	1.34	1.34	1.34	1.335	1.355	1.37	1.37	1.35	1.375	1.385	
Mean diameter of tailing, inches	0.026	0.078	0.088	0.098	0.104	0.112	0.119	0.126	0.147	0.148	0.151	0.162	0.164	
Capacity, tons per hour	3.0	3.7	2.9	3.5	4.4	2.8	2.5	3.2	5.2	3.9	4.7	5.3	4.7	
Refuse, tons per hour.	1.5	2.81	2.17	2.58	3.26	2.2	1.8	2.12	3.27	2.35	3.1	3.34	2.69	
Percentage tailing coarser than 0.18"	0	0	1.5	3.0	4.5	7.5	9.0	16.5	-	26.5	20.0	36.0	43.0	

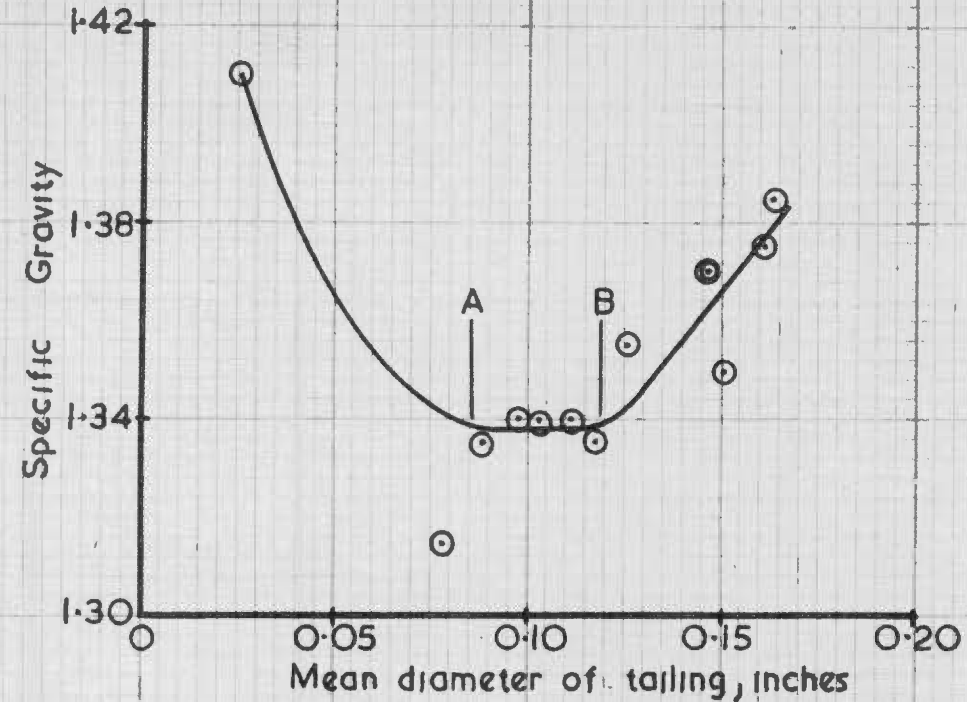

FIGURE 21 Influence of mean diameter of tailing on Specific Gravity of Separation

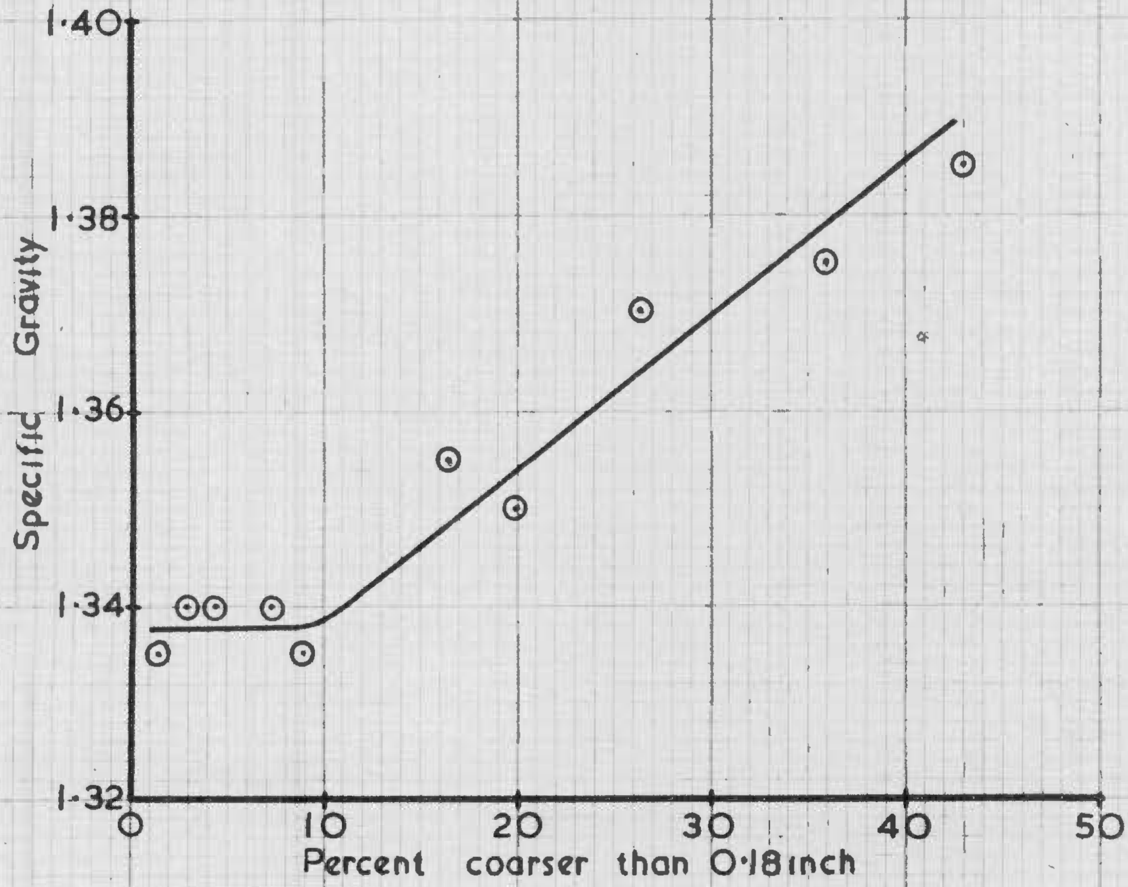

FIGURE 22 Influence of large particles in the tailing on Specific Gravity of Separation

The shape of the curve to the left of A (Fig. 21) could, then, be accounted for in this way and, in the absence of all other factors, the separation should be effected at a constant specific gravity when the particle size exceeds the limiting value mentioned above. Although this appears to be the case between A and B, the curve deviates from the theoretical to the right of B. This deviation is attributed to an additional "nozzling effect" caused by the passage of large particles through the nozzle i.e. the coal appears to be too coarse for a one inch diameter nozzle. Operation in this zone would be undesirable, as small fluctuations in the size grading of the feed would cause comparatively large variations in the specific gravity of separation and thus complicate control of the washer.

Although Fig. 21 indicates that the size of the coal undoubtedly affects the specific gravity of separation, it can not be assumed that the mean diameter of the tailing is the only criterion. Thus, it appears reasonable to assume that large particles would cause more disturbance than smaller particles. If this is so, one could expect the specific gravity of separation to be influenced to a greater extent by a tailing containing a comparatively large proportion of coarse particles than by a tailing composed only of smaller particles of such size that the mean diameter is the same for both.

In order to investigate this possibility, a size of 0.18 inch was arbitrarily chosen and the percentage of material coarser than this was determined for each tailing. (A size of 0.18 inch was selected as this is the largest value for which reasonably accurate readings can be taken from Fig. 20). If tests 74D and 70C are now compared, it will be observed that the mean size of the tailing was practically the same for both tests but that a higher specific gravity of separation was obtained in Test 70C. It will also be seen that the percentage of material coarser than 0.18 inch in the tailing was greater in test 70C than it was in test 74D. Since a similar effect will be noted if tests 70A and 51C are compared, there appears to be some ground

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for the assumption that the coarser particles have a greater influence than the finer particles. This is further supported by Fig.22, in which the percentage of tailing coarser than 0.18 inch has been plotted against specific gravity of separation. The diagram obtained indicates almost linear increase in the specific gravity of separation when the percentage of material coarser than 0.18 inch exceed about 10 per cent.

It is interesting to note that in all tests in which the percentage of tailing larger than 0.18 inch is less than 10 per cent, the percentage of material coarser than 0.2 inch in the tailing is negligible. This fact suggests that 0.2 inch is approximately the critical size of tailing for a one inch diameter nozzle.

Although the investigation of the influence of the relationship between the diameter of the nozzle and the size grading of the feed on cyclone performance can not be considered to have been completed, there appears to be sufficient evidence to support the conclusion that the specific gravity of separation is affected by the mean size of the tailing and the size distribution and particularly by large particles under the operating conditions specified. It was clear that a considerable amount of additional data would be required to establish the exact relationship between these factors and, bearing in mind the object of the investigation as a whole, it was considered that further work in this connection would not be justified.

In test 51C, the feed coal was screened -3/8 inch +1/4 inch and, although the coal was degraded by the time the samples were taken, these comparatively coarse particles did actually pass through the nozzle during the charging period, and no difficulty was experienced due to bridging of the nozzle. Feed coal screened -½ inch +¼ inch (test 52A), on the other hand, very soon obstructed the nozzle, and flow of tailing ceased. From this it appears that the coarsest particles present should not exceed, say, 3/8 inch when a one inch diameter nozzle is used

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(or about $\frac{1}{5}$ of the nozzle diameter). In several other tests in which the nozzle was obstructed, it was found that bridging only occured when the largest particles exceeded about $\frac{1}{3}$ the nozzle diameter. Naturally, bridging of the nozzle would be caused by the combined effect of size and relative abundance of the largest particles.

Referring again to Table 14, it will be observed that the rate of discharge of refuse varied considerably in the various tests and that the nozzle was overloaded in most cases. Since it appeared from previous tests that this factor influences both the specific gravity of separation and the efficiency, it is necessary to determine to what extent this variation in the rate of discharge of refuse affects the conclusions already drawn.

In tests 71A, 71C and 71E the size distribution and mean diameter of the tailings did not differ appreciably and, although the nozzle loading varied between 2.17 and 3.26 tons per hour per inch, the specific gravity of separation was virtually constant. Similarly, in tests 50G and 70C, the mean size of the tailing was nearly the same and the separations were effected at the same specific gravity despite the fact that the nozzle loading varied between 2.35 and 3.27 tons per hour. It does not, therefore, appear that variation of the nozzle loading between about 2.17 and 3.27 tons per hour affects the specific gravity of separation materially. In any event, this variable may be largely eliminated by considering only those tests in which the rate of discharge of refuse was approximately constant. For example, the tests in Table 14 may be divided into two main groups viz:-

Group 1:- tests 71E, 61B, 70E and 70C having an average nozzle loading of about 2.2 tons per hour.

Group 2:- tests 71A, 50G, 74D and 70A having an average nozzle loading of about 3.25 tons per hour.

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It will be seen that each of these groups indicate that the specific gravity of separation increases with size grading of tailing in the manner already described. It does not appear, then, that the general conclusions drawn before should be modified on account of the variations in nozzle loading. By operating at very low capacity, it may be possible to treat relatively coarse material satisfactorily, but this was not investigated.

Comparison of the Tromp distribution factor curves in Fig. 19 shows that the curves for the tests in Group 2 are distinctly inferior to those for Group 1. From this it appears that increase in the nozzle loading from about 2.2 to 3.25 tons per hour has resulted in a marked loss of separating efficiency.

It will be clear that the influence of large particles passing through the nozzle on efficiency is somewhat obscured by the variation in the nozzle loading. Since the efficiency of separation also depends on the size grading of the feed, even when the flow through the nozzle is not disturbed, it follows that no reliable conclusions can be drawn from these tests regarding the influence of the relationship between the nozzle diameter and the size grading of the feed on separating efficiency.

THE INFLUENCE OF FEED PRESSURE ON EFFICIENCY OF SEPARATION:

Since the feed pressure determines the rate of flow of pulp through the cyclone and hence the period of pulp retention and the magnitude of the centrifugal force acting on a particle, it was clear that this variable would have an important bearing on the capacity and the efficiency of separation.
In earlier tests planned to investigate the influence of this variable, disc-type orifices were used and the results obtained indicated that the feed pressure had a critical value for optimum efficiency. However, later tests proved that this only applied to disc-type orifices and that the pressure required for optimum efficiency depended on the size grading of the feed.

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A typical example of a series of tests carried out on a disc-type orifice at various feed pressures is shown in Table 15. At that stage of/the investigation, the value of the Tromp distribution factor curves was not fully appreciated and these curves were consequently not determined. An approximate method,[14] developed at the Fuel Research Institute, was used to estimate the specific gravity of separation and the efficiency or performance factor. The value of the performance factor does not differ appreciably from that obtained by the Fraser and Yancey expression.

TABLE 15.

The influence of feed pressure on cyclone performance when using a disc-type orifice.

The following conditions were constant for all tests :-

(1) $9\frac{1}{2}$ inches diameter cyclone, 38 degrees apex angle, $1\frac{1}{2}$ inches diameter feed pipe.
(2) Diameter of disc orifice, $1\frac{7}{8}$ inches.
(3) Diameter of nozzle, $1\frac{3}{4}$ inches.
(4) Specific gravity of suspension, 1.28.
(5) Feed coal:-Landau No. 3 duff, $-\frac{1}{4}$ inch + 1 mm.
(6) Pulp ratio, 6 to 1

Test No.	8A	8B	8C	8D	8E	8F	8G	8H	8I
Feed pressure, lb/sq.in.	24	19	14.2	9.4	7	4.6	2.6	1.6	1.2
Yield of product, %	32.8	39.2	40.1	43.1	44.6	41.6	30.8	19.5	nil
Ash content of product, %	7.0	6.8	6.3	5.9	5.7	5.7	5.1	4.9	-
Ash of feed, %	12.9	12.5	12.3	12.0	11.6	11.7	11.6	12.2	-
Sp.Gr.of separation.	1.36	1.375	1.37	1.37	1.37	1.365	1.335	1.31	-
Performance factor.	56	64	72	81	85	81	75	72	-
Capacity, tons per hr.	3.32	2.88	2.41	1.77	1.62	1.35	0.93	-	-

To facilitate interpretation, the results of these tests were plotted against feed pressure as shown in Fig.23. Since the ash content of the feed did not vary substantially, it was assumed that the composition of the feed remained reasonably constant and that the yield and ash values of the product could be taken as an indication of the efficiency.

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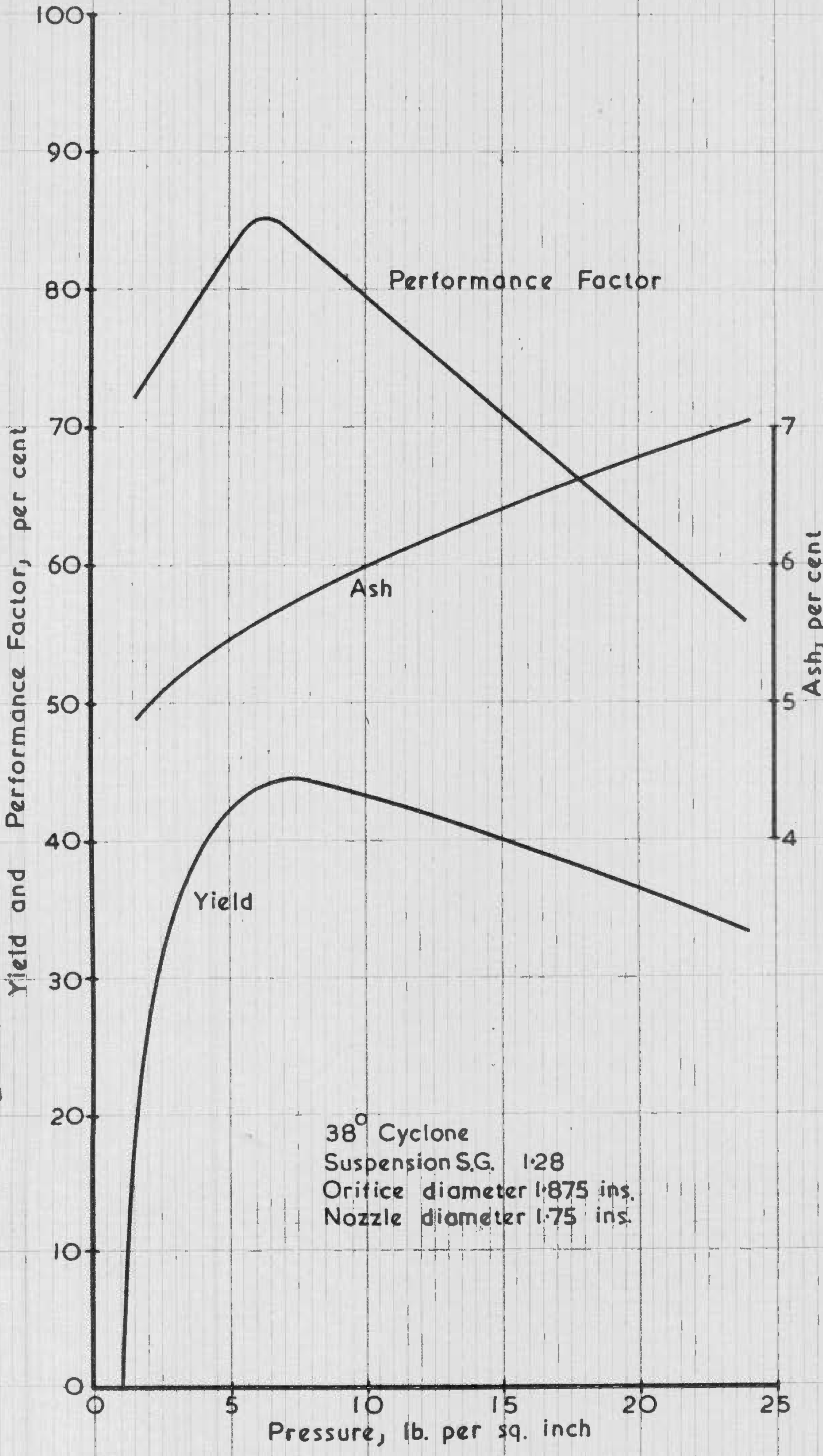

FIGURE 23 The influence of pressure when using a disc orifice

From Fig. 23, it will be seen that the yield of product increased rapidly as the pressure was raised from 1.2 to about 7 lb. per square inch and then decreased at a slower rate as the pressure was raised further. Since the ash content of the product increased steadily with rise in pressure, the performance factor (or efficiency) also increased at first and then commenced to decrease again, attaining a maximum value at about 7 lb. per square inch. From this it was concluded that the $9\frac{1}{2}$ inches diameter cyclone should be operated at about 7 lb. per square inch for optimum efficiency (when using the operating conditions stated) and that the maximum capacity attainable would, therefore, be comparatively low. Other tests appeared to confirm this view. These tests also indicated that the specific gravity of separation increased rapidly until the critical pressure was reached and then showed a slight tendency to decrease as the pressure was further increased.

When it had been established that a proportion of the feed by-passed the cyclone when using a disc-type orifice, it was thought that this might possibly account for the type of pressure characteristic illustrated by Fig.23. Several tests were consequently carried out to investigate the influence of feed pressure when using an orifice tube. A typical example of such a series of tests is given in Table 16. Comparatively fine feed coal was used for these tests in order to minimise the effects of degradation and settling of coal in the pulp.

TABLE 16 ..../

TABLE 16.

The influence of feed pressure on cyclone performance when using an Orifice Tube.

The following conditions were constant for all tests :-

(1) $9\frac{1}{2}$ inches diameter cyclone, 25 degrees apex angle, $1\frac{1}{2}$ inches diameter feed pipe.

(2) $1\frac{7}{8}$ inches diameter orifice tube projecting $1\frac{1}{4}$ inches beneath cover plate.

(3) $1\frac{5}{8}$ inches diameter nozzle.

(4) Feed coal:- Landau No. 3, -1 mm +60 mesh.

(5) Specific gravity of suspension 1.3.

(6) Pulp ratio, 6 to 1.

Test No.	31A	31B	31C	31D	31F
Feed pressure, lb/sq.inch	25	$17\frac{1}{2}$	10	$7\frac{1}{2}$	4
Yield of product, %	33.3	38.1	39.4	37.6	23.0
Ash of product, %	5.7	5.7	5.6	5.7	5.3
Ash of feed, %	15.6	15.0	14.6	15.0	15.7
Specific Gravity of separation	1.35	1.365	1.368	1.37	1.32
Washing efficiency, %	86	87	87	85	84
Capacity, tons per hour	3.6	2.9	2.0	1.8	1.38

The Tromp distribution factor curves for the series of tests in Table 16 are shown in Fig. 24. If these curves are compared, it will be seen that, possibly with the exception of test 31B, the efficiency of separation was not affected appreciably by variation in the feed pressure between 4 and 25 lb. per sq. inch. In other tests, (see Fig.26), even smaller differences in the shape of the Tromp distribution factor curves was observed when the pressure was varied between these limits.

It may be concluded, therefore, that the feed pressure is not critical as regards its influence on efficiency, and that the effect noted in the previous series of tests (Table 15) was due to feed by-passing the cyclone i.e. the percentage of feed by-passing increased as the pressure was raised.

The tests in Table 16, again indicate that the specific gravity of separation increases rapidly as the pressure is raised to about $7\frac{1}{2}$ lb. per square inch and then shows a tendency to decrease for further rise in pressure i.e. the specific gravity of

separation.../

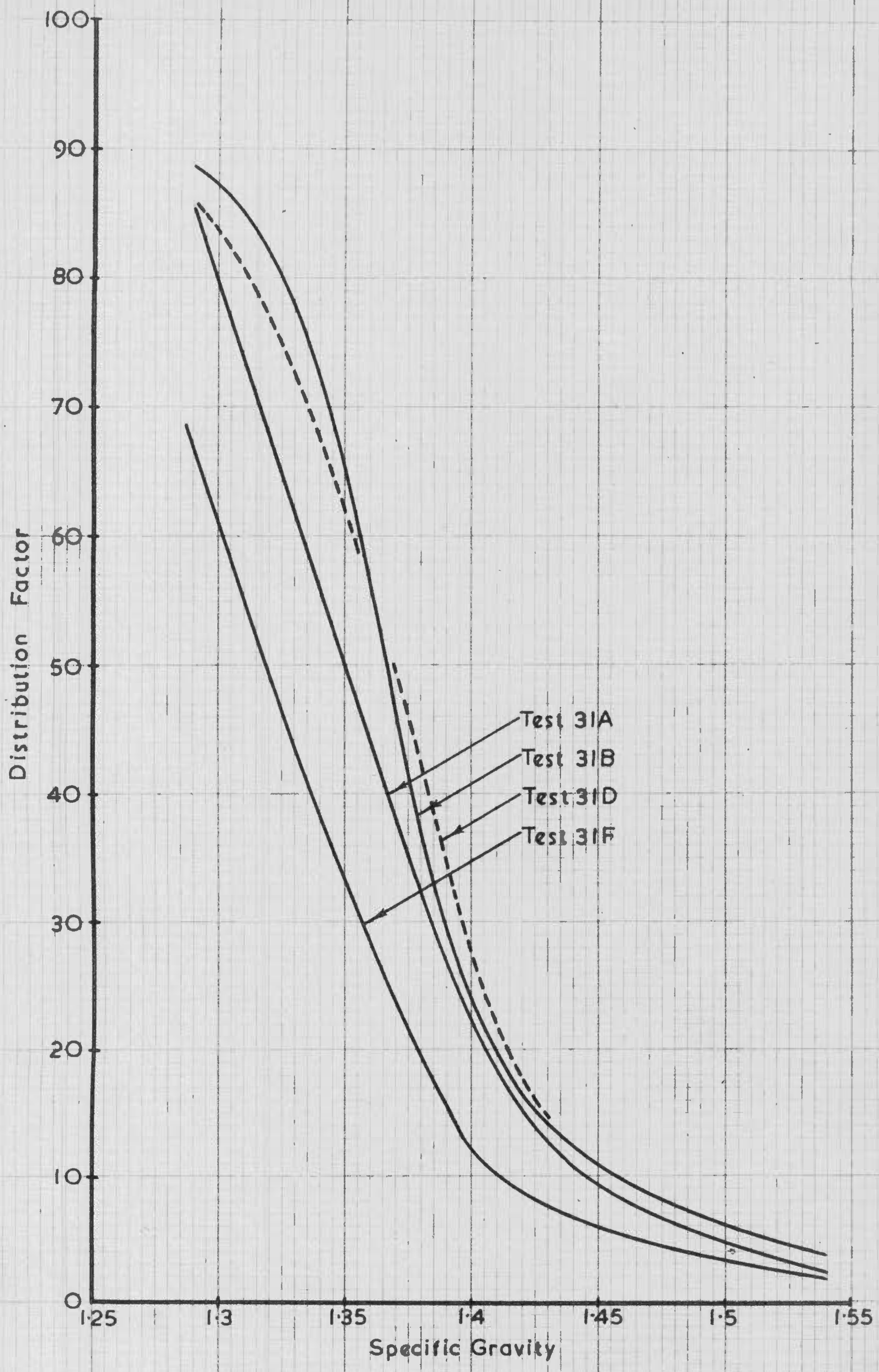

FIGURE 24 Tromp curves relating to Table 16

separation appears to attain a maximum value in the region of $7\frac{1}{2}$ to 10 lb. per sq. inch. It is for this reason that the majority of tests planned to investigate the influence of other variables were carried out at these low pressures. The influence of pressure on the specific gravity of separation will be discussed in detail in the next chapter where the effect of using feed pipes of different diameters is considered.

Although variation in the feed pressure between the limits indicated did not appear to affect the efficiency to any degree, it was reasoned that this would only apply to the relatively coarse feed coal which had been used, (i.e. the separation of small particles is more likely to be influenced by variation in the centrifugal forces, etc., than that of larger particles because their separating velocities are much lower). Since it was not feasible to prepare large quantities of suitably fine feed coal by screening, tests were carried out on feed coal of wide size grading. The products and tailings obtained were then divided into a number of suitable size fractions and each fraction was separately subjected to float and sink analysis. In this way it was hoped to determine the influence of pressure on efficiency for various sizes of feed particle.

The results of two tests, carried out at 10 and 40 lb. per sq. inch respectively, are shown in Table 17. Alteration of the specific gravity of suspension and nozzle diameter was necessary as the specific gravity of separation decreases substantially when a $1\frac{1}{2}$ inches diameter feed pipe is used and the feed pressure is increased from 10 to 40 lb. per sq. inch. The Tromp distribution factor curves for various size fractions relating to these tests are shown in Fig. 25.

TABLE 17...../

TABLE 17.

The following conditions were constant for both tests:-

(1) $9\frac{1}{2}$ inches diameter cyclone, 15 degrees apex angle, $1\frac{1}{2}$ inches diameter feed pipe.

(2) $1\frac{7}{8}$ inches diameter orifice tube projecting $1\frac{1}{4}$ inches beneath cover plate.

(3) Feed coal :- Coronation duff, $-\frac{1}{2}$" to 0 (plus 100 mesh material recovered after washing in the cyclone).

(4) Pulp ratio, 3 to 1.

Test No.	91A	93C
Feed pressure, lb.per sq.inch.	10	40
Suspension specific gravity	1.30	1.32
Nozzle diameter, inches.	$1\frac{5}{8}$	$1\frac{1}{2}$
Yield of product, %	21.9	31.4
Ash of product, %	5.2	5.4
Ash of feed, %	11.7	12.1
Specific gravity of separation	1.34	1.36
Capacity, tons per hour	5.7	9.5
Refuse, tons per hour, per inch	2.74	4.35
Rate of flow of water, lb.per sec.	8.98	19.0
Feed pipe velocity, ft. per sec.	11.7	24.8
Centrifugal acceleration at radius of orifice, ft. per sec.^{2}	7,400	33,000

If the Tromp distribution factor curves for corresponding size fractions of each test are compared (Fig.25), it will be noted that the efficiency of separation was practically the same at both pressures for particles larger than about 48 mesh. Sharper separations were obtained, however, at the higher pressure for smaller size fractions. This proves that the feed pressure is only of importance in the case of small particles and that the pressure to be used would thus be determined by the size grading of the feed i.e. the greater the percentage of material finer than about 48 mesh in the feed, the higher would be the feed pressure required for optimum efficiency.

Increase of pressure increases the rate of flow through the cyclone if all other variables are assumed to be constant. Increased rate of flow, in turn, increases the

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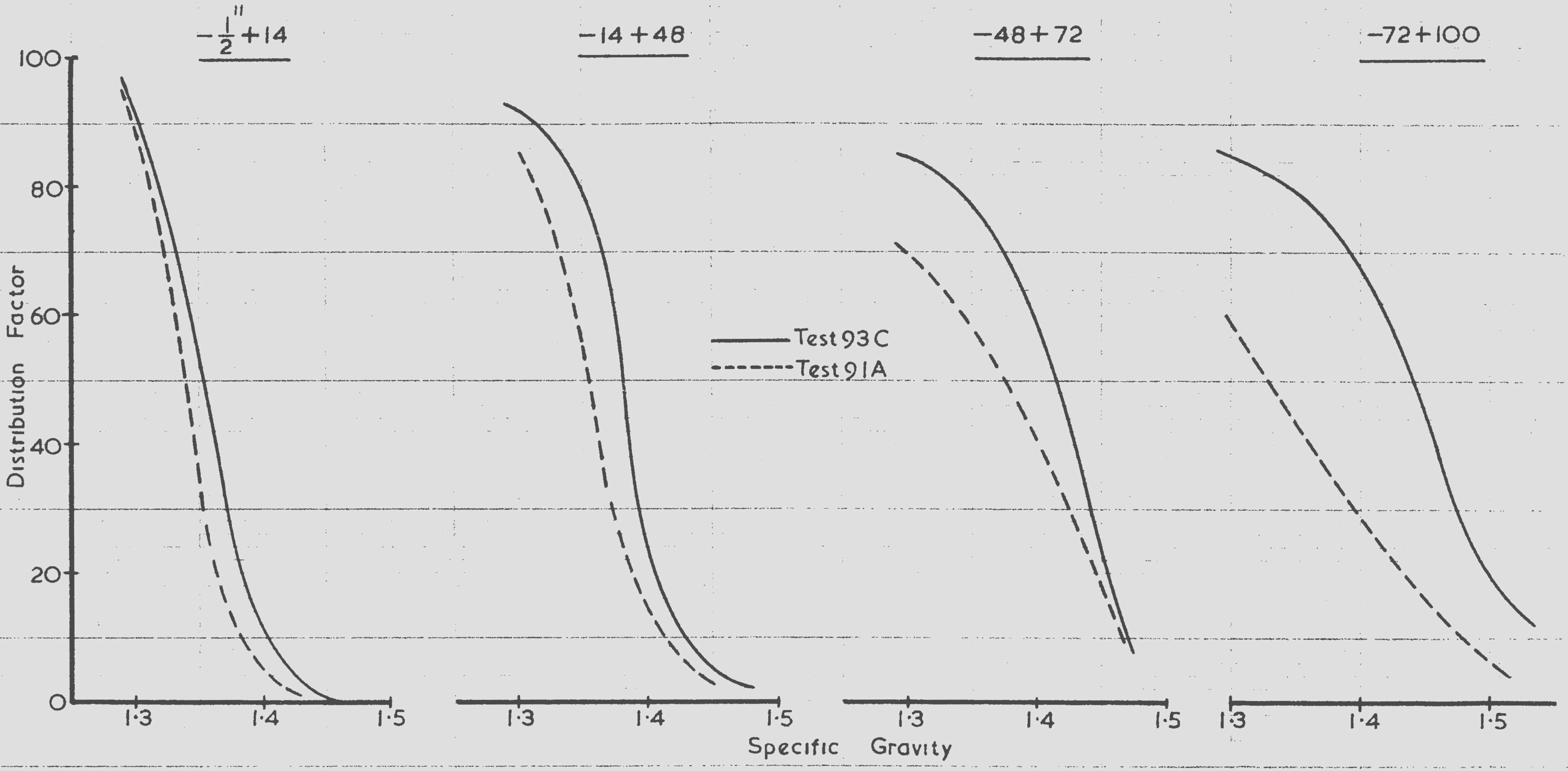

FIGURE 25 Tromp curves relating to Table 17

tangential velocity at all points in the cyclone and decreases the time spent by a particle in the apparatus. Now, the time required for the separation of a particle to be completed depends on its separating velocity, which is determined by the force acting on it, all other factors being equal. Thus, if the tangential velocity of a particle increases, the centrifugal force acting on the particle increases (as the square of tangential velocity) and the time required for separation to be effected is reduced. Since the time spent in the cyclone decreases and the efficiency increases for small particles when the pressure is raised,it may be concluded that the separating time required by a particle decreases at a greater rate than the period of pulp retention. In the case of larger particles, the rate of separation is apparently sufficiently high when acted upon by comparatively small centrifugal forces to ensure complete separation at low pressure. Increase of pressure would thus serve no useful purpose except as a means of increasing the capacity.

In order to obtain a conception of the influence of the feed pressure on the magnitude of the forces acting on the particles and on the rate of flow, the tests in Table 17 were repeated using water only. The rate of flow was determined from these tests and hence the centrifugal acceleration at the radius of the orifice, as shown in Table 17. It will be seen from these results that increase in pressure from 10 to 40 lb. per sq. inch approximately doubles the rate of flow and that the centrifugal acceleration is increased (theoretically) about four times.

Assuming the power required to be proportional to the feed pressure, it follows that increase of capacity by raising the pressure would increase the power required per unit of feed (e.g. if the capacity is doubled by raising the pressure from 10 to 40 lb. per sq. inch, four times the power would be required, and the power per unit of feed would be double).

It..../

It will be observed that the nozzle loading in Test 93C, was 4.35 tons per hour per inch, which is considerably higher than the maximum permissible at lower pressures. Since the efficiency of separation does not appear to have been impaired, it is evident that the permissible nozzle loading may be increased by increasing the pressure. Since the pressure determines the rate of flow through a given aperture, it may be concluded that the maximum permissible nozzle loading is a function of the rate of flow through the nozzle.

THE INFLUENCE OF FEED PIPE DIAMETER ON CYCLONE PERFORMANCE.

In order to investigate the influence of the feed pipe diameter on cyclone performance, a series of tests was carried out using feed pipes of 1, $1\frac{1}{2}$ and 2 inches diameter. Each feed pipe was tested at various pressures between the limits of 5 and 40 lb. per sq. inch and all other variables were kept constant for each test. The results of these tests are reported in Table 18 and the relevant Tromp distribution factor curves are shown in Fig 26. The cyclone shown in Fig. 27 was used for the tests on the $1\frac{1}{2}$ inches diameter feed pipe, while the upper portion was replaced by one fitted with both a 1 inch and a 2 inch feed pipe, situated diametrically opposite, for the other tests. When the 1 inch feed pipe was tested, the 2 inch pipe was blanked off with a plug machined to the curvature of the cylindrical section, and vice versa. Since the main supply pipe from the feed pump was 2 inches diameter, suitable sockets were used to connect the cyclone feed pipe to the supply. The distance between the supply pipe and the cyclone inlet was approximately two feet and consisted in each case of a length of pipe of the same diameter as the feed pipe to be tested. The feed pressure was measured in this length of pipe at a point approximately $8\frac{1}{2}$ inches from the cyclone inlet.

TABLE 18 ..../

TABLE 18.

THE INFLUENCE OF FEED PIPE DIAMETER ON CYCLONE PERFORMANCE.

The following conditions were constant for all tests :-

(1) 9½ inches diameter cyclone, 25 degrees apex angle.
(2) 1⅛ inches diameter orifice tube projecting 1¼ inches beneath cover plate.
(3) 1½ inches diameter nozzle.
(4) Suspension specific gravity, 1.30
(5) Feed coal: Landau No.3, -¼" +1 mm.
(6) Pulp ratio, 6 to 1

Test No.	96F	96E	96D	96C	96B	96A	95F	95E	95D	95C	95B	95A	80G	80C	80F	80E	80D	80H
Feed pipe diameter, ins.	1						1½						2					
Pressure, lb/sq.inch	5	7½	10	15	20	40	5	7½	10	15	20	40	5	7½	10	15	20	40
Yield of product, %	27.5	32.1	30.8	18.9	14.0	0.7	29.4	38.4	42.5	42.9	38.1	29.5	32.3	31.9	38.9	37.2	37.5	45.2
Ash of product, %	4.7	5.1	4.8	-	-	4.5	4.9	5.3	5.7	5.6	5.4	5.1	5.1	5.3	5.5	5.4	5.9	5.9
Ash of feed, %	11.8	13.1	13.0	-	-	12.6	12.6	12.9	12.7	12.1	11.2	13.0	13.2	14.4	13.6	13.7	14.3	12.5
Sp.Gr.of separation	1.33	1.335	1.33	1.32	1.315	-	1.325	1.35	1.365	1.365	1.36	1.33	1.33	1.36	1.37	1.365	1.37	1.37
Washing efficiency, %	91	99	98	-	-	-	93	95	95	92	92	90	-	87	90	90	87	90
Capacity, tons/hr.	1.05	1.54	1.73	2.25	2.62	4.05	1.45	1.94	2.40	3.10	3.72	5.48	1.78	2.87	2.83	3.65	4.55	4.43

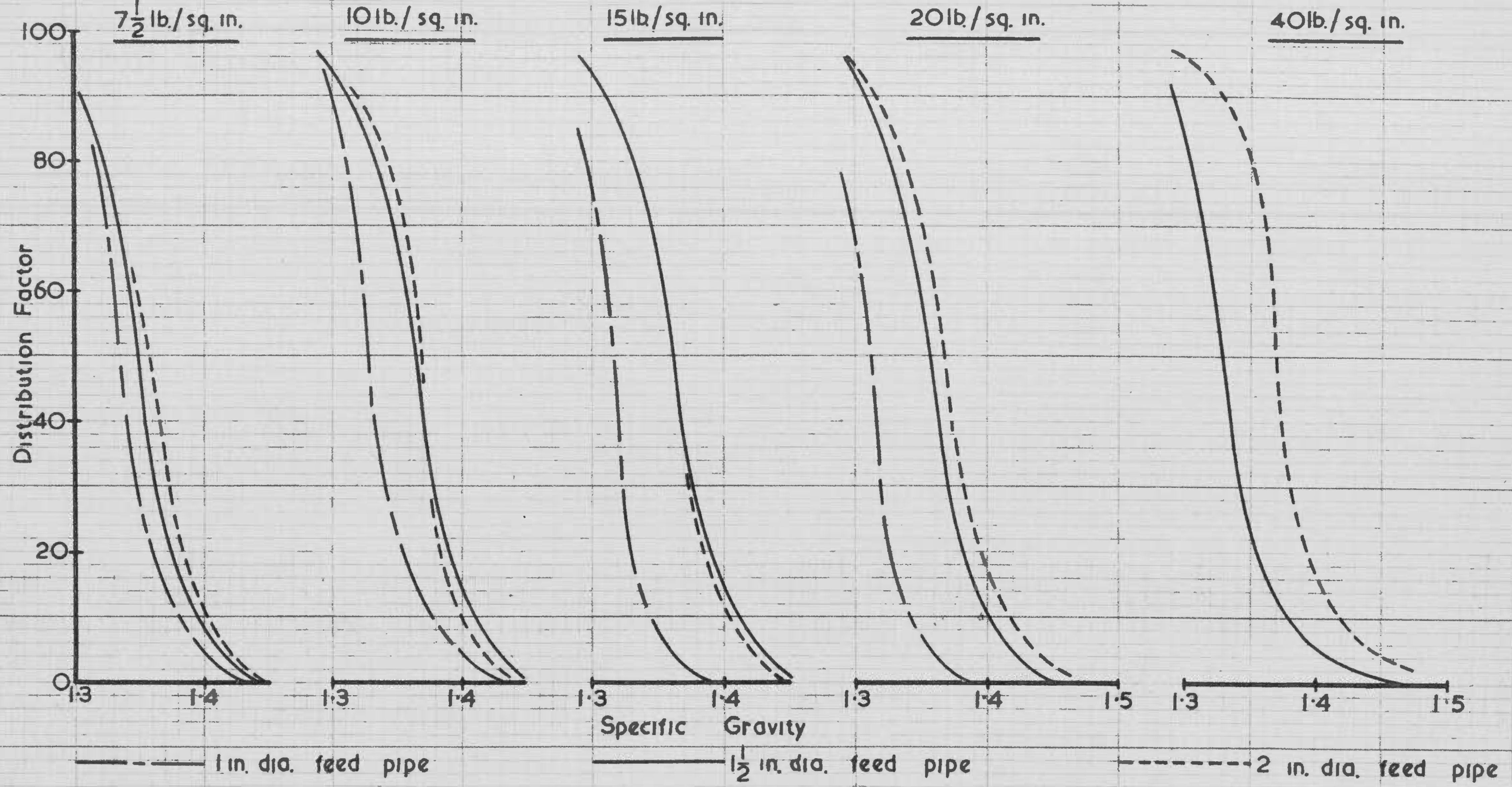

FIGURE 26 Tromp curves relating to Table 18

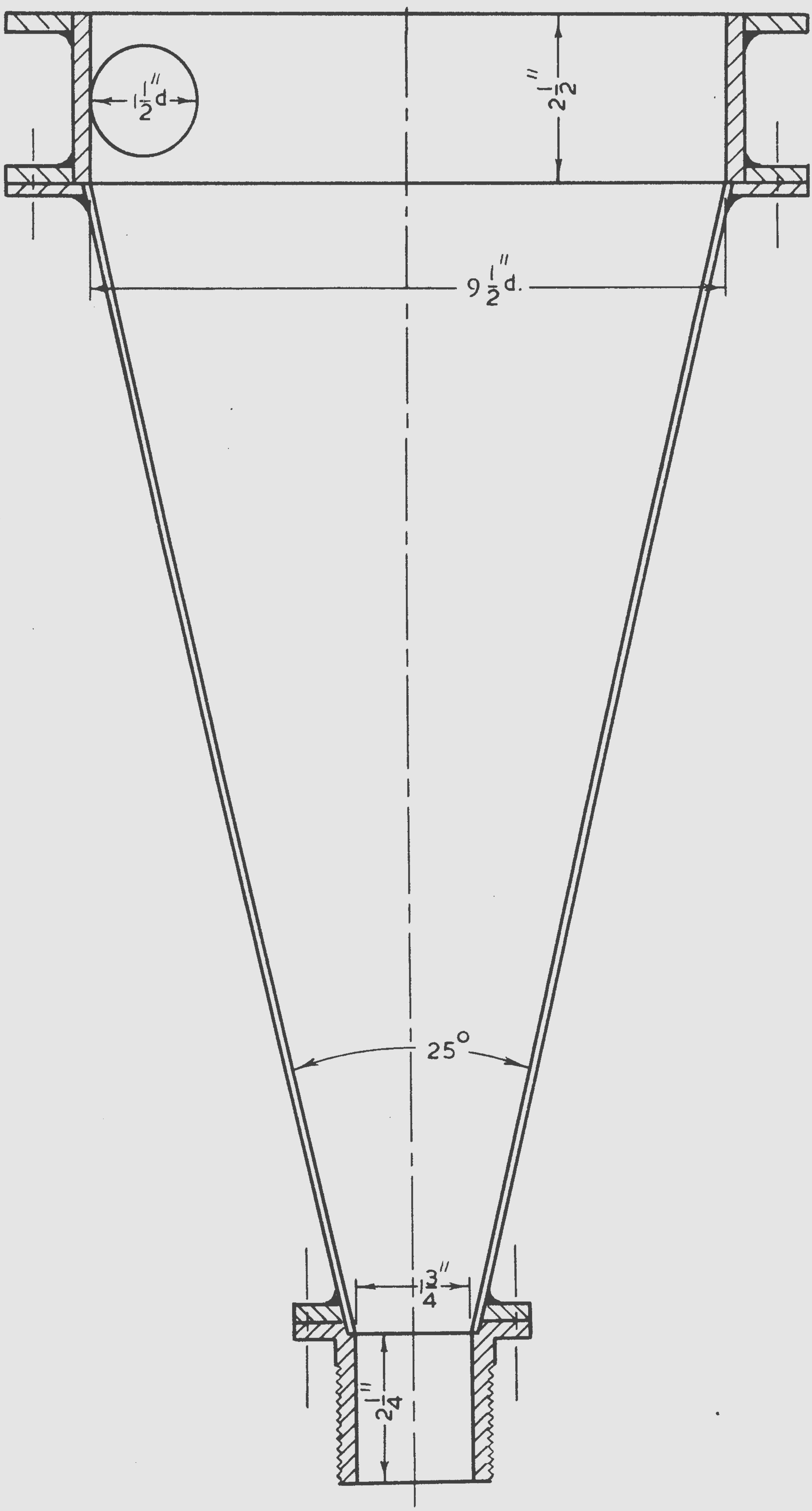

FIGURE 27 25° Cyclone

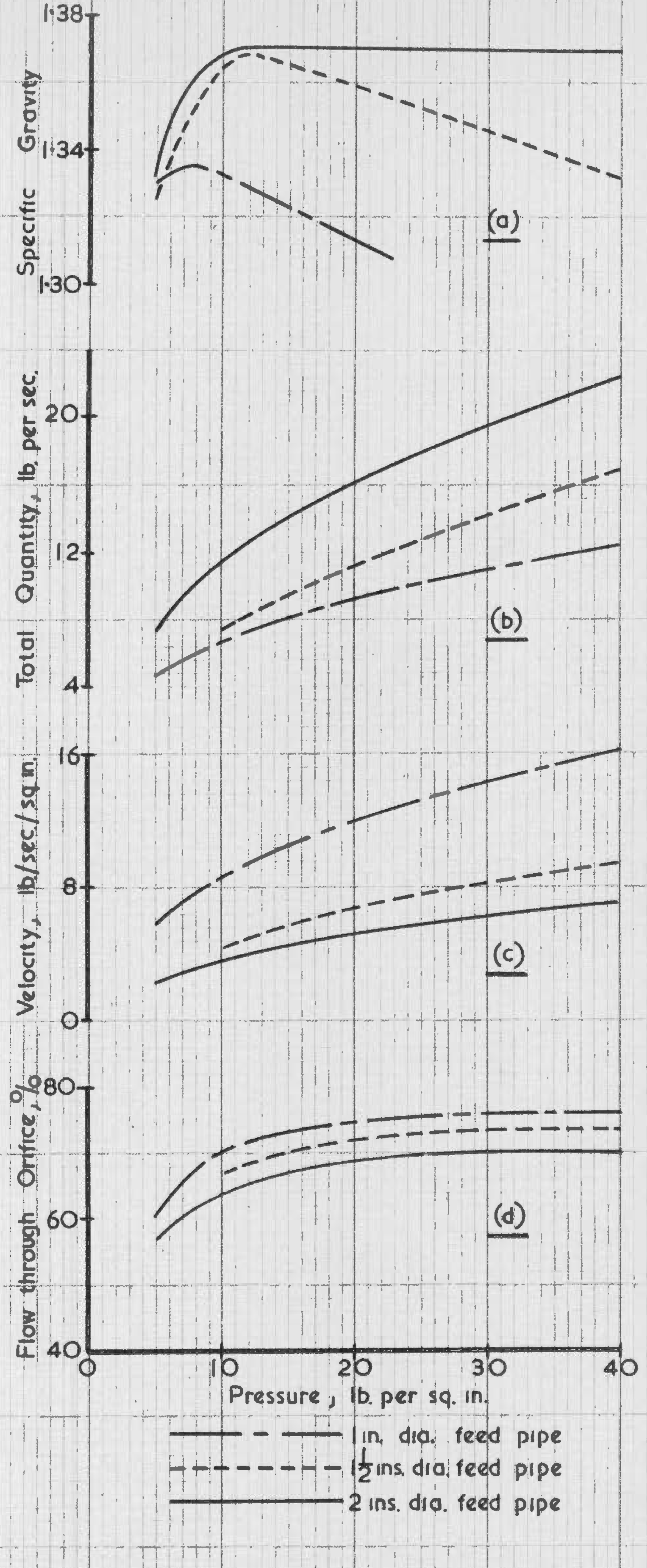

FIGURE 28 Influence of feed pipe diameter on Cyclone flow characteristics

It is evident from the results of these tests (See Table 18), that the diameter of the feed pipe has an important bearing on the specific gravity of separation and the capacity. The influence of feed pipe diameter on specific gravity of separation is readily seen in Fig. 28a, in which the latter has been plotted against feed pressure for each feed pipe. Thus, in the case of the 1 inch diameter feed pipe, the specific gravity of separation rises with increase of pressure from zero, attains a maximum value at about $7\frac{1}{2}$ lb. per sq. inch, and then decreases again comparatively rapidly for further rise in pressure. The curve for the 2 inch diameter feed pipe, on the other hand, also rises as the pressure is increased from zero, but the specific gravity of separation remains substantially constant for pressures between about 10 and 40 lb. per sq. inch. A characteristic more or less intermediate between these two is obtained when using the $1\frac{1}{2}$ inches diameter feed pipe.

It seemed reasonable to suppose that these phenomena could be attributed to changes in the flow characteristics of the cyclone caused by alteration of the feed pipe diameter. In order to obtain a clearer conception of the nature of the flow of fluid through the cyclone under these operating conditions, the tests in Table 18 were repeated using water instead of a coal pulp. The effluents delivered from the orifice and nozzle during a known period of time were collected and weighed. The total rate of flow, feed pipe velocity and the distribution of water between orifice and nozzle were then determined. The results of these tests are reported in Table 19 and curves of rate of flow, feed pipe velocity, etc. are shown in Figs. 28(b), 28(c) and 28(d).

TABLE 19..../

construction has the advantage of relatively low cost and it is also possible to alter the apex angle without the necessity of making a completely new unit. Thus, a conical section having an apex angle of 15 degrees, is all that was required to convert the 25 degree cyclone to a 15 degree cyclone.

The results of a series of comparative tests carried out on the three cyclones are shown in Table 20, and the corresponding Tromp distribution factor curves are shown in Fig. 29. Relatively coarse feed coal was used for these tests.

TABLE 20.

The influence of apex angle on cyclone performance (coarse feed).

The following conditions were constant for all tests:-

(1) $9\frac{1}{2}$ inches diameter cyclone, $1\frac{1}{2}$ inches diameter feed pipe.

(2) Orifice diameter, $1\frac{7}{8}$ inches.

(3) Feed coal:-Landau No. 3 duff, $-\frac{1}{4}$" +1 mm.

(4) Pulp ratio, 6 to 1.

Test No.	17C	17H	18C	95D	84A	88A	88B	90A
Apex Angle, degrees.	38	25	25	25	25	15	15	15
Type of orifice	Disc			Tube projecting $1\frac{1}{4}$" beneath cover plate.				
Specific gravity of suspension.	1.28	1.28	1.28	1.30	1.30	1.30	1.30	1.30
Feed pressure, lb. per sq. inch.	$7\frac{1}{2}$	$7\frac{1}{2}$	$7\frac{1}{2}$	10	10	10	10	10
Nozzle diam. inches.	$1\frac{3}{4}$	$1\frac{3}{4}$	$1\frac{5}{8}$	$1\frac{1}{2}$	$1\frac{3}{4}$	$1\frac{3}{4}$	$1\frac{5}{8}$	$1\frac{1}{2}$
Yield of product, %	29.0	22.4	30.8	42.5	23.0	12.8	20.0	33.0
Ash of product, %	6.2	5.1	5.8	5.7	4.6	3.9	4.1	5.1
Ash of feed, %	15.1	12.8	14.4	12.7	14.1	13.9	13.1	13.0
Sp.Gr. of separation	1.355	1.32	1.348	1.365	1.325	1.3 (estimated)	1.325	1.35
Washing efficiency, %	87	86	88	95	86	-	89	90
Capacity, tons/hour	1.59	-	2.13	2.4	2.96	4.2	3.6	3.0

It will be observed that tests 17C and 17H were carried out under the same operating conditions, with the exception of the apex angle, and that the 25 degree cyclone effected a separation at 1.32 specific gravity as compared with 1.355 in the case

of.../

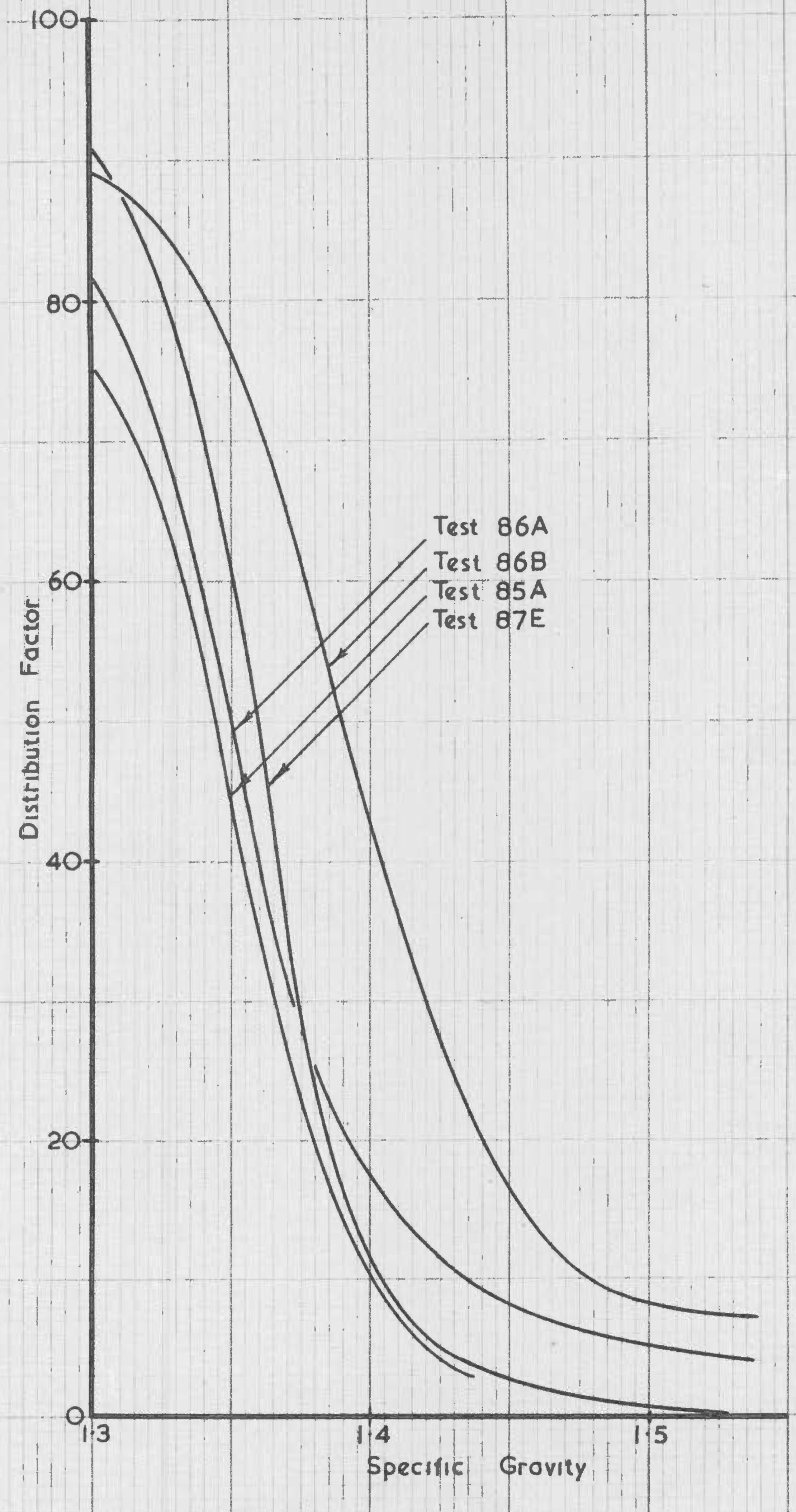

FIGURE 30 Tromp curves relating to Table 21

of the 38 degree cyclone. In test 18C, using the 25 degree cyclone the nozzle diameter was decreased from $1\frac{3}{4}$ to $1\frac{5}{8}$ inches in order to obtain a separation in the region of 1.35 specific gravity. A similar effect will be noted if tests on the 25 degree and 15 degree cyclones carried out under otherwise identical operating conditions, are compared i.e. that a lower specific gravity of separation is obtained when using the smaller apex angle. From this it appears that the thickening effect decreases with decreasing apex angle and consequently that the ratio between the orifice diameter and nozzle diameter must be increased in order to maintain a constant specific gravity of separation. Since the orifice diameter can not be greater than about $1\frac{7}{8}$ inches, it follows that the nozzle diameter will have to be reduced as the apex angle is decreased. Alternatively, a higher suspension specific gravity could be used to maintain the required separation without altering the nozzle diameter.

The Tromp distribution factor curves for these tests (Fig.29) do not differ appreciably in shape, thus indicating that the apex angle has little influence on the efficiency of separation of $-\frac{1}{4}$ inch +1 mm feed. In order to determine whether the apex angle would affect the efficiency of separation of smaller feed particles to a greater extent, the tests reported in Table 21 were carried out. The Tromp distribution factor curves relating to these tests are shown in Fig. 30.

TABLE 21..../

TABLE 21.

The influence of apex angle on cyclone performance (fine feed).

The following conditions were constant for all tests.

(1) $9\frac{1}{2}$ inches diameter cyclone, $1\frac{1}{2}$ inches diameter feed pipe.

(2) $1\frac{7}{8}$ inches diameter orifice tube projecting $1\frac{1}{4}$ inches beneath cover plate.

(3) Feed coal:- Landau No. 3 duff, -1 mm +44 mesh.

(4) Pulp ratio, 6 to 1.

(5) Suspension specific gravity, 1.30.

(6) Feed pressure, 10 lb. per sq. inch.

Test No.	86A	86B	85A	85E	87A	87E
Apex angle, degrees	38	38	25	25	15	15
Nozzle diameter, inches	$1\frac{3}{4}$	$1\frac{5}{8}$	$1\frac{3}{4}$	$1\frac{5}{8}$	$1\frac{3}{4}$	$1\frac{1}{2}$
Yield of product, %	34.7	47.7	31.6	48.4	10.2	40.1
Ash of product, %	5.5	5.9	4.8	5.4	3.8	4.4
Ash of feed, %	13.5	13.6	12.5	12.7	13.6	12.9
Specific gravity of separation.	1.35	1.39	1.345	1.395	1.3 (estimated)	1.36
Washing efficiency, %	83	84	80	90	-	91
Capacity, tons/hour.	1.48	1.52	2.25	1.85	1.85	1.85

These tests again show that the thickening effect is reduced when the 15 degree cyclone is used, but there appears to be little difference in this respect between the 38 and 25 degree cyclones. This discrepancy is ascribed to a difference in the specific gravity of suspension. For convenience, the feed coal was wet screened on a 44 mesh sieve and was then drained and dried in the sun prior to the cyclone tests. The feed was air dry in the case of the tests on the 15 and 25 degree cyclones, but was still visibly wet when used for the test on the 38 degree cyclone. As it was raining at the time, this test would have had to be postponed for some days in order to dry the feed still further. Since the investigation was primarily concerned with the influence of apex angle on efficiency (the influence on specific gravity of separation having already been established) it was considered that

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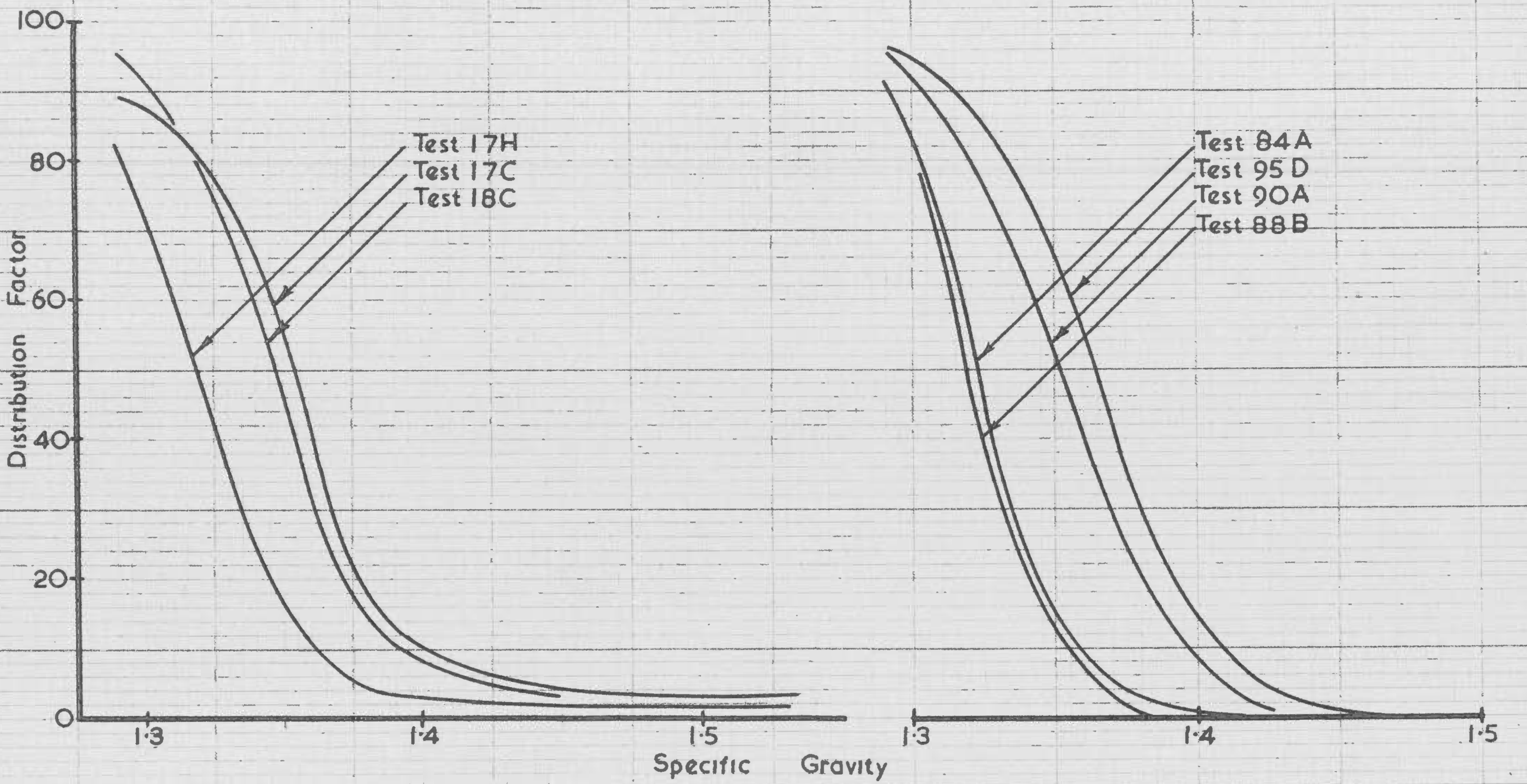

FIGURE 29 Tromp curves relating to Table 20

the diluting effect of damp feed (about 10% moisture) would be of little consequence.

Comparing the Tromp distribution factor curves shown in Fig. 30, and the results in Table 21, it will be clear that a decrease in the apex angle improves the separation of small particles, particularly in the case of the 15 degree cyclone. It seems reasonable to conclude, then, that the apex angle becomes of greater importance as the percentage of fines (say, minus 1 mm) in the feed increases. Since decrease of the apex angle increases the volume of the cyclone, and consequently the time taken for a particle to pass through the washer, all other dimensions being constant, it appears that the improved separating efficiency of small particles could be attributed to an increase of the period of pulp retention. This possibility is discussed later.

Now, Tables 20 and 21 show that there is a tendency for the capacity of the cyclone to increase as the apex angle is decreased. Neglecting small variations in the pulp ratio, it follows that this increase in the capacity must be due to an increase in the rate of flow of pulp. Since this factor is obscured to a certain extent by differences in the pulp ratio due to settling in the feed tank, several tests were carried out using water only, in order to determine the relationship between the rate of flow and the apex angle. The results of these tests are reported in Table 22.

TABLE 22..../

TABLE 22

The Influence of Apex Angle on the Flow Characteristics of the Cyclone.

The following conditions were constant for all these tests:-

(1) $9\frac{1}{2}$ inches diameter cyclone, $1\frac{1}{2}$ inches diameter feed pipe, apex angle as indicated.

(2) $1\frac{7}{8}$ inches diameter orifice tube projecting $1\frac{1}{4}$ inches beneath cover plate.

(3) Feed pressure, 10 lb. per sq. inch.

Nozzle diameter inches.	Total quantity of feed passing lb.per sec.			Flow through Orifice. percentage of total		
	38°	25°	15°	38°	25°	15°
$1\frac{3}{4}$	6.5	8.1	11.1	55.5	50.6	45.1
$1\frac{5}{8}$	6.7	7.3	9.0	65.7	60.3	56.5
$1\frac{1}{2}$	6.9	7.3	8.8	72.1	67.0	64.7
$1\frac{3}{8}$	6.5	7.2	8.6	81.1	74.9	74.1
$1\frac{1}{4}$	6.7	7.2	8.8	89.3	82.1	81.2

The results of these tests prove conclusively that the rate of flow increases as the apex angle is decreased, the applied pressure being constant. In other words, the resistance to flow decreases with decreasing apex angle.

This may be rather surprising in view of the fact that the wetted surface of the cyclone is increased when the apex angle is decreased. One would, therefore, expect the resistance to increase, if the friction between the fluid and the walls is the only criterion. Since the overall resistance is actually lower, however, one may conclude that the internal fluid resistance is decreased to a greater extent than the wall resistance is increased. This appears reasonable, as one may expect that the relative velocity between adjacent layers of fluid would be lower the smaller the apex angle. Thus, if the apex angle were zero (i.e. cyclone consisting of a cylinder) there would be little difference between the tangential velocities of adjacent layers and fluid friction losses would be small. In a cone, however, the

tangential.....

tangential velocity increases with decreasing radius and consequently the fluid friction losses would depend on the rate of decrease of diameter (i.e. on the apex angle).

Table 22 also shows that effluent through the orifice, expressed as a percentage of the total water passing through the cyclone, increases with increasing apex angle, all other variables being constant. Now, it will be remembered that an increase in this percentage is associated with increase in the specific gravity of separation i.e. the percentage passing through the orifice is increased by decreasing the nozzle diameter with consequent increase in the specific gravity of separation (see Tables 5 and 6). This difference in the percentage flow through the orifice thus appears to substantiate the conclusion drawn from Table 20 that the "thickening effect" increases with increasing apex angle.

In view of the increased rate of flow associated with a decrease in the apex angle, it is necessary to analyse to what extent the period of pulp retention is affected by the apex angle.

It appears reasonable to assume that the period of pulp retention is :-

(a) proportional to the volume of the cyclone - and

(b) inversely proportional to the rate of flow of fluid.

Neglecting other factors, the relationship between these variables could be expressed as -

$$t = \frac{K V}{Q}$$

where t = period of pulp retention
K = a constant
V = volume of the cyclone
Q = rate of flow.

The volumes of the three cyclones were estimated, allowing for the air space in the centre, and the following values were obtained:-

38 degree cyclone	424 cubic inches.
25 degree cyclone	550 cubic inches.
15 degree cyclone	816 cubic inches,

From..../

into four size fractions, as indicated, and each size fraction was separately subjected to float and sink analysis. From these data, the Tromp distribution factor curves shown in Fig. 31 were obtained. Details of the screen analyses and float and sink analyses are given in appendices 2 and 5.

The ash contents of the size fractions of the product and tailing were also determined and are given in Table 24 together with the relevant washing efficiencies, etc.

TABLE 23.

Washing Test on Minus half inch plus zero Feed.

Test No.	93C.
Cyclone.	$9\frac{1}{2}$ inches diameter, 15 degrees apex angle, $1\frac{1}{2}$ inches diameter feed pipe.
Orifice.	$1\frac{7}{8}$ inches diameter tube projecting $1\frac{1}{4}$ inches beneath cover plate.
Nozzle.	$1\frac{1}{2}$ inches diameter.
Feed coal.	Coronation duff screened $-\frac{1}{2}$" +0 (plus 100 mesh recovered).
Suspension S.G.	1.32
Pulp ratio	3 to 1
Feed pressure	40 lb. per sq. inch.
Yield of Product	31.4 %
Ash of Product	5.4 %
Ash of feed.	12.1 %
Sp.Gr. of separation	1.36
Washing efficiency	>100%
Capacity	9.5 tons per hour.

TABLE 24..../

tangential velocity increases with decreasing radius and consequently the fluid friction losses would depend on the rate of decrease of diameter (i.e. on the apex angle).

Table 22 also shows that effluent through the orifice, expressed as a percentage of the total water passing through the cyclone, increases with increasing apex angle, all other variables being constant. Now, it will be remembered that an increase in this percentage is associated with increase in the specific gravity of separation i.e. the percentage passing through the orifice is increased by decreasing the nozzle diameter with consequent increase in the specific gravity of separation (see Tables 5 and 6). This difference in the percentage flow through the orifice thus appears to substantiate the conclusion drawn from Table 20 that the "thickening effect" increases with increasing apex angle.

In view of the increased rate of flow associated with a decrease in the apex angle, it is necessary to analyse to what extent the period of pulp retention is affected by the apex angle.

It appears reasonable to assume that the period of pulp retention is :-

(a) proportional to the volume of the cyclone - and

(b) inversely proportional to the rate of flow of fluid.

Neglecting other factors, the relationship between these variables could be expressed as -

$$t = \frac{K\,V}{Q}$$

where t = period of pulp retention
K = a constant
V = volume of the cyclone
Q = rate of flow.

The volumes of the three cyclones were estimated, allowing for the air space in the centre, and the following values were obtained:-

38 degree cyclone	424 cubic inches.
25 degree cyclone	550 cubic inches.
15 degree cyclone	816 cubic inches,

From..../

From Table 22, the mean rate of flow for the 38, 25 and 15 degree cyclones is 6.7, 7.4 and 9.3 lb. per second respectively.

Using the above values of the volume and flow rate, the relative periods of pulp retention were estimated for the three cyclones. It is found that the relative periods are as follows :-

38 degree cyclone	1
25 degree cyclone	1.18
15 degree cyclone	1.39

These figures indicate, then, that the period of pulp retention increases with decreasing apex angle, notwithstanding the increased rate of flow. Thus, in the case of the 15 degree cyclone the period of pulp retention is increased by some 39 per cent and the rate of flow is also increased by about 39 per cent compared with the corresponding values for the 38 degree cyclone. Since the higher rate of flow would give rise to greater centrifugal forces acting on the particles, thus decreasing the time required for separation, the nett result should be a substantial improvement in the efficiency of separation of small particles.

CYCLONE PERFORMANCE WHEN TREATING VARIOUS SIZES OF COAL.

As would be expected, the efficiency of separation decreases with decreasing feed particle size. This is illustrated by the test shown in Table 23. This test was carried out under conditions best suited to the separation of small particles viz. small apex angle and high feed pressure. The results of this test are, therefore, a fair indication of the efficiency which may be expected in practice for various size grades.

The raw duff was screened at ½ inch before washing in the cyclone and all fines were included in the feed, but only plus 100 mesh material was recovered after washing in the cyclone. The product and tailing obtained were then screened

into.../

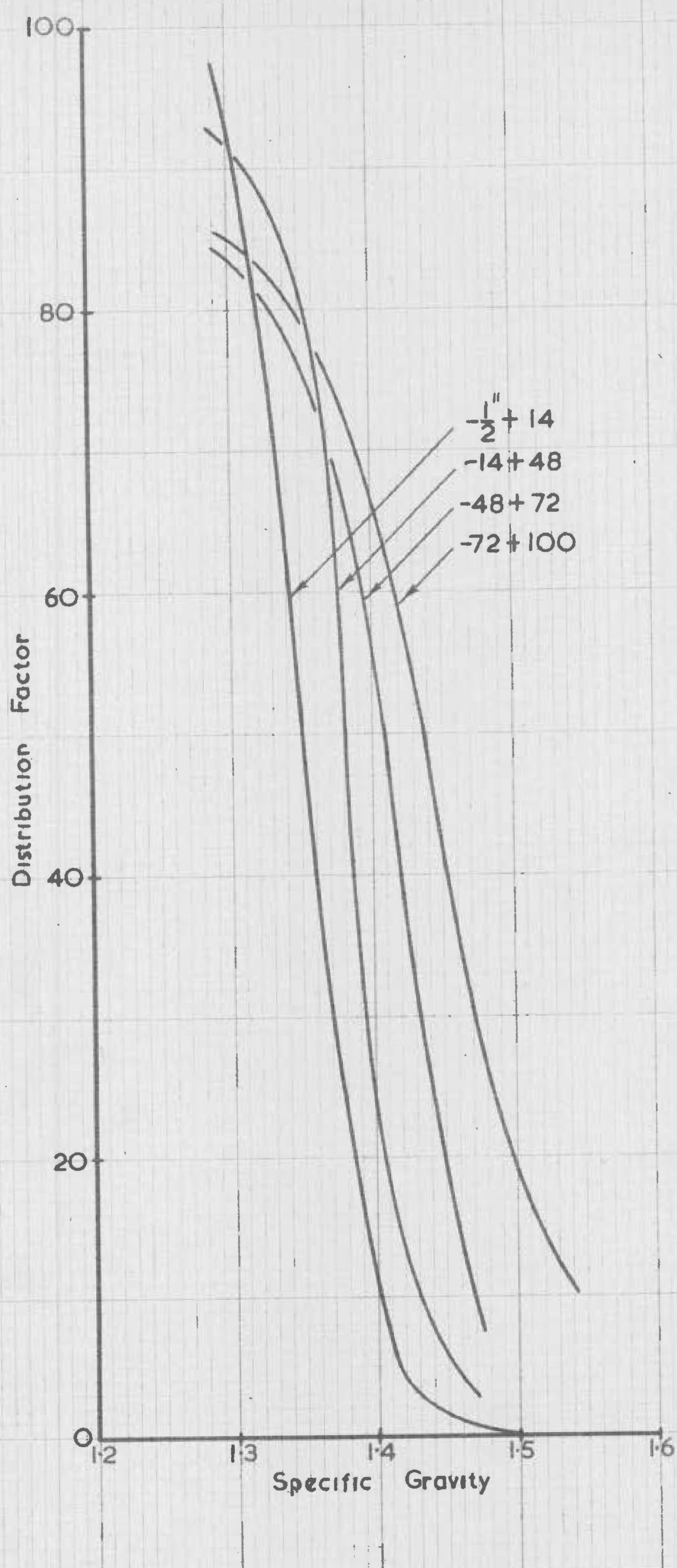

FIGURE 31 Tromp curves of size fractions of Test 93C

TABLE 24.

CYCLONE PERFORMANCE ON THE SIZE FRACTIONS OF TEST 93C.

Screen Fraction.	$-\frac{1}{2}$"+100#	$-\frac{1}{2}$"+14#	-14# +48#	-48# +72#	-72# +100#
Ash of product, %	5.4	5.5*	5.1	6.1	6.3
Ash of feed, %	12.1	11.5*	12.2	15.4	15.4
Sp.Gr.of separation	1.36	1.35	1.38	1.41	1.44
Washing efficiency, %	>100	>100	94	85	80
Mean diameter, inches	-	0.153	0.029	0.01	0.007

* Calculated from float and sink analysis of the $-\frac{1}{2}$"+14 mesh fraction.

The Tromp distribution factor curves for this test show that the cyclone is capable of very sharp separation down to about 48 mesh. Although the efficiency falls off for particles below this size, the separations on material down to 100 mesh are reasonably exact and are probably superior to those obtained with jigs, classifiers etc., when washing relatively coarse coal. The efficiency of separation of the smaller particles will depend, however, on the heavy medium used. A barytes suspension of 1.35 to 1.4 specific gravity has comparatively low viscosity, while a sand suspension of similar density for example, has higher viscosity, and lower efficiencies must consequently be expected under the same operating conditions. The influence of viscosity on efficiency is discussed in greater detail in Appendix 1.

It will be seen from Table 24 that the specific gravity of separation is not constant for all size fractions, but increases with decreasing particle size. This phenomenon is quite usual in processes using heavy media and is due to the fact that the effective density of a suspension with respect to any given particle depends on the relative sizes of the coal particle and the medium particles. According to Hirst,(18) this relationship is given by the expression:-

5 ...../

$$\rho' = \rho + (s - \rho)\left(\frac{d}{D}\right)^{1-\phi}$$

where ρ' = effective density of the suspension with respect to a particle of diameter D.

ρ = density of the suspension.

s = density of the heavy medium particles.

ϕ = -1 for heavy medium particles obeying Stoke's law.

-0.5 for heavy medium particles obeying Allen's law.

0 for heavy medium particles obeying Newton's law.

In order to obtain a clearer conception of the relationship between the specific gravity of separation and particle size for test 93C, specific gravity of separation was plotted against the mean diameter of the corresponding size fraction of the feed as shown in Fig. 32. The mean diameter of the fraction, $-\frac{1}{2}$" +14 mesh, was calculated from the formula shown on page 58 while the mean diameter for the remaining fractions was taken as the arithmetic mean between the sieve apertures. Although this procedure is not accurate, it will serve the present purpose.

It will be seen from Fig. 32 that the specific gravity of separation appears to be reasonably constant for particles down to about 0.08 inch and then commences to rise fairly rapidly for smaller particles. Naturally, this conclusion only applies to the barytes suspension used. For example, if coarser medium particles are used, the specific gravity of separation would commence to rise at a larger coal size than 0.08 inch. The effective specific gravity is also influenced by the specific gravity of the medium as shown by Hirst's expression.

It will be observed that the relationship obtained closely resembles that proposed by Hirst. Unfortunately, as the mean size of the barytes particles is not known, it is not possible to determine how closely the results obtained agree with Hirst's expression.

SUMMARY..../

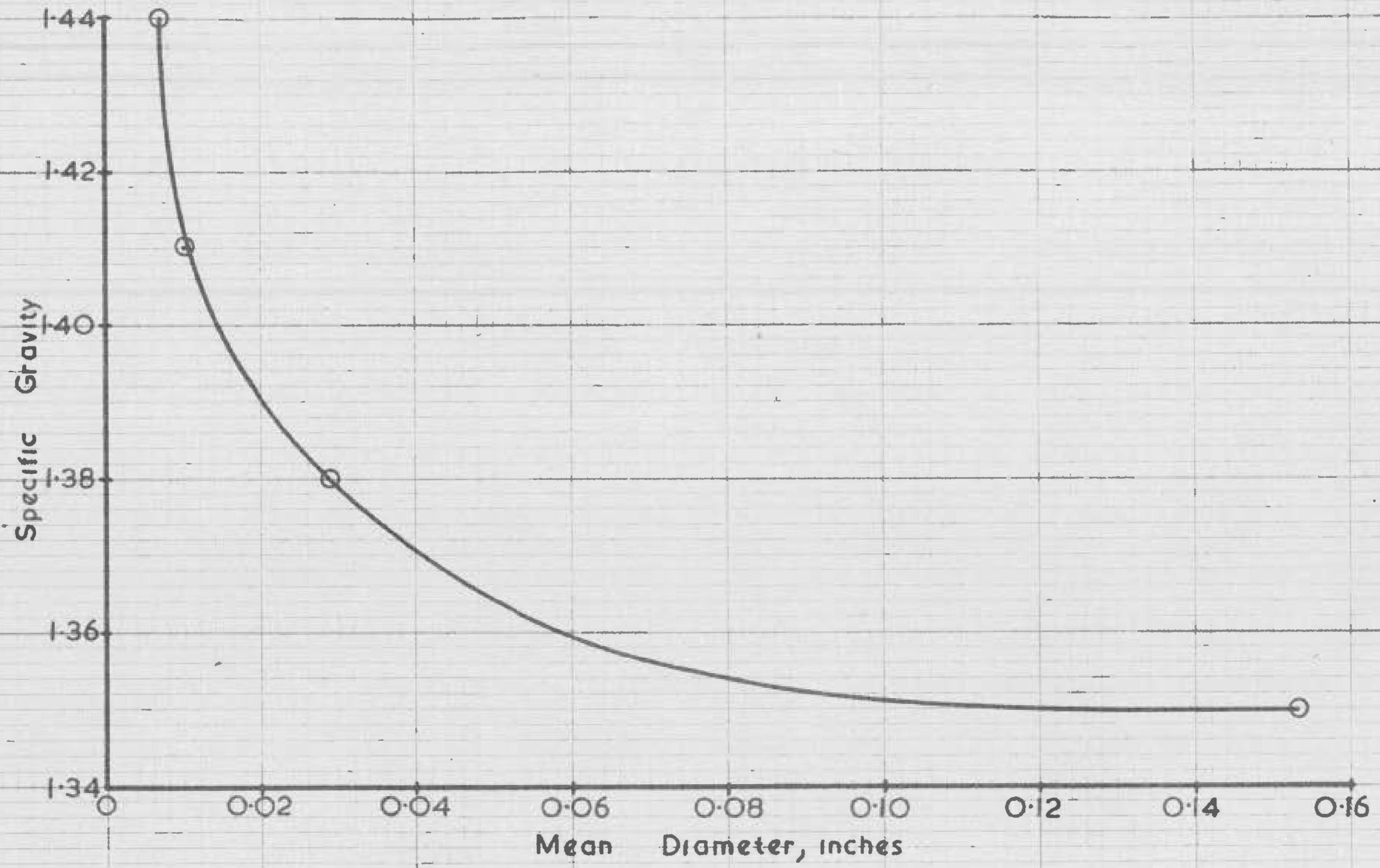

FIGURE 32 Variation of Specific Gravity of Separation with particle diameter

SUMMARY OF CONCLUSIONS AND THE SELECTION OF OPERATING CONDITIONS SUITABLE FOR WASHING WITBANK DUFF COAL AT 1.35 SPECIFIC GRAVITY.

Summary of Conclusions:

As already explained, the investigation of the main operating characteristics of the cyclone washer was primarily undertaken in order to obtain sufficient data to enable Witbank duff coals to be washed at about 1.35 specific gravity. Since the results of these final washing tests on Witbank duff were to be used to assess the suitability of this process for extraction of the coking fraction on a commercial scale, maximum efficiency and capacity were aimed at, these being the principal technical criteria in the final economic analysis. Before discussing the selection of the optimum operating conditions for the desired separations, it may be of assistance to summarise relevant conclusions drawn from the various tests described. For convenience, each variable will be discussed on the assumption that all other variables are constant except those specifically mentioned.

The Relationship Between Orifice and Nozzle and Other Factors Influencing the Specific Gravity of Separation.

The relationship between the orifice and nozzle diameters determines the specific gravity of separation and, in general, the greater the orifice diameter in relation to the nozzle diameter the higher is the specific gravity of separation. In other words, the specific gravity of separation may be increased either by increasing the orifice diameter or by decreasing the nozzle diameter, the other aperture being kept constant. The relationship between orifice and nozzle required for any specified separation depends on the specific gravity of suspension used, the lower the latter, the greater is the ratio between the orifice and nozzle diameters.

The specific gravity of separation decreases with decreasing apex angle and feed pipe diameter. When small feed pipes (1 to $1\frac{1}{2}$ inches) are used, specific gravity of separation

increases..../

increases with rise in pressure, attains a maximum value in the region of $7\frac{1}{2}$ to 10 lb. per square inch and then decreases again for further rise in pressure.

When the mean size of the tailing exceeds about 1/10 the nozzle diameter and the percentage material in the tailing coarser than 1/5 the nozzle diameter is greater than 1 or 2 per cent, the specific gravity of separation increases. In order to prevent bridging of the nozzle, no particles should exceed 1/3 of the nozzle diameter.

When using the barytes suspension, the specific gravity of separation remains constant for particles down to about 0.08 inch but increases for smaller particles.

Factors Influencing the Feed Capacity.

The throughput, or feed capacity of the cyclone is determined by the rate of flow of pulp and the percentage of coal in the pulp.

The rate of flow of pulp is affected by various factors as follows :-

(a) The use of a small orifice and its appropriate nozzle decreases the rate of flow.

(b) The smaller the apex angle the lower is the internal resistance and the greater the rate of flow for a constant applied pressure.

(c) Increase of pressure increases the rate of flow.

(d) The larger the feed pipe diameter (within the limits investigated), the greater is the rate of flow.

The maximum percentage of coal permissible in the pulp is determined by the diameter of the orifice or nozzle, the specific gravity of separation and the specific gravity composition of the feed. This is best explained by examples. Consider first a separation in which the yield of product is to be 30 per cent i.e. 70 per cent of the feed must pass through the nozzle. Since the nozzle is invariably smaller than the orifice,

it.../

it follows that the quantity of coal which can just pass through the nozzle in unit time, without overloading it, will determine the percentage of coal permissible in the pulp i.e. the smaller the diameter of the nozzle the smaller the quantity of tailing which can be discharged in unit time. If, on the other hand, a separation is to be effected at a specific gravity where the yield of product is, say, 90 per cent, the nozzle could be small relative to the orifice without becoming overloaded, since the rate of discharge of refuse will be small compared to the rate of discharge of product through the orifice and the diameter of the orifice would determine the pulp ratio. In other words, if the percentage yield is low the diameter of the nozzle determines the optimum pulp ratio and if the percentage yield is high the diameter of the orifice determines the pulp ratio (provided that the nozzle is not excessively small).

Factors Influencing the Efficiency of Separation.

If the nozzle (or orifice) becomes overloaded the efficiency is impaired.

The use of orifices larger than about $1\frac{7}{8}$ inches diameter appears to result in loss of efficiency. Smaller orifices increase the period of pulp retention and would, therefore, be advantageous if fine material is to be treated.

Decrease of the apex angle increases the period of pulp retention and consequently the efficiency of separation of small particles.

High feed pressures also increase the efficiency of separation of small particles.

When a disc-type orifice is used, a percentage of the feed by-passes the cyclone and impairs the efficiency. This can be prevented by replacing the disc-type orifice by a tube projecting $1\frac{1}{4}$ to $2\frac{1}{2}$ inches beneath the cover plate.

Under..../

Under optimum operating conditions, the efficiency of separation is constant for particles down to about 48 mesh in size. The efficiency falls off for particles below this size, but the separation is still remarkably sharp for particles down to 100 mesh.

The Selection of Optimum Operating Conditions for Washing Witbank Duff at 1.35 to 1.4 Specific Gravity.

It will be clear from the preceeding summary of conclusions that the percentage yield of product required and the size grading of the feed are the main criteria when selecting operating conditions to give the maximum possible efficiency and capacity. In the case of Witbank duff coals, the yield at 1.35 specific gravity is about 30% on the average, and the percentage of material finer than 1 mm is of the order of 20 to 25 per cent. The top size of the duff ranges from ¼ to ½ inch.

Since the percentage of fines is high, particular attention must be paid to the separation of small particles. This requires that -

(a) The feed pressure should be high and

(b) The apex angle should be small.

Both these requirements are also an advantage for large capacity.

In view of the small yield, the diameter of the nozzle determines the pulp ratio and it should, therefore, be as large as possible, and a relatively high suspension specific gravity will be required. A large feed pipe will also be necessary, as the feed pressure will be high and maximum thickening effect is required. A large feed pipe also increases the capacity.

Since the top size of the feed is likely to be ½ inch, the nozzle, orifice and feed pipe diameters should be at least 1½ inches. The grading of the feed is also such that a 1½ inches diameter nozzle would permit flow without disturbance.

Interpreting../

Interpreting these requirements in terms of the facilities available in the laboratory, the following operating conditions would be selected for these separations.

(1) $9\frac{1}{2}$ inches diameter cyclone.

(2) 15 degrees apex angle.

(3) Orifice tube having $1\frac{7}{8}$ inches diameter and projecting at least $1\frac{1}{4}$ inches beneath cover plate.

(4) Suspension specific gravity of 1.3 or greater, thus requiring a nozzle of $1\frac{1}{2}$ to $1\frac{5}{8}$ inches diameter (the maximum nozzle is $1\frac{3}{4}$ inches and if $1\frac{5}{8}$ inches is aimed at, there is still a measure of control should the suspension specific gravity selected be too high).

(5) Pulp ratio of 3 to 1, which has been proved to be satisfactory for a nozzle of $1\frac{1}{2}$ inches diameter.

(6) Feed pressure of 40 lb. per sq. inch, i.e. the maximum attainable with the present set up.

(7) Feed pipe of $1\frac{1}{2}$ or 2 inches diameter, preferably the latter.

(8) The barytes used previously appears satisfactory as regards size and viscosity.

WASHING TESTS ON TYPICAL WITBANK DUFF COALS.

Having determined the optimum operating adjustments for separations in the region of 1.35 to 1.4 specific gravity, it was possible to carry out washing tests on typical Witbank duff coals, in order to assess the suitability of the cyclone washer for the recovery of the coking fraction.

For this purpose, approximately 15 tons of duff were obtained from each of three collieries situated in the Witbank area. Since the investigation was primarily concerned with the general washing aspect, no effort was made to obtain samples which would be truly representative of each colliery's normal output of duff coal.

All..../

All samples were screened at $\frac{1}{2}$ inch in order to remove the small percentage of large particles present in the raw coal. The presence of this material is due to inefficient screening at the colliery and it is not a normal constituent of duff. Thereafter, each batch of coal was sampled and analysed as shown in Table 25.

TABLE 25.

ANALYSIS OF THE RAW DUFFS.

Colliery.		Coronation	Landau No. 3.	Springbok
Ash %		12.4	14.3	11.8
Moisture %		3.8	1.8	2.6
Volatile Matter %		-	24.5	27.2
Swelling No.		1Ag	1Ag	F
Calorific Value lb.per lb.		12.4	12.5	12.7
Float & Sink Analysis	(Yield % F.1.35	17.7	31.8	28.8
	(Sw.No. F.1.35	3	$5\frac{1}{2}$	$3\frac{1}{2}$ to 4
	(Yield % F.1.4	49.9	50.0	53.1
	(Sw.No. F.1.4	$1-1\frac{1}{2}$Ag	4	2
Screen Analysis*	($\frac{1}{2}$" +14 mesh	74.5	69.5	82.2
	(-14 +48 mesh	16.8	21.1	11.8
	(-48 +72 mesh	3.1	4.1	1.3
	(-72 +100 mesh	1.1	2.0	1.5
	(-100 mesh	4.5	3.3	3.2

* 14 and 48 mesh, Tyler sieves.
72 and 100 mesh, B.S.sieves.

Coals having a swelling index lower than 3 are usually not regarded as coking coals. On this basis, therefore, it is clear that the duff coals shown in Table 25 are unsuitable for coking purposes in the raw state. However, it will be noted that a coking fraction may be obtained in all three cases by effecting a separation at 1.35 specific gravity. Recovery of the coking fraction from this particular sample of Coronation duff is clearly not an economic proposition as the yield is extremely low and, in addition, it appears that a theoretical separation would be required in order to obtain a product of acceptable quality. The Landau No. 3 duff, on the other hand, contains a comparatively large proportion of material having well developed swelling properties and it is thus theoretically possible to obtain a

satisfactory..../

satisfactory coking product by effecting a separation at 1.4 specific gravity.

Washing Tests.

A study of the washability characteristics of these duffs showed that the most formidable separations would be encountered at specific gravities between 1.35 and 1.4. It was decided, therefore, to confine the cyclone washing tests to separations in this region and not to endeavour to wash each coal at the optimum specific gravity from a coking point of view. In this way, it was thought, the cyclone would be subjected to the most rigorous trial possible and, if it was found to be satisfactory, it could then be recommended for less difficult separations without conducting further tests.

One ton batches of the raw duffs (screened at ½ inch) were accordingly washed in the cyclone, using the optimum operating conditions for the required separations. The products and tailings obtained were recovered on 100 mesh screens, minus 100 mesh material being neglected as it constitutes only a small proportion of the raw coal (see Table 25). The recovered products and tailings were dried in the sun, weighed, and sampled in accordance with standard practice. One sample of each was crushed and analysed, while another sample was screened into four size fractions. Each size fraction was then separately subjected to float and sink analysis, and the specific gravity composition of the original sample was calculated from these data.

The results of the washing tests are reported in Table 26, and the Tromp distribution factor curves for the minus ½ inch plus 100 mesh material in the feed are shown in Fig.33. Details of the float and sink and screen analyses of the samples are given in Appendices 2 and 5 respectively. The Tromp distribution factor curves for the various size fractions of tests 99B and 100C are shown in Fig. 34. The shape of the relevant distribution factor curves for the remainder of the tests in Table 26,

do.../

do not differ appreciably from those shown in Fig. 34 and have been omitted in order to simlify this diagram (see Appendix 4)

TABLE 26.

CYCLONE WASHING TESTS ON TYPICAL WITBANK DUFF COALS.

The following conditions were constant for these tests :-

(1) $9\frac{1}{2}$ inches diameter cyclone, 15° apex angle.
(2) Orifice tube diameter $1\frac{7}{8}$ inches.
(3) Nozzle diameter $1\frac{1}{2}$ inches.
(4) Feed coal screened $-\frac{1}{2}$" +0, plus 100 mesh material recovered.
(5) Pulp ratio 3 to 1

Colliery	Coronation	Landau No. 3.		Springbok.	
Test No.	93C	99B	100C	101A	102B
Feed pipe diameter, inches	$1\frac{1}{2}$	$1\frac{1}{2}$	$1\frac{1}{2}$	2	2
Orifice tube length, inches	$1\frac{1}{4}$	$2\frac{1}{2}$	$2\frac{1}{2}$	$2\frac{1}{2}$	$2\frac{1}{2}$
Suspension specific gravity	1.32	1.32	1.35	1.3	1.35
Feed pressure, lb/sq.inch	40	40	40	35	35
Yield of Product, %	31.4	36.0	47.7	35.7	53.5
Ash content of product, %	5.4	5.6	6.1	5.8	6.8
Moisture content of Product %	2.9	1.9	2.0	2.5	2.7
Swelling No. of product.	2	$5\frac{1}{2}$	4	$3\frac{1}{2}$	$1\frac{1}{2}$
Ash content of feed	12.1	14.3	14.0	12.5	11.9
Specific gravity of separation	1.36	1.360	1.385	1.365	1.405
Theoretical yield of product %	26.0	37.0	47.5	35.5	54
Ash content of theoretical product %	4.8	5.1	5.8	5.3	6.5
$\pm$ 0.10 Sp.Gr. distribution	82	75	78	80	77
Efficiency %	>100	92	96	93	94
Capacity, tons per hr.of feed.	9.5	11.7	10.1	12.5	11.8

Comparing the values of the yield and the ash content of the actual products obtained with the corresponding theoretical values (obtained from the washability characteristics in Appendix 3), it will be observed that the agreement is very close in the case of all tests except 93C. This close agreement is all the more remarkable in view of the extremely high $\pm$0.10 specific

gravity.../

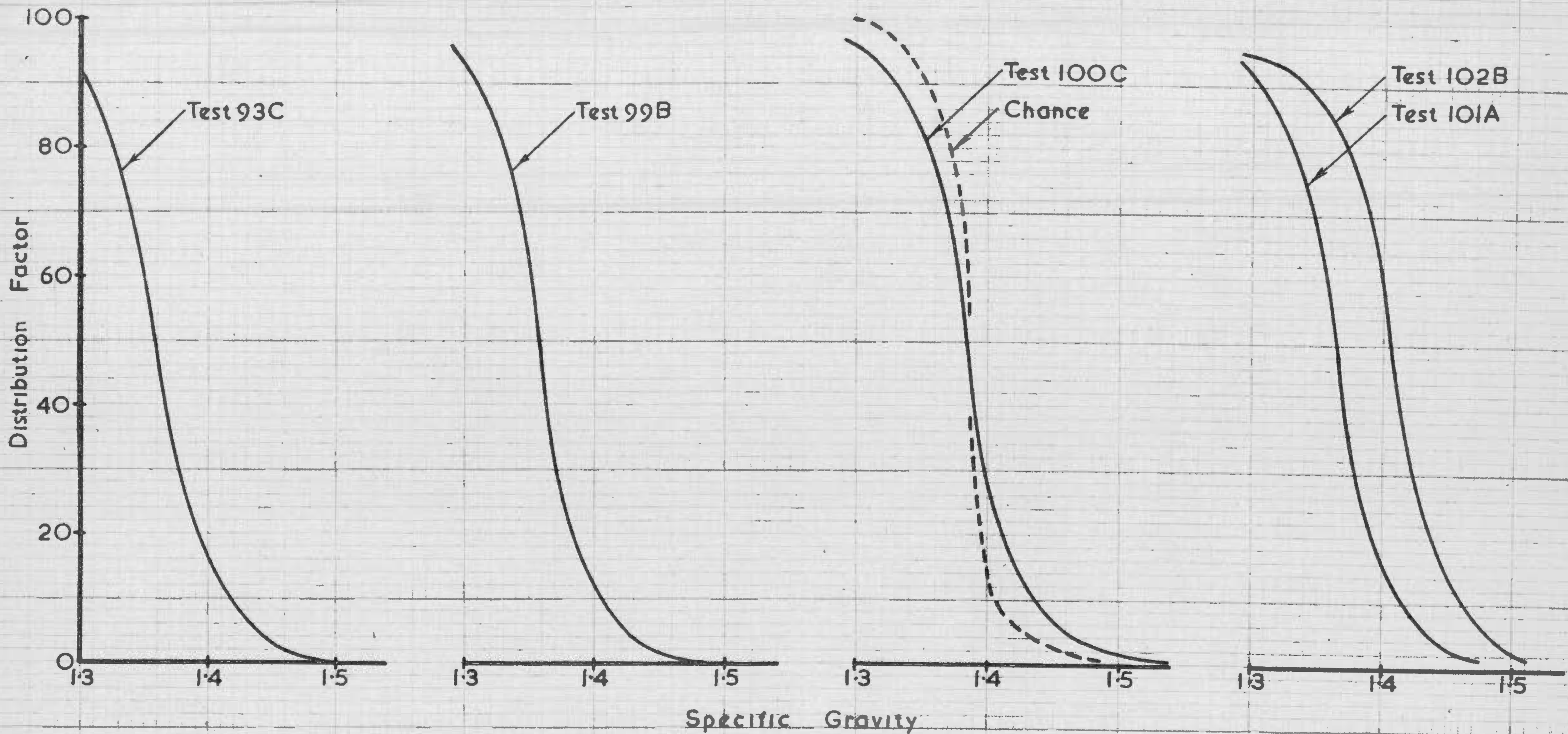

FIGURE 33 Tromp curves relating to Table 26

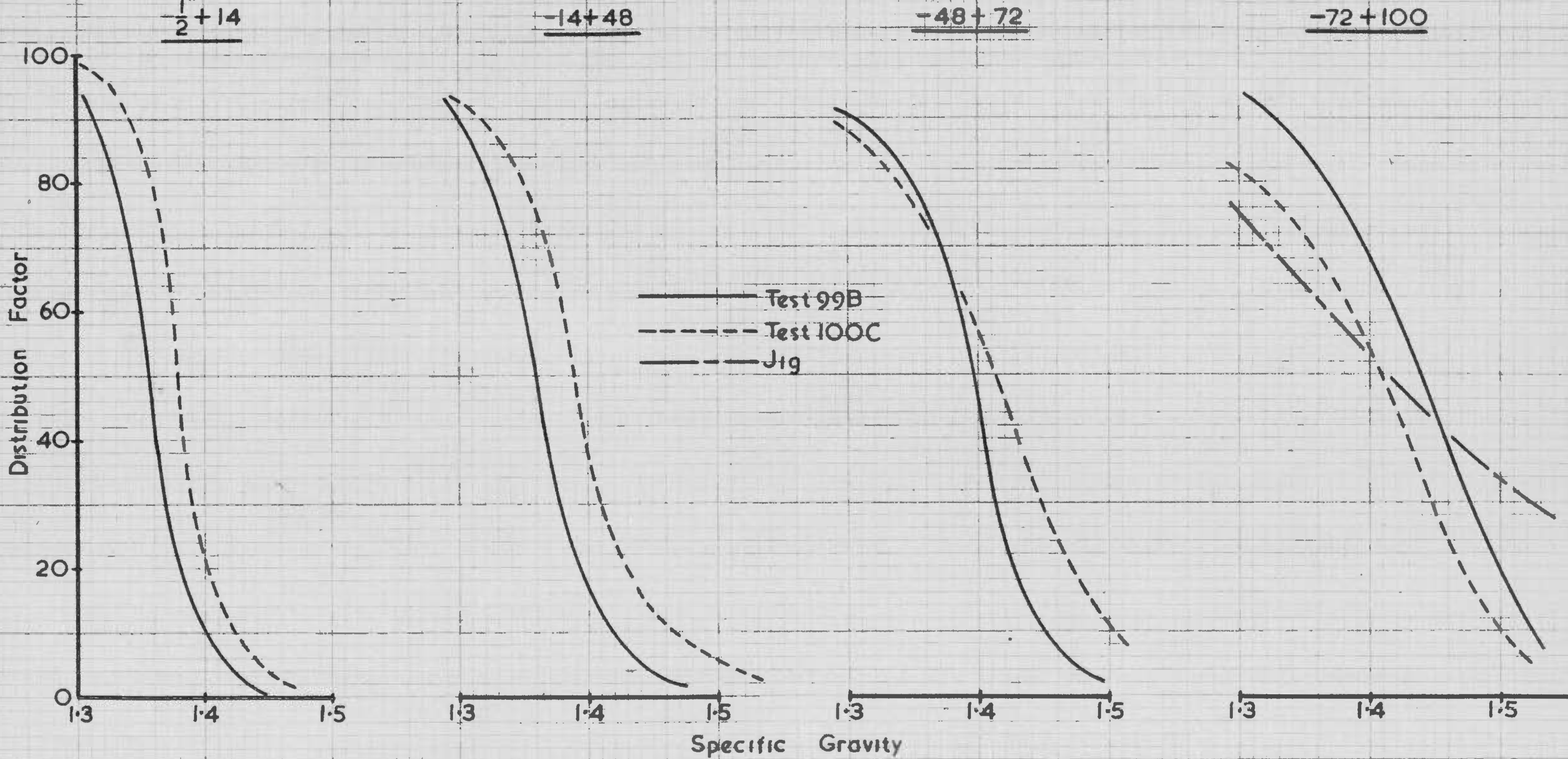

FIGURE 34 Tromp curves of the size fractions of Tests 99B and 100C

gravity distributions of the feed at the cutting points (see Table 26). As will be noted, the efficiency values obtained exceed 90 per cent in all cases. Since Fraser and Yancey efficiency values of 80 to 85 per cent are usually considered acceptable in South Africa,[20] it may be concluded that the performance of the cyclone was entirely satisfactory.

The composite Tromp distribution factor curves (i.e. -½" +100 mesh material) in Fig. 33, show that extremely sharp separations were obtained in all tests. The Tromp distribution factor curve for a Chance washer, shown superimposed on that of Test 100C, was obtained when washing South African coal, (graded minus 5" plus ¾"), at 1.525 specific gravity by means of this process.[19] It will be observed that the two curves do not differ appreciably, thus indicating that the performance of the cyclone washer, when treating fine coal, is very nearly equal to that of the conventional types of heavy medium washer treating coarse coal. In other words, the high efficiencies commonly associated with heavy medium washers, such as the Chance process, may be obtained in fine coal cleaning by applying the Cyclone washer.

The Tromp distribution factor curves for the various size fractions of the feed (Fig. 34) show that the efficiency of separation is very high for particles ranging in size from ½ inch down to about 48 mesh and that the efficiency begins to fall off for smaller particles. However, comparing the Cyclone Tromp distribution factor curves for minus 72 mesh plus 100 mesh material with that of a Jig treating minus 8 mm plus 0.5 mm feed,[5] it will be seen that, compared with a Jig, the performance is quite satisfactory at least for material down to 100 mesh in size.

It will be observed in Table 26 that the cyclone not only effected remarkably sharp separations at very difficult specific gravities, but that the capacity was also phenomenally high for so small an apparatus in all cases. In view of these high capacities, it may be concluded that the results obtained are a fair reflection of practical operation. Since each cyclone

unit..../

unit has such a large capacity, it will be appreciated that the total capital cost of these components would be relatively low in a commercial installation.

Coking Tests.

Although the separations described were not effected with a view to producing the best coking products from the raw duffs, the products actually obtained were coked in a small electrically heated oven of about 600 lb. capacity, in order to obtain some idea of the coking qualities of this material. While it is beyond the scope of the present paper to discuss in detail the relative qualities of the various cokes made, a summary of the principal conclusions drawn from these coking tests may not be out of place.

The product obtained by washing Coronation duff at 1.36 specific gravity (Test 93C) was not considered to be of any value as a straight coking coal. However, it was found that this product constituted a suitable material for blending with coals characterised by fairly high free swelling indices e.g. Waterberg coking coal.

Contrary to the Coronation duff, Landau duff coal when washed at both 1.36 and 1.385 specific gravities (Tests 99B and 100C) yielded products which could be regarded as being of a type closely approaching that of straight coking coal. Both products were considered to be excellent blend coking coals.

The products produced from Springbok duff (Tests 101A and 102B) were found to be below par as straight coking coals. A suitable blend material was, however, obtained after washing this coal at 1.365 specific gravity (Test 101A).

Utilisation of Tailings.

Detailed analyses of the tailings obtained from the Tests in Table 26 are reported in Table 27. In order to simulate two stage washing, the tailings were refloated at 1.58 specific gravity in Carbon Tetrachloride. These results are also shown in Table 27.

Table 27 ..../

TABLE 27.

ANALYSES OF THE TAILINGS OBTAINED FROM THE TESTS IN TABLE 26

Test No.	Whole Tailing					Float 1.58 material			
	Yield % of Feed	Ash %	H_2O %	Cal.Val lbs/lb	Vol.Mat. %	Yield % of Feed	Ash %	Vol.Mat %	Cal.Val. lbs/lb.
93C	68.6	15.1	2.7	12.2	-	62.6	10.5	-	12.8
99B	64.0	19.2	1.6	11.6	23.3	52.4	12.7	27.7	12.7
100C	52.3	21.3	1.9	11.2	22.1	42.2	13.6	23.6	12.5
101A	64.3	16.2	2.8	12.2	23.7	57.6	11.6	24.3	12.7
102B	46.5	17.7	2.9	11.5	22.9	40.3	13.0	23.4	12.5

Comparing the values in Table 27 with the corresponding values for the raw duff shown in Table 25, it will be observed that the ash content of the whole tailing is somewhat higher than that of the raw duff in all cases and the calorific value is correspondingly lower. These differences are however, not appreciable.

Although the tailings could undoubtedly be used as boiler fuel without further preparation, it will be noted that the quality of product which can be produced by rewashing the tailing at 1.58 specific gravity is in all cases at least equal to the raw duff and that the proportion of waste material is only of the order of 7 to 10 per cent of the raw duff.

These tests illustrate, then, that the coal which remains after the coking fraction has been removed from the duff is not waste material but that a large proportion of it, if not all, could be utilised as boiler fuel. The need for rewashing the tailing will depend largely on economic considerations and can not be considered in the present paper.

Drainage of Washery Products.

Since the moisture content of fine coal after washing is an important economic factor, it was decided to carry out a few drainage tests in order to obtain some idea of the difficulty

of..../

of reducing the moisture content of the cyclone products to an acceptable value.

In practice, the product and tailing issuing from the cyclone would be rinsed free of heavy medium on 30 to 52 mesh screens, and finer material would be recovered separately from the medium as explained in Appendix 1. Since the percentage of material finer than 30 to 52 mesh in the raw duff is comparatively low (see Table 25), the drainage tests were confined to the coarser fraction.

A quantity of tailing from Test 102B was accordingly dedusted on a 40 mesh screen and was thoroughly wetted. This material was then placed in a 40 gallon oil drum, fitted with approximately 1 square foot of 40 mesh screen in the base, and was allowed to drain. Several tests were carried out in which the bulk density of the charge and the height of the drainage bunker were the chief variables. Average results of these tests are shown below.

Initial Moisture Content %	Time to drain to 10% moisture hours.	Time to drain to 9% moisture hours.	Time to drain to $8\frac{1}{2}$% moisture hours.
17.2	$1\frac{1}{2}$	5	11

These tests indicate that the majority of the material would drain readily to 10 per cent moisture and that even lower values could be attained in a reasonable period of time. In other words, comparatively small drainage bunker capacity would be required in order to drain the coal to 9 or 10 per cent moisture content.

In view of the fact that coking coal is usually wetted to about 8% moisture before being charged into the coke oven, and fine coal used for steam raising is wetted to about 10% moisture in order to minimise dust nuisance etc., both the product and tailing should be satisfactory after the short drainage periods indicated.

It..../

It will not, therefore, be necessary to make provision for centrifugal or thermal drying except, possibly, in the case of the small percentage of minus 30 to 52 mesh material. In practice the coal will probably drain to a sufficient degree during transportation from the washery to the consumer and would not even require the provision of drainage bunkers.

Conclusions.

It is concluded from the tests described in this chapter that, from the technical point of view at least, the Cyclone washer would be entirely satisfactory for the recovery of the coking fraction from Witbank duff coal on a commercial scale. The coking coal produced in this way appears to be suitable for metallurgical purposes, and it is not anticipated that difficulty will be experienced in reducing the moisture content to an acceptable value. In addition, the non-coking fraction appears to be suitable for steam raising purposes, especially if it is rewashed at a higher specific gravity. In the latter case, only a relatively small proportion of the raw coal would represent waste material.

Only the economic aspect of the problem now remains to be investigated. The cost of cyclone washing is the most important information required in this connection and, in order to obtain reliable data, it would be necessary to erect a continuous pilot plant of suitable capacity. Since such a plant is not available at present, a discussion of the economic aspect of the problem will not be attempted.

APPENDIX 1.

CHOICE OF HEAVY MEDIA.

While it is beyond the scope of the present paper to discuss all the factors which would influence the choice of a heavy medium, two of the more important pertaining to the washing of fine coal in a cyclone may be considered, viz. -

(1) The influence of the viscosity of the suspension.

(2) Recovery of heavy medium after use.

Viscosity of Heavy Medium Suspensions.

According to Driessen,[5] the motion in a heavy liquid of the majority of the small particles encountered in fine coal cleaning may be approximated by Allen's formula :-

$$w = \left(\frac{4g}{30\gamma\sqrt{v}}\right)^{2/3} d \left(\gamma_s - \gamma\right)^{2/3} \qquad (1)$$

where
- w = terminal velocity of the particle
- d = diameter of a spherical particle.
- γ = specific gravity of the fluid
- γ_s = specific gravity of the particle
- g = acceleration due to gravity.
- v = kinematic viscosity of the fluid (stokes) $= \frac{\mu}{\gamma}$
- μ = absolute viscosity of the fluid (poises)

In the case of the cyclone washer, it can be shown that the above formula remains unchanged except that the constant, g, must be replaced by the centrifugal acceleration. Since the centrifugal acceleration is several times that due to gravity, it follows that the use of a cyclone increases the separating velocity of a particle substantially.

As explained previously, the time required for a separation to be completed is inversely proportional to the velocity of the particles. Now, it will be appreciated that the time required for separation has an important bearing on the efficiency, if the size of the washer, period of pulp retention, etc., are constant. Thus, the efficiency of separation will be

impaired.../

impaired if the separating velocity is less than a certain minimum value which depends on the operating conditions. In other words, the velocities of all particles should be as high as possible in order to ensure optimum efficiency.

It is clear from equation (1) that the terminal velocity of a particle decreases -

(a) as its diameter decreases,

(b) as its specific gravity approaches that of the separating fluid.

(c) as the viscosity of the separating fluid increases.

The viscosity of the fluid (the only factor over which one can exercise control) is thus of great importance in the case of small, near gravity particles and should, therefore, be as low as possible to ensure optimum efficiency of separation of such material.

The viscosity of a heavy medium suspension may be determined by means of a consistometer.[21] This apparatus consists of a vertical glass cylinder, having a volume of about 250 c.c., to which a capillary tube (approximately 2.5 mm bore) is attached at the lower end. The suspension to be tested is placed in the glass cylinder and is agitated by means of a small stirrer. The suspension is then allowed to flow through the capillary tube and the time taken for 100 cc to pass is noted, timing being commenced at a fixed level in the glass vessel. The consistometer is calibrated in the same manner using sucrose solutions of known viscosity, and the viscosity of the suspension is readily determined from the calibration data.

A few samples of potential heavy media, ground to suitable fineness, were obtained and suspensions of various specific gravities were prepared and tested in a consistometer as explained. The temperatures of the suspensions were kept constant at $25^{\circ}C \pm \frac{1}{2}^{\circ}$ while determining the viscosity. Details of the heavy media tested are given in Table 28 and curves of viscosity versus specific gravity of suspension are shown in

Fig..../

Fig. 35.

TABLE 28.

Heavy Medium	Sieve Analysis (B.S.S.)			Specific Gravity.
	+200	-200 +240	-240	
Shale	0	0	100	2.46
Silica Flour	5.6	4.1	90.3	2.61
Barytes	0	0.76	99.24	4.17
Magnetite	1.1	1.1	97.8	4.76
Ferrosilicon	28	5.1	66.9	6.5

Each curve in Fig 35 indicates that the viscosity is approximately proportional to the specific gravity of the suspension until a point is reached above which the viscosity commences to rise at a continuously increasing rate. A critical value of the suspension specific gravity is soon reached, at which point the viscosity commences to increase very rapidly.

It will be noted that the critical suspension specific gravity is not constant for all the heavy media, but appears to depend on the specific gravity of the medium. Taking the critical suspension specific gravities for shale and barytes as 1.47 and 2.1, it can be calculated that the concentration of solids by volume in the pulp was 32 per cent and 34 per cent respectively. In other words, the viscosity of a suspension is a function of the concentration of solids by volume, and since barytes has a higher specific gravity than shale, a higher suspension specific gravity is attained when the concentration by volume is the same in both cases. It can also be shown that the critical suspension specific gravity for any given heavy medium depends on the size of the particles. In general, the finer the medium the lower is the critical specific gravity and vice versa. The size of the medium particles which should be used, on the other hand, is governed by the size of the coal to be treated (see Hirst's equation, page 86)

Neglecting..../

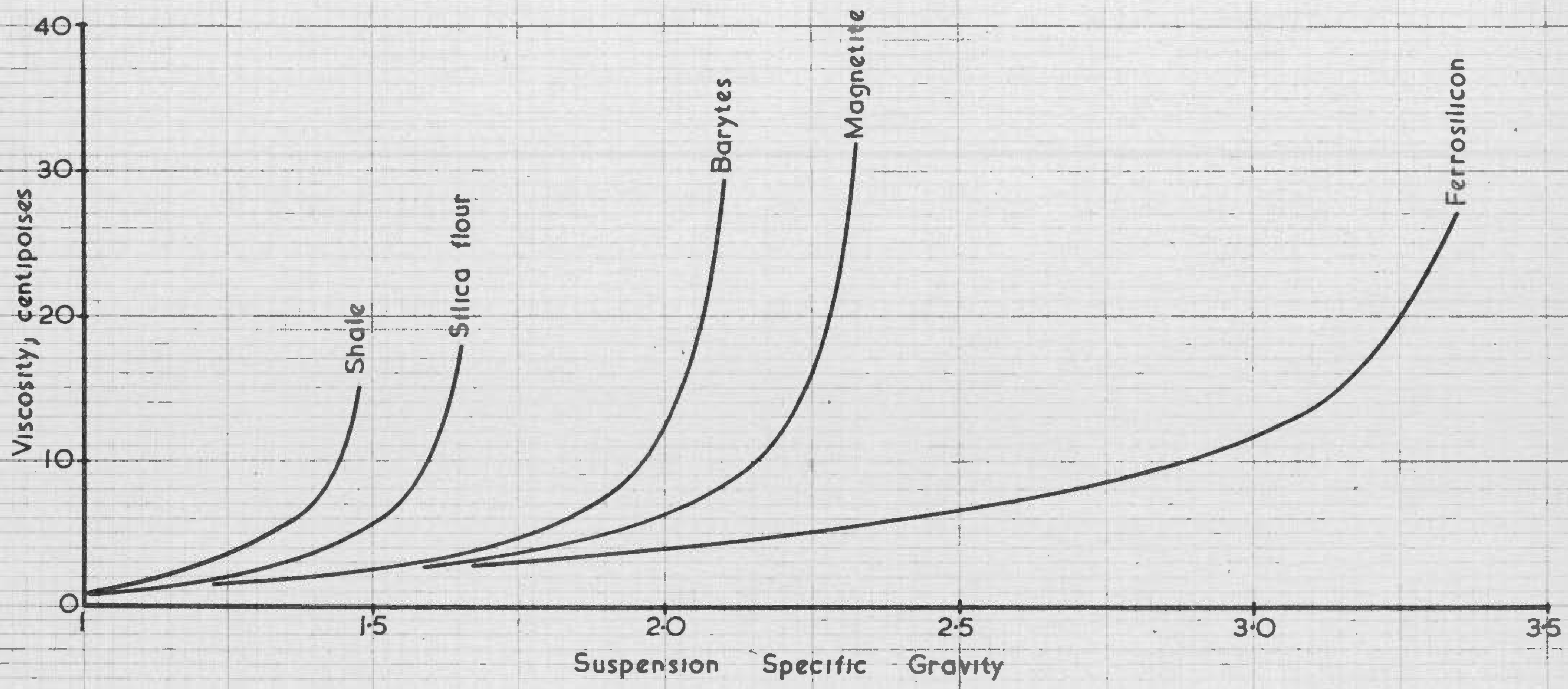

FIGURE 35 Relationship between Viscosity and Suspension Specific Gravity

Neglecting the differences in the size gradings of the various media for the purpose of the present discussion, it is clear from Figure 35 that shale and silica flour could not be used for separations higher than about 1.4 and 1.55 specific gravity respectively. Even at specific gravities lower than these, barytes, magnetite and ferrosilicon would probably be preferred when fine material is to be washed, on account of the lower viscosities obtained when using these media. For example, assume that a particle of given diameter and specific gravity is to be washed at 1.35 specific gravity. From Figure 35 it is found that the viscosities of shale, silica flour and barytes suspensions having 1.35 specific gravity are approximately 5.0, 3.0 and 2.0 centi poises respectively. Since the separation of a definite particle at a fixed specific gravity is being considered, it follows from equation (1) that

$$w = K \mu^{-\frac{1}{3}} \quad \text{(where K is a constant)} \qquad (2)$$

Substituting the values of the viscosities for the various media in equation (2), and taking the velocity of the particle in the shale suspension as unity, it is found that the relative velocities of the particle would be as follows :-

in shale suspension	1
in silica flour suspension	1.19
in barytes suspension	1.36

It will thus be noted that the separating velocity of the particle would be about 36 per cent greater in a barytes suspension of 1.35 specific gravity, than it would be in a shale suspension having the same specific gravity. As explained previously, this difference in the separating velocities would become progressively more important as the diameter of the particle decreases and its specific gravity approaches that of the suspension.

Unfortunately, a sufficiently large quantity of shale or silica flour of suitable size grading could not be obtained, and it was, therefore, not possible to determine the influence of

increased..../

increased viscosity on cyclone performance by actually conducting washing tests.

Recovery of Heavy Medium after use.

As is the case in all heavy media processes, the separated products are associated with heavy medium particles when they leave the cyclone. Since the presence of heavy medium would increase the ash content of the clean coal, it follows that it must be removed. In addition, finely ground heavy medium is comparatively expensive (possibly with the exception of naturally occuring media such as Loess) and must, therefore, be recovered from the tailing for re-use.

It is customary to pass the product and tailing over vibrating screens in order to drain off as much of the medium as possible and the remainder is rinsed off on these screens by means of water sprays. However, as it is considered that it would not be practical to use screens finer than about 30 to 52 mesh for this purpose, on a commerical scale, a significant proportion of the coal would pass through the draining and rinsing screens with the heavy medium. This fine material can not be allowed to accumulate in the medium, as it increases the viscosity of the suspension and, in addition, the fines from the product represent saleable coal.

The heavy media in general use may be divided into two groups according to their magnetic properties. The non-magnetic media include sand, shale and barytes, while magnetite and ferro silicon constitute the magnetic group. Coal particles may be removed from the former by froth flotation and from the latter by magnetic separation.

Froth flotation is an expensive process, and the use of a non-magnetic medium would probably not be an economic proposition in cases where all the fines are included in the cyclone feed. The fine coal would constitute only a small percentage of the pulp from which it is to be separated and a froth flotation

plant.../

plant of comparatively large capacity would consequently be required. It would thus appear advantageous when using a non-magnetic medium to remove all minus 30 to 52 mesh material from the feed prior to washing in the cyclone. The percentage of coal slime in the heavy medium would be reduced appreciably in this way, and it would probably be satisfactory to bleed only a comparatively small percentage of heavy medium for treatment in a froth flotation cell, in order to control the concentration of slimes. The fines removed from the cyclone feed could either be dumped, or be cleaned by froth flotation or by other means as circumstances demand, or be used as such.

Magnetic separation, on the other hand, is relatively inexpensive in operation and it should consequently be feasible to treat all the coal in the cyclone and thus take full advantage of the high separating efficiency of this washer.

APPENDIX 2.

FLOAT AND SINK ANALYSES OF THE SAMPLES.

NOTE. In all tests in which the raw feed consisted entirely of material larger than 1 mm. the float and sink analyses of the products and tailings obtained were carried out fractionally. When the raw feed consisted entirely of material smaller than 1 mm., the float and sink analyses were carried out cumulatively. The same procedure was adopted in the case of tests in which the products and tailings were divided into a number of size fractions (i.e. plus 1 mm material was analysed fractionally and minus 1mm material was analysed cumulatively).

For the purpose of the following tables, material larger than 1 mm and material finer than 1 mm will be referred to as "coarse" and "fine" respectively.

APPENDIX 2.

TABLE 29: Fractional Yields and Ash Contents of Coarse Products

Test No.		Specific Gravity Fractions						
		<1.3	1.3-1.35	1.35-1.4	1.4-1.45	1.45-1.5	1.5-1.58	>1.58
11B	Yield %	13.5	47.9	26.9	8.3	1.9	0.8	0.8
	Ash %	3.0	5.9	9.3	12.6	15.9	21.7	38.4
17C	Yield %	27.6	49.6	15.4	3.8	1.5	1.6	.5
	Ash %	3.3	5.7	8.2	11.5	16.0	-	-
17H	Yield %	52.6	37.6	4.5	2.9	1.0	1.0	0.4
	Ash%	3.3	5.4	8.0	11.9	15.6	-	-
18C	Yield %	43.3	41.6	10.6	2.4	2.2	0	0
	Ash %	3.4	5.9	7.7	10.3	20.9	-	-
19C	Yield %	43.0	45.2	8.4	2.5	1.0	0	0
	Ash %	3.6	6.3	7.8	10.2	-	-	-
20D	Yield %	36.3	44.3	16.5	2.2	0.7	0	0
	Ash %	3.4	6.0	7.9	10.0	20.5	-	-
20G	Yield %	28.6	41.6	26.6	2.7	0.4	0	0
	Ash %	3.3	6.0	8.1	9.8	16.5	-	-
21A (-$\frac{1}{4}$" +1mm)	Yield %	35.2	44.2	18.0	2.3	0.3	0	0
	Ash %	-	-	-	-	-	-	-
21B (-$\frac{1}{4}$" +1mm)	Yield %	34.0	45.8	18.8	1.4	0	0	0
	Ash %	-	-	-	-	-	-	-
21G (-$\frac{1}{4}$" +1mm)	Yield %	37.7	42.6	12.5	4.3	3.0	0	0
	Ash %	-	-	-	-	-	-	-
21H ($\frac{1}{4}$" +1mm)	Yield %	44.3	45.5	7.4	2.2	0.7	0	0
	Ash %	-	-	-	-	-	-	-
28A	Yield %	19.8	46.1	30.7	3.2	0.3	0	0
	Ash %	3.2	5.4	7.8	9.9	-	-	-
28B	Yield %	32.6	57.4	9.6	0.4	0	0	0
	Ash %	3.4	5.7	7.5	-	-	-	-
29A	Yield %	21.4	34.1	19.9	17.1	5.7	1.9	0
	Ash %	4.1	5.6	7.8	10.6	-	-	-
29D	Yield %	32.9	59.1	6.5	1.5	0	0	0
	Ash %	3.1	5.2	7.4	-	-	-	-
29F	Yield %	24.1	52.9	20.7	1.5	0.9	0	0
	Ash %	3.2	5.5	7.4	9.8	-	-	-

APPENDIX 2.

TABLE 29 Continued:

Test No.		Specific Gravity Fractions.						
		<1.3	1.3-1.35	1.35-1.4	1.4-1.45	1.45-1.5	1.5-1.58	>1.58
29G	Yield%	30.4	40.6	25.4	3.3	0.3	0	0
	Ash%	-	-	-	-	-	-	-
29H	Yield%	18.8	37.6	30.1	13.3	0.2	0	0
	Ash%	-	-	-	-	-	-	-
29I	Yield%	16.3	30.6	26.5	24.4	2.2	0	0
	Ash%	-	-	-	-	-	-	-
32A	Yield%	27.0	38.7	31.5	1.9	0.9	0	0
	Ash%	3.0	8.5	14.3	20.7	34.7	-	-
33A	Yield%	35.8	37.2	21.9	2.7	0.9	0.8	0.8
	Ash%	3.0	8.9	14.8	20.2	23.4	26.5	35.4
34A	Yield%	30.0	39.4	23.1	5.3	0.8	1.0	0.5
	Ash%	3.0	8.5	14.8	19.3	24.1	30.0	36.6
35A	Yield%	31.6	34.0	28.0	4.1	0.9	0.7	0.6
	Ash%	2.7	9.0	15.3	19.1	23.0	27.2	38.3
50G	Yield%	17.3	32.2	28.3	13.4	6.1	2.6	0
	Ash%	3.6	5.4	7.6	-	-	-	-
51C	Yield%	13.9	44.4	27.8	12.3	1.6	0	0
	Ash%	3.6	5.7	7.9	-	-	-	-
61B	Yield%	31.3	55.8	10.4	1.9	0.6	0	0
	Ash%	3.1	5.1	7.4	9.3	12.5	-	-
70A	Yield%	23.6	32.6	28.6	12.0	2.6	0.7	0
	Ash%	3.9	5.7	8.1	10.9	15.4	21.0	-
70C	Yield%	27.9	31.8	29.6	9.0	1.8	0	0
	Ash%	3.4	7.0	8.1	10.8	15.4	-	-
70E	Yield%	37.1	39.3	19.6	3.7	0.4	0	0
	Ash%	-	5.9	8.1	10.9	17.8	-	-
70G	Yield%	46.5	39.9	12.2	1.3	0	0	0
	Ash%	-	6.1	7.7	11.0	-	-	-
71A	Yield%	29.3	33.5	28.3	6.6	2.3	0	0
	Ash%	4.3	5.3	7.7	10.4	15.6	-	-
71C	Yield%	29.6	40.9	22.6	5.0	1.9	0	0
	Ash%	4.7	5.3	7.3	10.3	15.5	-	-
71E	Yield%	41.2	38.8	15.2	4.1	0.7	0	0
	Ash%	3.3	5.2	7.2	8.7	18.6	-	-
72C	Yield%	23.8	33.9	19.4	14.2	5.7	3.0	0
	Ash%	3.1	5.4	8.1	10.8	-	-	-
72G	Yield%	16.8	34.8	20.2	17.7	6.0	4.1	0.4
	Ash%	3.1	5.2	8.3	10.8	-	-	-
73A	Yield%	19.4	24.7	18.5	19.6	9.0	6.7	2.1
	Ash%	3.4	5.7	8.0	11.0	-	-	-

APPENDIX 2.

TABLE 29 Continued:

Test No.		Specific Gravity Fractions						
		<1.3	1.3-1.35	1.35-1.4	1.4-1.45	1.45-1.5	1.5-1.58	>1.58
73E	Yield%	23.5	26.6	19.0	17.8	8.0	4.0	1.1
	Ash%	3.4	6.7	8.0	11.0	-	-	-
73H	Yield%	41.2	37.9	14.3	4.8	1.8	0	0
	Ash%	3.2	5.5	7.8	-	-	-	-
74A	Yield%	26.0	43.2	17.8	7.3	4.3	1.4	0
	Ash%	2.9	5.3	8.3	-	-	-	-
74D	Yield%	18.4	44.3	21.3	8.1	6.7	1.2	0
	Ash%	3.3	5.9	8.9	11.5	-	-	-
75D	Yield%	22.4	35.4	21.1	13.6	4.9	2.0	0.6
	Ash%	3.1	5.5	8.2	-	-	-	-
75F	Yield%	23.4	34.5	20.5	12.5	6.1	2.4	0.6
	Ash%	3.1	5.7	8.3	-	-	-	-
80C	Yield%	36.5	46.4	15.7	1.4	0	0	0
	Ash%	3.2	5.6	7.6	-	-	-	-
80E	Yield%	37.4	46.7	14.1	1.8	0	0	0
	Ash%	3.3	5.8	7.8	-	-	-	-
80D	Yield%	33.3	46.1	16.8	3.0	0.8	0	0
	Ash%	3.2	5.7	7.8	-	-	-	-
80F	Yield%	33.0	51.8	13.4	1.8	0	0	0
	Ash%	3.2	5.9	7.9	-	-	-	-
80G	Yield%	44.8	46.7	6.8	1.8	0	0	0
	Ash%	3.4	5.9	7.9	-	-	-	-
80H	Yield%	34.2	44.9	17.6	2.3	1.0	0	0
	Ash%	3.1	5.9	7.9	-	-	-	-
84A	Yield%	59.6	37.4	2.0	1.0	0	0	0
	Ash%	3.6	5.5	7.7	-	-	-	-
88B	Yield%	64.0	35.0	1.0	0	0	0	0
	Ash%	3.3	5.3	9.0	-	-	-	-
90A	Yield%	40.0	10.7	0.8	0	0	0	0
	Ash%	3.5	5.6	7.6	-	-	-	-
91A (-½"+14#)	Yield%	29.4	41.0	27.2	2.3	0.1	0	0
	Ash%	3.8	5.1	6.6	10.7	20.1	-	-
93C (-½"+14#)	Yield%	20.6	33.2	43.9	2.2	0.1	0	0
	Ash%	3.1	5.1	6.6	9.9	19.8	-	-
95A	Yield%	58.9	32.8	6.8	1.5	0	0	0
	Ash%	3.7	5.9	8.1	-	-	-	-
95B	Yield%	48.3	35.3	15.4	1.0	0	0	0
	Ash %	3.8	6.1	7.9	-	-	-	-

APPENDIX 2.

TABLE 29 Continued:

Test No.		Specific Gravity Fractions						
		<1.3	1.3-1.35	1.35-1.4	1.4-1.45	1.45-1.5	1.5-1.58	>1.58
95C	Yield%	41.7	36.4	20.6	1.3	0	0	0
	Ash%	3.5	5.9	7.7	-	-	-	-
95D	Yield%	43.3	34.1	21.4	1.2	0	0	0
	Ash%	3.8	6.0	7.7	-	-	-	-
95E	Yield%	52.4	36.1	11.0	0.5	0	0	0
	Ash%	3.9	6.4	7.9	-	-	-	-
95F	Yield%	64.0	32.9	2.7	0.4	0	0	0
	Ash%	3.8	6.1	8.0	-	-	-	-
96B	Yield%	68.3	30.7	0.7	0.3	0	0	0
	Ash%	3.5	5.3	6.9	10.7	-	-	-
96C	Yield%	67.9	30.9	1.1	0.1	0	0	0
	Ash%	3.5	5.6	7.1	11.4	-	-	-
96D	Yield%	45.8	47.2	6.7	0.3	0	0	0
	Ash%	3.4	5.7	7.7	-	-	-	-
96E	Yield%	26.1	65.5	8.2	0.2	0	0	0
	Ash%	3.1	5.1	7.0	-	-	-	-
96F	Yield%	44.8	53.0	2.0	0.2	0	0	0
	Ash%	3.2	5.2	7.0	-	-	-	-
99B (-½" +14#)	Yield%	35.6	48.2	15.5	0.7	0	0	0
	Ash%	3.4	5.7	7.9	12.9	-	-	-
100C (-½" +14#)	Yield%	27.3	41.7	27.8	2.8	0.4	0	0
	Ash%	3.5	6.1	8.2	10.5	23.7	-	-
101A (-½" +14#)	Yield%	30.3	46.3	20.8	2.1	0.5	0	0
	Ash%	3.9	5.8	7.5	10.4	20.2	-	-
102B (-½" +14#)	Yield%	15.4	33.8	35.0	14.9	0.9	0	0
	Ash%	3.7	5.6	8.0	9.8	15.7	-	-

N.B.# = square mesh.

APPENDIX 2.

TABLE 30: Fractional Yields and Ash Contents of Coarse Tailings.

Test No.		Specific Gravity Fractions.						
		<1.3	1.3-1.35	1.35-1.4	1.4-1.45	1.45-1.5	1.5-1.58	1.58
11B	Yield %	0.3	1.8	5.3	20.1	18.0	20.3	34.2
	Ash %	3.6	6.8	9.8	13.1	16.9	22.5	43.6
17C	Yield %	1.4	5.4	23.2	26.4	11.5	16.9	15.2
	Ash %	3.8	6.4	8.6	11.7	15.5	-	-
17H	Yield %	3.1	14.4	25.5	25.3	9.0	12.2	10.5
	Ash %	3.6	6.5	8.5	11.6	15.5	-	-
18C	Yield %	0.9	6.2	22.4	24.0	17.9	12.0	16.5
	Ash %	4.7	6.7	8.5	11.2	15.0	21.9	46.7
19C	Yield %	0.7	6.5	24.0	24.6	16.8	12.0	15.5
	Ash %	4.6	6.7	8.4	11.4	15.5	21.5	44.9
20D	Yield %	0	4.8	21.0	23.3	21.2	12.6	17.2
	Ash %	-	7.1	8.7	11.3	15.4	21.4	45.7
20G	Yield %	0	3.4	16.8	26.8	21.8	14.4	16.8
	Ash %	-	6.5	8.7	12.1	15.3	20.9	45.3
21A (-$\frac{1}{4}$"+1mm)	Yield %	0.6	3.1	19.9	29.1	19.6	12.1	15.7
	Ash %	-	-	-	-	-	-	-
21B (-$\frac{1}{4}$"+1mm)	Yield %	0.6	3.3	16.9	33.5	15.3	15.0	15.4
	Ash %	-	-	-	-	-	-	-
21G (-$\frac{1}{4}$"+1mm)	Yield %	1.1	6.5	21.0	27.6	17.5	12.7	13.4
	Ash %	-	-	-	-	-	-	-
21H (-$\frac{1}{4}$"+1mm)	Yield %	0.9	7.6	17.1	32.7	16.5	11.8	13.2
	Ash	-	-	-	-	-	-	-
28A	Yield %	1.0	4.0	13.5	26.9	19.5	35.0(>1.5)	
	Ash %	-	5.8	8.1	10.7	-	-	
28B	Yield %	0.6	4.3	18.3	34.1	18.3	24.4(>1.5)	
	Ash %	-	5.9	7.8	10.6	-	-	-
29A	Yield %	1.8	10.5	16.7	25.0	18.8	12.5	14.7
	Ash %	4.4	5.5	8.1	10.8	-	-	-
29D	Yield %	0.7	4.8	22.4	27.3	44.8(>1.45)		
	Ash %	4.0	6.3	7.6	10.5	-	-	-
29F	Yield %	0.2	2.5	21.5	26.0	19.2	30.6(>1.5)	
	Ash %	-	5.7	8.0	10.7	-	-	-
29G	Yield %	0	1.6	10.3	35.2	23.5	29.4(>1.5)	
	Ash %	-	-	-	-	-	-	-
29H	Yield %	0	0.8	2.1	35.3	29.9	32.0(>1.5)	
	Ash %	-	-	-	-	-	-	-
29I	Yield %	0	0	1.8	17.8	39.2	41.1(>1.5)	
	Ash %	-	-	-	-	-	-	-

APPENDIX 2.

TABLE 30 Continued:

Test No.		Specific Gravity Fractions						
		<1.3	1.3-1.35	1.35-1.4	1.4-1.45	1.45-1.5	1.5-1.58	>1.58
32A	Yield	1.7	4.9	21.4	22.1	15.2	10.9	23.8
	Ash%	3.7	10.6	16.6	21.9	26.6	31.9	59.5
33A	Yield%	0.6	2.2	7.8	18.3	10.2	15.8	45.2
	Ash%	2.9	9.1	16.2	21.6	26.8	31.9	49.8
34A	Yield%	0.6	2.3	9.3	21.2	17.3	16.8	32.5
	Ash%	2.8	8.4	15.9	21.2	26.3	31.6	47.7
35A	Yield%	0.5	1.6	9.7	13.7	12.2	10.6	51.7
	Ash%	2.7	9.4	16.8	21.2	26.0	31.0	63.0
50G	Yield	2.4	8.6	17.2	24.7	19.2	15.2	12.6
	Ash%	3.6	5.8	7.9	-	-	-	-
51C	Yield%	0.7	5.7	15.0	26.1	23.8	16.3	12.6
	Ash%	3.3	5.6	7.8	-	-	-	-
61B	Yield%	1.5	9.4	20.7	19.2	21.1	28.1(>1.5)	
	Ash%	3.3	5.4	7.8	9.9	14.4	30.2	-
70A	Yield%	2.1	9.6	15.7	20.9	20.9	14.8	16.0
	Ash%	3.7	6.1	8.5	11.1	15.1	21.3	43.2
70C	Yield%	1.7	8.5	15.0	24.9	20.3	29.6(>1.5)	
	Ash%	3.4	6.0	8.2	11.0	14.6	31.6	-
70E	Yield%	1.5	8.1	19.1	26.0	18.6	26.7(>1.5)	
	Ash%	3.4	6.0	8.3	10.7	14.5	32.6	
70E	Yield%	1.5	8.1	19.1	26.0	18.6	26.7(>1.5)	
	Ash%	3.4	6.0	8.3	10.7	14.5	32.6	
70G	Yield%	1.8	11.5	24.1	24.5	38.0(>1.45)		
	Ash%	3.6	6.2	8.4	10.7	26.1		
71A	Yield%	3.7	9.5	16.2	24.1	16.3	30.1(>1.5)	
	Ash%	3.5	5.6	8.2	11.0	14.5	33.8	
71C	Yield%	2.7	9.7	20.0	24.3	16.8	26.6(>1.5)	
	Ash%	3.5	5.5	8.1	10.8	14.8	33.3	
71E	Yield%	2.6	9.6	24.4	23.3	16.4	23.7(>1.5)	
	Ash%	3.3	5.7	8.0	10.7	14.7	32.2	
72C	Yield%	3.9	16.0	16.1	20.7	15.9	13.0	14.4
	Ash	3.0	5.5	8.1	10.5	-	-	-
72G	Yield%	2.4	12.8	16.8	21.5	14.9	14.6	17.0
	Ash%	3.1	5.5	8.3	11.0	-	-	-
73A	Yield%	5.5	13.4	14.0	19.4	12.7	14.9	19.9
	Ash%	3.1	6.0	8.3	11.2	-	-	-
73E	Yield%	4.2	13.4	12.7	19.6	14.5	15.3	20.3
	Ash%	3.0	5.8	8.3	11.1	-	-	-
73H	Yield%	4.4	18.9	21.8	20.2	13.1	21.6(>1.5)	
	Ash%	3.1	5.8	8.7	11.6	-	-	-

APPENDIX 2.

TABLE 30 Continued:

Test No.		Specific Gravity Fractions						
		<1.3	1.3-1.35	1.35-1.4	1.4-1.45	1.45-1.5	1.5-1.58	>1.58
74A	Yield%	7.7	15.4	19.6	17.2	15.8	12.9	11.4
	Ash%	3.5	5.9	8.3	11.4	-	-	-
74D	Yield%	4.8	13.8	21.2	17.1	16.8	14.5	11.8
	Ash%	3.5	6.0	8.8	11.5	-	-	-
75D	Yield%	4.8	16.0	17.9	16.5	17.3	11.5	16.0
	Ash%	2.9	5.6	8.3	-	-	-	-
75F	Yield%	5.9	13.2	17.9	18.1	16.8	11.8	16.3
	Ash%	3.3	6.0	8.7	-	-	-	-
80C	Yield%	1.3	7.0	18.2	26.1	47.5(>1.45)		
	Ash%	3.5	6.4	8.5	-	-	-	-
80E	Yield%	0.9	4.6	20.0	25.2	49.3(>1.45)		
	Ash%	3.7	6.1	8.6	-	-	-	-
80D	Yield%	0.8	4.4	15.9	28.5	18.5	31.8(>1.5)	
	Ash%	3.7	6.3	8.3	-	-	-	-
80F	Yield%	1.0	4.1	20.7	23.2	51.0(>1.45)		
	Ash%	4.0	6.2	8.5	-	-	-	
80G	Yield%	1.6	11.7	22.9	20.3	43.5(>1.45)		
	Ash%	3.6	6.5	8.1	-	-	-	-
80H	Yield%	0.8	3.1	20.6	25.7	20.3	29.5(>1.5)	
	Ash%	3.6	6.2	8.4	-	-	-	-
84A	Yield%	1.6	14.3	16.1	29.4	38.6(>1.45)		
	Ash%	3.6	6.3	8.1	-	-	-	-
88B	Yield%	1.3	14.2	22.7	61.8(>1.4)		-	-
	Ash%	3.7	5.9	8.1	-	-	-	-
90A	Yield%	0.8	7.4	18.6	30.5	42.7(>1.45)		
	Ash%	3.6	6.5	8.3	-			
91A (-½" +14#)	Yield%	0.3	4.8	49.3	24.7	11.4	9.5(>1.5)	
	Ash%	3.3	5.2	7.5	11.5	16.8	41.7	
93C (-½" +14#)	Yield%	0.2	3.8	45.0	25.5	12.5	13.0(>1.5)	
	Ash%	3.8	5.2	7.4	11.1	16.4	41.3	
95A	Yield%	2.0	11.2	25.0	19.2	42.6(>1.45)		
	Ash%	3.8	6.4	8.7	-	-	-	-
95B	Yield%	1.1	5.3	29.0	21.6	43.0(>1.45)		
	Ash%	3.8	6.4	9.0	-	-	-	-
95C	Yield%	1.0	4.5	33.8	15.7	45.0(>1.45)		
	Ash%	3.9	6.8	8.6	-	-	-	-
95D	Yield%	1.2	4.0	29.3	20.7	44.8(>1.45)		
	Ash%	4.3	6.7	9.2	-	-	-	-
95E	Yield%	1.3	6.0	27.6	22.0	43.1(>1.45)		
	Ash%	4.0	6.6	9.1	-	-	-	-

APPENDIX 2.

TABLE 30 Continued:

Test No.		Specific Gravity Fractions						
		<1.3	1.3-1.35	1.35-1.44	1.4-1.45	1.45-1.5	1.5-1.58	>1.58
95F	Yield%	2.7	12.3	29.2	20.0	35.8(>1.45)		
	Ash%	4.4	6.9	9.0	-	-	-	-
96B	Yield%	3.0	21.3	19.5	20.6	35.6(>1.45)		
	Ash%	3.2	6.0	8.2	10.7	25.8	-	-
96C	Yield%	2.8	18.4	23.3	23.8	31.7(>1.45)		
	Ash%	3.6	6.3	8.7	-	-	-	-
96D	Yield%	0.8	12.8	23.4	22.9	40.1(>1.45)		
	Ash%	3.7	6.6	8.8	-	-	-	-
96E	Yield%	0.3	10.7	27.5	20.2	41.3(>1.45)		
	Ash%	4.1	6.7	8.7	-	-	-	-
96F	Yield%	0.6	12.9	31.1	21.2	34.2(>1.45)		
	Ash%	3.3	6.2	9.4	-	-	-	-
99B (-½" +14#)	Yield%	0.3	4.1	28.0	24.7	42.9(>1.45)		
	Ash%	3.4	6.3	8.9	11.9	28.8	-	-
100C (-½" +14#)	Yield%	0.2	1.5	13.6	29.3	25.0	30.4(>1.5)	
	Ash%	3.6	6.6	9.1	11.7	15.3	37.8	
101A (-½" +14#)	Yield%	0.5	4.3	21.1	37.4	19.8	16.9(>1.5)	
	Ash%	3.8	6.1	8.2	10.2	14.6	33.1	
102B (-½" +14#)	Yield%	0.3	1.6	7.8	39.2	27.2	23.9(>1.5)	
	Ash%	3.7	5.8	8.3	10.5	15.4	35.4	

APPENDIX 2.

TABLE 31. Cumulative Yields and Ash Contents of Fine Products.

Test No.		Specific Gravity.						
		1.3	1.35	1.4	1.45	1.5	1.58	>1.58
26C	Yield % Ash %	29.8 2.6	69.4 3.9	88.0 4.7	95.0 4.9	97.3 5.1	98.7 5.4	1.3 -
31A	Yield % Ash %	42.2 2.6	72.4 3.7	89.1 4.3	97.1 -	99.3 -	100 -	0 -
31B	Yield % Ash %	35.8 2.6	73.4 3.8	89.3 4.4	95.8 -	98.3 -	99.2 -	0.8 -
31C	Yield % Ash %	41.0 2.7	74.2 3.9	89.6 4.3	95.0 -	97.9 -	99.0 -	1.0 -
31D	Yield % Ash %	39.2 2.6	72.0 3.8	88.7 4.4	95.7 -	98.2 -	98.9 -	1.1 -
31F	Yield % Ash %	47.2 2.7	79.1 3.6	89.6 3.8	95.4 -	98.2 -	99.2 -	0.8 -
31T 31U 31W 31X 31Y 31Z 31AA	Yield % do % do % do % do % do % do %	34.2 36.8 31.3 34.5 29.4 34.3 24.6	66.9 71.9 65.6 66.2 65.2 67.5 47.8	83.0 89.9 85.0 86.2 88.1 90.1 66.1	92.4 94.7 94.9 95.0 95.9 96.8 81.5	95.8 97.6 98.3 97.3 99.0 99.5 90.6	97.5 99.0 98.6 99.6 99.6 99.7 96.3	2.5 1.0 1.4 0.4 0.4 0.3 3.7
47B	Yield % Ash %	34.3 -	56.7 3.8	75.0 4.9	93.5 5.5	96.8 -	99.0 -	1.0 -
47F	Yield % Ash %	33.8 2.4	65.6 3.8	83.2 4.4	93.6 5.0	96.9 -	99.1 -	0.9 -
48I	Yield % Ash %	23.5 -	48.4 4.2	63.0 4.9	82.9 5.8	91.2 -	98.3 -	1.7 -
76A	Yield % Ash %	28.0 2.9	53.9 3.9	72.1 4.9	85.0 5.7	92.9 -	99.2 -	0.8 -
76C	Yield % Ash %	47.5 2.5	76.5 3.4	92.0 4.2	94.8 -	97.9 -	99.8 -	0.2 -
76E	Yield % Ash %	24.8 2.4	48.2 3.6	67.5 5.3	83.8 6.2	90.2 6.5	98.7 -	1.3 -
85A	Yield % Ash %	32.0 2.4	85.3 3.7	97.6 4.2	99.0 -	100.0 -	- -	- -
85E	Yield % Ash %	21.9 2.4	66.2 3.8	89.2 4.8	98.6 -	100.0 -	- -	- -
86A	Yield % Ash %	30.7 2.3	77.2 3.8	91.4 -	96.8 -	99.2 -	100.0 -	- -
86B	Yield % Ash %	28.5 2.4	63.8 3.7	88.1 4.8	96.4 -	99.0 -	100.0 -	- -

APPENDIX 2.

TABLE 31, continued.

Test No.		Specific Gravity.						
		1.3	1.35	1.4	1.45	1.5	1.58	>1.58
87A	Yield %	76.2	92.0	96.7	98.8	100.0	-	-
	Ash %	2.5	3.0	3.5	-	-	-	-
87E	Yield %	41.7	88.1	97.6	99.3	100.0	-	-
	Ash %	2.5	3.8	4.2	-	-	-	-
91A -14#+48#	Yield %	27.3	72.4	96.3	98.3	100.0	-	-
	Ash %	2.4	3.8	4.3	4.5	4.8	-	-
91A -48#+72#	Yield %	20.7	53.6	89.9	97.9	100.0	-	-
	Ash %	2.1	3.4	4.8	5.3	5.8	-	-
91A -72#+100#	Yield %	22.9	58.1	77.6	96.8	99.2	100.0	-
	Ash %	2.3	3.8	4.5	5.4	5.8	-	-
93C -14#+48#	Yield %	19.8	59.5	94.0	99.4	100.0	-	-
	Ash %	2.3	4.0	4.8	4.9	5.1	-	-
93C -48#+72#	Yield %	16.5	42.8	78.5	98.0	100.0	-	-
	Ash %	2.1	3.4	4.9	5.7	6.1	-	-
93C -72#+100#	Yield %	17.4	41.7	72.2	94.4	99.0	100.0	-
	Ash %	2.4	3.4	4.6	5.7	5.9	6.3	-
99B -14#+48#	Yield %	47.1	83.9	97.1	99.3	100.0	-	-
	Ash %	2.9	4.7	4.8	5.0	5.2	-	-
99B -48#+72#	Yield %	38.2	65.0	95.6	98.8	100.0	-	-
	Ash %	2.4	3.5	5.6	5.6	5.8	-	-
99B -72#+100#	Yield %	31.0	60.8	81.0	96.7	99.1	100.0	-
	Ash %	2.4	3.7	4.7	5.9	6.0	6.3	-
100C -14#+48#	Yield %	34.6	65.2	92.3	98.1	99.7	100.0	-
	Ash %	3.8	4.3	5.4	5.6	5.8	6.0	-
100C -48#+72#	Yield %	29.1	52.2	79.3	98.3	99.0	100.0	-
	Ash %	2.4	3.6	4.9	6.2	6.3	6.6	-
100C -72#+100#	Yield %	25.9	48.6	85.0	93.3	98.9	100.0	-
	Ash %	2.5	3.6	5.3	6.1	6.5	6.9	-
101A -14#+48#	Yield %	24.9	68.9	91.9	98.1	99.5	100.0	-
	Ash %	3.2	4.4	5.1	5.5	5.6	5.8	-
101A -48#+72#	Yield %	22.5	54.5	77.9	98.6	99.0	100.0	-
	Ash %	2.7	4.0	5.1	6.3	6.4	6.6	-
101A -72#+100#	Yield %	19.9	48.3	72.0	93.7	98.1	100.0	-
	Ash %	2.8	4.0	4.8	6.5	6.7	7.1	-
102B -14#+48#	Yield %	24.0	52.2	79.6	96.7	98.8	99.1	0.9
	Ash %	3.0	4.4	5.4	6.4	6.6	6.8	WC 6.9
102B -48#+72#	Yield %	21.2	45.1	68.4	93.2	97.9	99.1	0.9
	Ash %	2.7	4.1	5.6	6.5	6.8	7.2	WC 7.5
102B -72#+100#	Yield %	22.4	42.6	62.7	92.4	98.1	99.1	0.9
	Ash %	2.9	4.0	5.6	7.4	7.5	7.8	WC 8.1

Note: W.C. = Ash content of whole coal.

APPENDIX 2.

TABLE 32: Cumulative Yields and Ash Contents of Fine Tailings.

Test No.		Specific Gravity						
		1.3	1.35	1.4	1.45	1.5	1.58	>1.58
26C	Yield %	3.0	6.5	21.7	44.9	64.6	78.1	21.9
	Ash %	2.9	4.4	6.5	8.0	9.6	11.2	48.9
31A	Yield %	3.5	11.9	27.2	52.0	66.8	79.0	21.0
	Ash %	-	4.4	5.9	-	-	-	-
31B	Yield %	2.8	8.3	22.9	48.6	67.5	79.3	20.7
	Ash %	-	4.2	6.3	-	-	-	-
31C	Yield %	2.9	8.8	22.0	47.9	66.4	81.0	19.0
	Ash %	-	4.8	6.4	-	-	-	-
31D	Yield %	3.9	10.8	22.1	47.9	66.3	78.8	21.2
	Ash %	-	4.8	6.3	-	-	-	-
31F	Yield %	6.7	17.7	29.4	54.3	69.9	82.1	17.9
	Ash %	2.8	4.7	6.4	-	-	-	-
31T	Yield %	3.3	9.3	20.4	51.3	69.6	79.5	20.5
31U	do	2.1	7.2	22.5	40.2	64.7	76.8	23.2
31W	do	2.5	6.6	14.7	40.3	63.5	78.6	21.4
31X	do	2.4	5.9	16.8	37.7	64.0	79.5	20.5
31Y	do	1.7	5.2	15.9	35.8	59.1	79.7	20.3
31Z	do	1.4	3.9	11.3	36.1	56.5	78.5	21.5
31AA	do	8.5	19.7	37.5	56.0	69.4	81.8	18.2
47B	Yield %	3.6	8.5	20.1	42.2	59.3	75.2	24.8
	Ash %	-	5.0	6.4	8.2	-	-	-
47F	Yield %	2.6	7.8	19.8	49.8	65.1	82.6	17.4
	Ash %	-	4.1	6.2	8.2	-	-	-
48I	Yield %	6.8	15.2	26.5	46.3	62.3	77.7	22.3
	Ash %	-	4.1	5.7	7.1	-	-	-
76A	Yield %	7.2	18.5	32.4	49.5	62.3	87.9	12.1
	Ash %	2.5	4.3	6.5	7.4	-	-	-
76C	Yield %	5.6	18.7	35.7	57.5	72.2	86.2	13.8
	Ash %	2.4	4.4	5.9	-	-	-	-
76E	Yield %	8.7	21.7	31.8	53.2	64.2	79.8	20.2
	Ash %	2.6	4.1	5.5	7.2	8.5	-	-
85A	Yield %	4.3	17.5	36.2	61.3	77.5	22.5(>1.5)	-
	Ash %	2.7	4.2	4.4	-	-	-	-
85E	Yield %	1.5	8.1	20.7	42.9	67.1	32.9(>1.5)	-
	Ash %	2.8	4.2	6.2	-	-	-	-
86A	Yield %	2.9	13.4	32.9	55.7	73.1	83.4	16.6
	Ash %	2.7	4.4	6.4	-	-	-	-
86B	Yield %	3.0	8.3	22.0	41.9	65.1	77.0	23.0
	Ash %	2.7	4.3	6.5	-	-	-	-
87A	Yield %	7.8	26.6	44.5	60.7	76.9	23.1(>1.5)	-
	Ash %	2.6	4.3	5.5	-	-	-	-

APPENDIX 3.

Cumulative Yield and Ash of the Feed Calculated from the data in Appendix 2.

Test No.		Specific Gravity.					
		1.3	1.35	1.4	1.45	1.5	1.58
11B	Yield%	8.6	39.5	58.4	71.1	78.9	86.9
	Ash%	3.0	5.3	6.6	7.7	8.6	9.9
17C	Yield%	9.0	27.3	48.3	68.3	76.9	-
	Ash%	3.4	5.0	6.5	8.0	8.9	-
17H	Yield%	14.2	33.8	54.6	74.8	82.0	-
	Ash%	3.4	4.9	6.3	7.7	8.4	-
18C	Yield%	13.9	31.0	49.8	67.1	-	-
	Ash%	3.5	4.9	6.3	7.5	-	-
19C	Yield%	13.3	31.4	50.8	68.9	-	-
	Ash%	3.6	5.2	6.4	7.7	-	-
20D	Yield%	12.4	30.8	50.3	66.4	-	-
	Ash%	3.4	5.1	6.4	7.6	-	-
20G	Yield%	12.0	31.4	52.3	69.0	-	-
	Ash%	3.3	5.0	6.4	7.7	-	-
21A	Yield%	13.0	29.7	48.0	66.9	79.6	87.9
(-¼"+100#)	Ash%	-	-	-	-	-	-
21B	Yield%	12.6	30.7	48.4	69.0	79.2	88.5
(-¼"+100#)	Ash%	-	-	-	-	-	-
21G	Yield%	12.4	27.2	46.6	66.6	79.6	88.7
(-¼"+100#)	Ash%	-	-	-	-	-	-
21H	Yield%	12.8	29.6	44.4	67.2	79.7	88.8
(-¼"+100#)	Ash%	-	-	-	-	-	-
26C	Yield%	14.2	32.7	49.3	65.7	78.2	86.7
	Ash%	2.6	4.0	5.2	6.1	-	-
28A	Yield%	6.8	23.9	42.7	62.2	75.7	-
	Ash%	3.2	4.8	6.2	7.6	-	-
28B	Yield%	8.8	26.9	42.9	68.2	-	-
	Ash%	-	-	-	-	-	-
29A	Yield%	9.5	29.4	47.4	69.3	82.9	91.3
	Ash%	4.1	5.1	6.2	-	-	-
29D	Yield%	7.8	24.7	43.6	65.1	-	-
	Ash%	3.2	4.7	6.0	-	-	-
29F	Yield%	8.0	26.9	48.2	66.2	79.5	-
	Ash%	3.2	4.8	6.2	-	-	-
29G	Yield%	12.2	29.5	45.9	68.2	82.3	-
	Ash%	-	-	-	-	-	-
29H	Yield%	10.7	32.4	50.3	73.1	86.2	-
	Ash%	-	-	-	-	-	-
29I	Yield%	10.8	31.0	49.1	71.2	86.0	-
	Ash%	-	-	-	-	-	-
31A	Yield%	16.4	32.1	47.8	67.0	77.6	85.7
	Ash%	2.6	3.9	4.9	-	-	-
31B	Yield%	15.4	33.1	48.2	66.6	79.3	86.9
	Ash%	2.6	3.9	5.0	-	-	-
31C	Yield%	18.0	34.7	48.8	66.6	78.9	88.2
	Ash%	2.7	4.0	4.9	-	-	-
31D	Yield%	17.2	33.8	47.2	65.9	78.3	86.4
	Ash%	2.6	4.0	5.0	-	-	-
31F	Yield%	16.0	31.8	43.2	63.7	76.3	85.9
	Ash%	2.7	4.1	5.1	-	-	-
31T	Yield%	16.1	33.2	46.3	68.3	80.5	87.0
31U	Yield%	15.0	31.2	47.5	60.4	76.9	85.0
31W	Yield%	15.7	33.7	47.1	65.5	79.6	87.8
31X	Yield%	17.0	33.3	48.4	63.8	79.2	88.7
31Y	Yield%	14.9	33.7	50.2	64.4	78.1	89.2
31Z	Yield%	17.2	34.5	49.1	65.2	77.1	88.6
31AA	Yield%	13.5	28.5	46.4	63.9	76.0	86.3

APPENDIX 3 Continued.

Test No.		Specific Gravity					
		1.3	1.35	1.4	1.45	1.5	1.58
32A	Yield%	11.6	29.7	55.0	69.2	78.8	85.5
	Ash%	3.1	6.6	10.7	13.0	14.7	-
33A	Yield%	12.2	26.0	38.4	51.6	58.7	69.6
	Ash%	3.0	6.1	9.1	12.3	14.0	16.8
34A	Yield%	12.0	28.7	43.4	58.4	69.3	80.0
	Ash%	3.0	6.2	9.2	12.3	14.5	16.7
35A	Yield%	9.6	20.8	35.8	46.7	55.6	63.3
	Ash%	2.7	6.1	10.3	12.8	14.9	16.8
47B	Yield%	19.0	32.7	47.6	67.9	78.1	87.1
	Ash%	-	3.9	5.2	6.3	-	-
47F	Yield%	16.5	33.5	48.0	69.3	79.2	89.9
	Ash%	2.4	3.8	4.8	-	-	-
48I	Yield%	16.0	33.4	46.5	66.4	78.3	89.1
	Ash%	-	4.2	5.1	6.2	-	-
50G	Yield%	7.9	25.3	46.6	67.4	81.8	92.3
	Ash%	-	-	-	-	-	-
51C	Yield	6.3	28.6	49.0	69.3	83.6	92.9
	Ash%	3.6	5.2	6.3	-	-	-
61B	Yield%	7.8	27.0	45.5	61.0	77.8	-
	Ash%	3.1	4.6	5.9	-	-	-
70A	Yield%	10.0	28.1	48.6	66.2	80.4	90.0
70C	Yield%	12.1	29.8	50.6	69.2	82.3	-
70E	Yield%	13.5	32.2	51.4	69.9	82.3	-
70G	Yield%	14.3	33.8	54.5	72.5	-	-
71A	Yield%	10.4	26.1	45.5	65.0	77.7	-
71C	Yield%	9.8	27.6	48.3	67.6	80.5	-
71E	Yield%	12.4	29.4	51.4	69.9	82.3	-
72C	Yield%	10.0	31.5	48.6	67.3	80.1	90.0
72G	Yield%	9.6	33.5	52.0	71.6	82.0	91.4
73A	Yield%	12.9	32.2	48.6	68.1	78.9	89.4
73E	Yield%	14.2	34.5	50.5	69.1	80.2	89.7
73H	Yield%	14.5	38.5	58.3	74.3	84.3	-
74A	Yield%	12.3	34.6	53.9	68.8	81.8	92.0
74D	Yield%	9.4	33.6	54.8	68.8	82.2	92.1
75D	Yield%	10.6	33.0	51.9	67.5	80.7	89.1
75F	Yield%	13.7	36.4	55.5	71.1	83.1	90.7
76A	Yield%	18.2	37.3	53.4	68.3	78.5	93.9
	Ash%	2.8	4.0	5.4	6.3	-	-
76C	Yield%	16.4	33.6	50.2	67.1	78.8	89.7
	Ash%	-	-	-	-	-	-
76E	Yield%	17.2	35.7	50.7	69.4	77.9	89.8
	Ash%	2.5	3.8	5.4	6.6	-	-
80C	Yield%	12.5	32.1	49.5	67.7	-	-
	Ash%	3.2	4.8	6.0	-	-	-
80E	Yield%	14.5	34.7	52.6	69.1	-	-
	Ash%	3.3	4.8	6.0	-	-	-
80D	Yield%	13.0	33.0	49.3	68.2	80.1	-
	Ash%	3.2	4.8	5.9	-	-	-

APPENDIX 3 Continued.

Test No.		Specific Gravity. 1.3	1.35	1.4	1.45	1.5	1.58
80F	Yield%	13.5	36.1	54.0	68.9	-	-
	Ash%	3.2	4.9	6.0	-	-	-
80G	Yield%	15.6	38.6	56.3	70.6	-	-
	Ash%	3.4	5.1	6.2	-	-	-
80H	Yield%	15.9	37.9	57.1	72.3	83.8	-
	Ash%	3.1	4.8	5.9	-	-	-
84A	Yield%	14.9	34.6	47.4	70.3	-	-
	Ash%	3.6	4.9	5.8	-	-	-
85A	Yield%	13.1	38.9	55.6	73.2	84.6	-
	Ash%	2.5	3.9	-	-	-	-
85E	Yield%	11.4	36.2	53.9	69.9	83.0	-
	Ash%	2.4	3.8	5.1	-	-	-
86A	Yield%	12.5	35.5	53.2	70.0	82.2	89.2
	Ash%	2.4	3.9	-	-	-	-
86B	Yield%	15.2	34.8	53.5	67.9	81.3	88.0
	Ash%	2.4	3.8	5.2	-	-	-
87A	Yield%	14.8	33.3	49.8	64.6	79.2	-
	Ash%	2.5	4.2	5.4	-	-	-
87E	Yield%	17.9	40.8	54.5	71.3	82.0	-
	Ash%	2.5	3.9	5.9	-	-	-
88B	Yield%	13.8	32.2	50.6	-	-	-
	Ash%	3.3	4.7	5.9	-	-	-
90A	Yield%	13.7	34.7	50.7	71.4	-	-
	Ash%	3.6	4.9	5.9	-	-	-
91A	Yield%	6.8	19.8	61.4	82.0	91.5	-
(-½"+100#)	Ash%	3.3	4.5	6.3	7.5	8.4	-
93C	Yield%	6.6	20.1	60.5	80.4	90.3	-
(-½"+100#)	Ash%	2.8	4.3	6.1	7.2	8.2	-
95A	Yield%	18.8	36.4	56.0	70.0	-	-
	Ash%	3.7	4.9	6.2	-	-	-
95B	Yield%	19.1	35.8	59.6	73.4	-	-
	Ash%	3.8	4.9	6.4	-	-	-
95C	Yield%	18.5	36.7	64.8	74.3	-	-
	Ash%	3.5	4.8	6.3	-	-	-
95D	Yield%	19.1	35.9	61.8	74.2	-	-
	Ash%	3.8	4.9	6.5	-	-	-
95E	Yield%	20.9	38.5	59.7	73.4	-	-
	Ash%	3.9	5.1	6.4	-	-	-
95F	Yield%	20.7	39.1	60.5	74.7	-	-
	Ash%	3.9	5.1	6.5	-	-	-
96B	Yield	12.1	34.8	51.6	69.4	-	-
	Ash%	3.4	5.0	6.1	7.2	-	-
96C	Yield%	15.1	35.9	55.0	74.3	-	-
	Ash%	3.5	5.0	6.3	-	-	-
96D	Yield%	14.7	38.1	56.3	72.3	-	-
	Ash%	3.4	5.0	6.2	-	-	-
96E	Yield%	8.6	36.9	58.2	72.0	-	-
	Ash%	3.1	5.0	6.2	-	-	-
96F	Yield%	12.8	36.7	59.8	75.2	-	-
	Ash%	3.2	4.8	6.5	-	-	-
99B	Yield%	14.6	33.2	55.3	71.9	86.5	-
(-½"+100#)	Ash%	3.2	4.8	6.2	7.5	-	-
100C	Yield%	14.5	33.7	54.0	70.7	83.2	-
(-½"+100#)	Ash%	3.5	4.8	6.2	7.4	8.6	-
101A	Yield%	10.8	29.8	50.0	75.4	88.2	-
(-½"+100#)	Ash%	3.7	4.9	6.1	7.5	8.5	-
102B#	Yield%	9.6	28.2	50.1	75.3	87.8	-
(-½"+100	Ash%	3.5	4.9	6.3	7.6	8.7	-

APPENDIX 4

Tromp Distribution Factors Calculated from the data in Appendix 2

Test No.	Specific Gravity					
	<1.3	1.3-1.35	1.35-1.4	1.4-1.45	1.45-1.5	1.5-1.58
11B	98.7	97.8	89.6	41.3	15.3	6.2
17C	89.0	79.0	21.0	5.5	4.9	3.7
17H	83.0	43.0	4.8	3.2	3.2	2.3
18C	96.0	75.0	17.5	4.3	5.2	0
19C	96.0	75.0	12.9	4.1	2.5	0
20D	100.0	83.0	29.0	4.6	1.7	0
20G	100.0	90.0	53.0	6.6	1.2	0
21A $(-\frac{1}{4}"+100\#)$	94.0	89.0	39.4	12.0	6.8	5.0
21B $(-\frac{1}{4}"+100\#)$	94.4	88,8	43.3	8.1	6.4	3.1
21G $(-\frac{1}{4}"+100\#)$	91.0	73.1	35.7	12.2	11.6	6.3
21H $(-\frac{1}{4}"+100\#)$	91.4	71.4	22.2	6.6	7.6	3.1
26C	87.7	89.0	46.5	17.7	7.7	6.8
28A	90.0	84.0	50.0	5.1	0.7	0
28B	95.5	82.2	15.6	0.4	0	0
29A	88.5	68.0	44.0	31.0	16.2	9.5
29D	93.0	78.0	7.9	1.4	0	0
29F	99.0	91.0	32.0	2.8	2.3	0
29G	100.0	94.0	62.0	5.8	0.7	0
29H	100.0	98.0	95.0	33.0	0.8	0
29I	100.0	100.0	97.0	73.0	10.1	0
31A	85.7	64.2	35.3	13.9	6.6	2.5
31B	88.7	80.8	40.1	13.5	7.9	4.0
31C	90.0	78.4	43.3	11.8	8.9	4.3
31D	86.1	74.0	47.0	13.9	7.3	3.2
31F	68.2	46.2	21.0	6.3	4.8	2.1
31T	88.0	79.4	50.6	17.7	11.6	10.7
31U	91.2	80.2	41.0	13.8	6.6	6.4
31W	92.0	88.0	67.0	25.0	11.3	1.2
31X	92.4	88.3	60.0	26.0	6.5	10.5
31Y	94.0	90.0	66.0	26.0	11.0	2.7

APPENDIX 4 Continued.

Test No.	Specific Gravity					
	<1.3	1.3-1.35	1.35-1.4	1.4-1.45	1.45-1.5	1.5-1.58
31Z	96.0	92.5	74.0	20.0	10.9	0.9
31AA	57.0	48.5	31.8	27.5	24.0	17.5
32A	91.0	83.5	48.5	5.2	3.6	0
33A	96.7	89.2	58.0	6.8	4.2	2.4
34A	96.9	91.6	61.2	13.7	2.8	3.7
35A	96.4	89.8	54.6	11.1	3.0	2.7
47B	90.5	82.1	61.3	45.7	16.2	12.2
47F	91.2	83.0	53.9	21.7	14.7	9.2
48I	80.8	78.2	61.0	54.9	38.3	35.9
50G	80.9	68.7	49.1	25.5	15.8	9.1
51C	93.7	85.3	58.0	26.0	4.7	0
61B	84.8	61.3	11.8	2.2	0.8	0
70A	86.8	66.5	51.6	25.1	6.8	2.9
70C	91.5	71.0	56.4	19.1	5.5	0
70E	92.7	71.2	34.4	6.8	1.1	0
70G	90.9	57.4	16.5	2.0	0	0
71A	73.6	55.3	38.0	8.8	4.7	0
71C	79.6	60.0	22.1	6.8	3.9	0
71E	84.3	57.8	17.4	5.6	1.4	0
72C	73.0	48.4	34.7	23.3	13.7	9.3
72G	87.7	73.3	54.9	45.4	29.0	22.1
73A	79.8	67.4	59.8	53.2	44.3	33.5
73E	85.8	68.2	61.7	49.5	37.3	22.0
73H	77.9	43.0	19.7	8.2	4.9	0
74A	57.2	46.4	22.8	12.1	8.1	3.4
74D	66.5	62.4	34.2	19.7	17.1	4.1
75D	69.7	52.2	36.7	28.9	12.3	7.9
75F	76.1	67.8	48.0	35.7	22.6	14.0
76A	81.4	72.1	59.6	46.0	41.0	21.7
76C	74.7	43.5	24.1	4.3	0.7	0.5
76E	76.3	67.0	68.3	46.3	39.6	38.1
80C	92.9	75.6	28.8	2.5	0	0

APPENDIX 4 Continued.

Test No.	Specific Gravity.					
	<1.3	1.3-1.35	1.35-1.4	1.4-1.45	1.45-1.5	1.5-1.58
80E	96.1	85.7	29.5	4.1	0	0
80D	96.2	86.3	38.8	6.0	2.5	0
80F	95.5	88.9	29.2	4.7	0	0
80G	93.1	65.6	12.4	4.1	0	0
80H	97.2	92.3	41.4	6.9	3.9	0
84A	91.8	43.9	3.6	1.0	0	0
85A	77.5	65.1	23.3	2.5	2.8	0
85E	93.2	86.3	63.1	28.4	5.2	0
86A	84.9	70.1	27.9	11.2	6.8	4.0
86B	89.6	85.9	61.8	27.6	9.3	7.7
87A	52.6	8.7	2.9	1.4	0.8	0
87E	93.6	81.2	27.7	4.1	2.6	0
88B	92.5	38.1	1.1	0	0	0
90A	96.1	76.3	22.1	1.3	0	0
91A(-½"+100#)	91.5	70.8	13.9	2.7	1.4	0
91A(-½"+14#)	95.9	67.0	11.6	2.2	0.2	0
91A(-14#+48#)	83.9	83.1	25.6	2.9	4.9	
91A(-48#+72#)	71.4	63.2	51.5	19.4	10.4	0
91A(-72#+100#)	57.2	57.0	31.0	25.2	7.2	4.8
93C(-½"+100#)	95.5	80.6	31.6	6.4	1.3	0
93C(-½"+14#)	97.5	76.6	26.7	3.1	0.3	0
93C(-14#+48#)	92.8	89.8	56.2	10.6	2.1	0
93C(-48#+72#)	84.7	78.5	76.9	43.9	7.6	0
93C(-72#+100#)	86.0	81.7	75.6	57.0	27.1	12.6
95A	92.5	55.1	10.2	3.1	0	0
95B	96.4	80.4	24.6	2.8	0	0
95C	96.9	85.9	31.4	5.9	0	0
95D	96.3	86.3	35.1	4.1	0	0
95E	96.2	78.9	19.9	1.4	0	0
95F	90.8	52.7	3.7	0.8	0	0
96B	78.7	19.0	0.6	0.2	0	0
96C	85.0	28.1	1.1	0.1	0	0

APPENDIX 4 Continued.

Test No.	Specific Gravity Fractions.					
	<1.3	1.3-1.35	1.35-1.4	1.4-1.45	1.45-1.5	1-5-1.58
96D	96.2	62.1	11.3	0.6	0	0
96E	97.7	74.3	12.3	0.5	0	0
96F	96.6	60.9	2.4	0.4	0	0
99B (-½"+100#)	96.0	83.9	25.4	3.7	1.0	0
99B (-½"+14#)	98.3	85.1	21.2	1.4	0	0
99B (-14#+48#)	92.9	80.4	33.5	5.6	3.0	0
99B (-48#+72#)	91.7	85.3	71.4	14.7	7.0	0
99B (-72#+100#)	92.3	92.6	75.9	59.3	23.2	13.0
100C (-½"+100#)	96.4	93.5	65.3	12.0	3.4	0.8
100C (-½"+14#)	99.1	95.9	63.3	7.5	1.3	0
100C (-14#+48#)	92.8	89.4	69.4	20.9	9.1	2.7
100C (-48#+72#)	89.3	80.2	71.5	44.6	3.8	6.8
100C (-72#+100#)	83.3	71.2	71.6	24.6	24.8	6.5
101A (-½"+100#)	94.1	84.1	38.0	6.1	2.3	-
101A (-½"+14#)	96.7	84.0	32.5	2.7	1.2	0
101A (-14#+48#)	87.7	84.6	57.1	11.2	5.7	3.6
101A (-48#+72#)	85.8	83.4	73.6	48.7	2.2	7.0
101A (-72#+100#)	83.1	84.5	78.1	52.3	25.7	15.2
102B (-½"+100#)	94.7	93.4	81.5	33.2	5.1	1.3
102B (-½"+14#)	98.5	96.4	84.9	32.2	4.0	0
102B (-14#+48#)	86.8	83.2	66.8	33.2	8.1	2.6
102B (-48#+72#)	83.8	76.2	67.8	56.2	16.8	6.8
102B (-72#+100#)	78.4	65.5	58.2	60.4	19.1	4.2

APPENDIX 5.

SIEVE ANALYSES.

A Sieve analyses relating to Table 3.

Test No.		Fractional Yield, %			
		$-\frac{1}{4}$" + 1 mm	-1 mm + 52#	-52# + 72#	-72# +100#
21A	Product	59.0	30.1	5.4	5.5
	Tailing	74.2	19.1	3.7	3.0
21B	Product	61.4	33.8	2.9	1.9
	Tailing	74.6	22.3	2.1	1.0
21G	Product	60.0	33.0	4.1	2.9
	Tailing	78.0	18.5	2.2	1.3
21H	Product	61.2	29.7	5.9	3.2
	Tailing	74.4	20.0	3.5	3.1

52, 72 and 100 B.S.S.

B Sieve Analyses relating to Table 14.

Test No.		Cumulative Yield, %											
		$2\frac{1}{2}$#	3#	$\frac{1}{4}$"	4#	5#	6#	8#	10#	14#	20#	35#	48#
70A	Product	0.3	4.0	5.1	50.4	85.2	92.9	96.6	97.9	98.9	99.3	99.8	99.9
	Tail.	0.2	1.8	4.7	32.0	74.5	87.1	93.1	94.9	96.7	97.7	98.9	99.3
70C	Prod.	0.1	1.1	1.3	34.3	74.8	85.4	91.7	94.1	97.0	98.2	99.7	99.8
	Tail.	0.0	0.2	0.3	22.9	66.2	77.9	86.6	90.4	93.8	95.7	98.1	99.0
70E	Prod.	0.0	0.4	0.5	22.8	67.8	78.2	86.3	89.0	93.7	96.2	99.6	99.8
	Tail.	0.0	0.1	0.1	13.8	49.8	64.7	76.0	81.4	87.1	90.7	95.6	97.6
70G	Prod.	0.0	0.1	0.1	13.1	43.5	58.2	72.8	80.3	91.5	94.2	99.6	99.8
	Tail.	0.0	0.1	0.1	5.1	44.7	59.8	74.8	81.6	88.7	92.4	96.8	98.2
71A	Prod.	-	-	-	5.8	31.6	44.1	65.1	78.8	94.5	97.2	99.5	99.8
	Tail.	-	-	-	3.5	27.1	41.7	62.3	75.9	92.6	96.5	98.6	99.0
71C	Prod.	-	-	-	3.3	25.5	38.6	57.8	69.3	85.5	91.7	96.4	97.7
	Tail.	-	-	-	1.9	22.2	37.2	60.1	72.6	88.6	94.4	97.9	98.7
71E	Prod.	-	-	-	0.6	12.5	23.0	44.4	54.4	71.0	79.5	89.0	92.2
	Tail.	-	-	-	1.1	12.8	27.5	53.8	67.5	85.4	92.6	97.3	98.7
74A	Prod.	-	-	-	-	0.5	15.7	47.4	66.3	92.1	97.1	99.6	99.9
	Tail.	-	-	-	-	0.3	11.5	44.3	66.8	89.7	97.1	99.2	99.6
74D	Prod.	-	-	-	16.8	76.6	89.0	95.3	97.2	98.5	99.1	99.8	99.9
	Tail.	-	-	-	14.3	73.0	86.7	93.8	95.9	97.7	98.6	99.6	99.8

$2\frac{1}{2}$, 3, 4, 14, 20, 35, 48 mesh Tyler sieves.
$\frac{1}{4}$", 5, 6, 8, 10, mesh B.S.S.

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Test No.		Cumulative Yield, %								
		¼"	5#	8#	10#	14#	24#	35#	60#	100#
47B	Product	-	-	-	-	1.6	29.2	59.2	82.8	98.1
	Tailing	-	-	-	-	3.2	48.2	73.2	91.0	98.7
50G	Product	-	76.5	91.9	94.5	-	97.5	98.0	98.7	99.6
	Tailing	-	57.1	87.8	91.5	-	96.9	98.3	99.2	99.8
51C	Product	22.3	86.3	92.2	93.4	-	96.1	97.1	98.1	99.6
	Tailing	8.9	67.9	84.4	86.6	-	94.3	96.8	98.3	99.6
52A	Product	22.6	87.6	93.3	94.3	-	95.9	96.8	97.8	99.4
	Tailing	9.9	70.1	87.6	90.1	-	95.7	97.5	98.7	99.6
61B	Product	-	40.7	65.0	75.2	-	94.0	96.0	97.6	99.7
	Tailing	-	33.0	65.4	76.4	-	97.4	98.9	99.5	99.9

14, 35 mesh Tyler, 24 mesh sieve aperture = 0.0276"
¼", 5, 8, 10, 60, 100 mesh B.S.S.

C. <u>Sieve Analyses relating to Table 17</u>.

Test No.		Cumulative Yield, %											
		3/8"	2½#	3#	4#	5#	6#	8#	10#	14#	48#	72#	100#
91A	Prod.	3.9	9.2	16.8	34.2	44.5	49.8	56.7	62.0	69.0	95.8	98.5	99.4
	Tail.	1.2	2.9	6.3	29.7	47.0	55.4	65.8	72.9	81.4	97.8	99.0	99.6
93C	Prod.	2.1	3.5	7.5	20.4	34.7	41.8	50.6	57.1	65.5	93.4	96.6	98.7
	Tail.	0.8	1.8	6.1	22.8	43.4	53.3	64.6	72.4	81.0	97.5	99.0	99.7

2½, 3, 4, 14, 48 mesh Tyler sieves.
5, 6, 8, 10, 72, 100 mesh B.S.S.

D. <u>Sieve Analyses relating to Table 26</u>.

Test No.		Cumulative Yield, %											
		3/8"	2½#	3#	4#	5#	6#	8#	10#	14#	48#	72#	100#
93C	Prod.	2.1	3.5	7.5	20.4	34.7	41.8	50.6	57.1	65.5	93.4	96.6	98.7
	Tail.	0.8	1.8	6.1	22.8	43.4	53.3	64.6	72.4	81.0	97.5	99.0	99.7
99B	Prod.	0.1	0.2	1.0	10.0	25.5	34.1	45.2	54.1	64.6	93.2	96.1	99.8
	Tail.	0.1	0.1	0.7	14.5	33.2	42.9	55.4	64.4	74.9	97.0	98.5	99.6
100C	Prod.	0.0	0.1	0.7	10.5	27.9	36.7	48.0	56.6	67.0	93.9	96.9	99.1
	Tail.	0.0	0.1	0.6	12.7	32.8	42.4	54.5	63.1	72.7	94.1	96.9	99.5
101A	Prod.	1.4	3.0	6.9	24.1	38.7	45.8	54.8	61.6	70.4	91.6	95.7	98.8
	Tail.	0.1	1.0	5.0	23.2	43.9	53.4	64.6	72.3	80.8	96.1	98.1	99.5
102B	Prod.	0.2	1.0	3.7	23.8	43.2	52.2	63.1	70.6	80.0	95.7	97.5	99.0
	Tail.	0.9	2.7	6.6	22.8	40.5	48.1	57.6	64.7	73.8	94.5	96.9	99.3

2½, 3, 4, 14, 48 mesh Tyler sieves.
5, 6, 8, 10, 72, 100 mesh B.S.S.

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