OPTIMUM PROCESSING OF 1 MM BY ZERO COAL

Dennis I. Phillips

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R.-H. Yoon, PhD, Chair
G. H. Luttrell, PhD
G. T. Adel, PhD
A. L. Wicks, PhD
E. R. Palowitch, PhD

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(ABSTRACT)

Coal in the finer particle size ranges (below 1 mm) has always suffered from poor cleaning efficiencies. This problem has been exacerbated in recent years with the increased amount of high ash fines due to continuous mining machines and the mining of dirtier coal seams. In the present work, it is proposed to improve overall plant efficiencies by processing coarser coal in column flotation than is now commonly treated by that method. Column flotation for coarse coal is supported by actual lab and plant test data that result in a full-scale column plant installation. The fundamentals of coarse particle detachment from bubbles are reviewed and a new simplified model is developed which better handles cubical and rectangular coal particles.

Much of the lower efficiency of fine coal cleaning is due to poor size separation of the fine-sized raw coal which results in misplaced high ash fines reporting to the coarser size streams. By sending coarser material to column flotation, the finest size separation that takes place in a plant can be as coarse as 0.5 mm or greater. The proper use of wash water in a flotation column then becomes the best mechanism for desliming of the high ash clays. This work quantifies the benefits of removing the high ash fines from the plant product and increasing overall plant yield by increasing the amount of near-gravity coarse material. The resulting yield gain is greater than that obtained from only the increased fine coal recovery. Methods of column operation for improved coarse coal recovery are also evaluated.

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CHAPTER 1 INTRODUCTION

1.1 Fine Coal Processing

Although much research regarding improvements in fine coal cleaning has been performed in the last twenty years, the processing of material smaller than 1 mm remains difficult and results in very poor efficiencies when compared to those of coarser sizes. While some cleaning devices have higher efficiencies than others, they often involve greater economic expense, whether real or perceived. Not only is there little consensus in the coal industry as to the overall best cleaning methods for particles finer than 1 mm, there is little agreement as to what particle size ranges are best suited for each of the devices employed.

For any coal cleaning device, separation efficiency is reduced as the particle size becomes smaller. A major problem that is seldom properly addressed is that of high ash fine particles (slimes) reporting to a coarser size stream from which they are not properly removed or cleaned. This problem of misplaced sizes is often the source of much of the high ash and low yields found in fine coal cleaning.

The nomenclature of coal sizes varies but a general guide and one that will be used in this work is:

Coarse	+10 mm	(plus 3/8 inch)
Intermediate	10 x 0.5 mm	(3/8 inch x 28 mesh)
Fine	0.5 (or 1.0) mm x 0	(< 28 mesh)
Ultrafine	0.150 mm x 0	(100 mesh x 0)

History of Fine Coal Processing

Fine coal processing on even a moderate scale had its beginnings as recently as the 1940’s and 1950’s. Some of the common methods and sizes for coal processing are illustrated in Figure 1-1, but the actual particle size at which one processing method gave way to another was never well defined and continues to vary from one company to another.

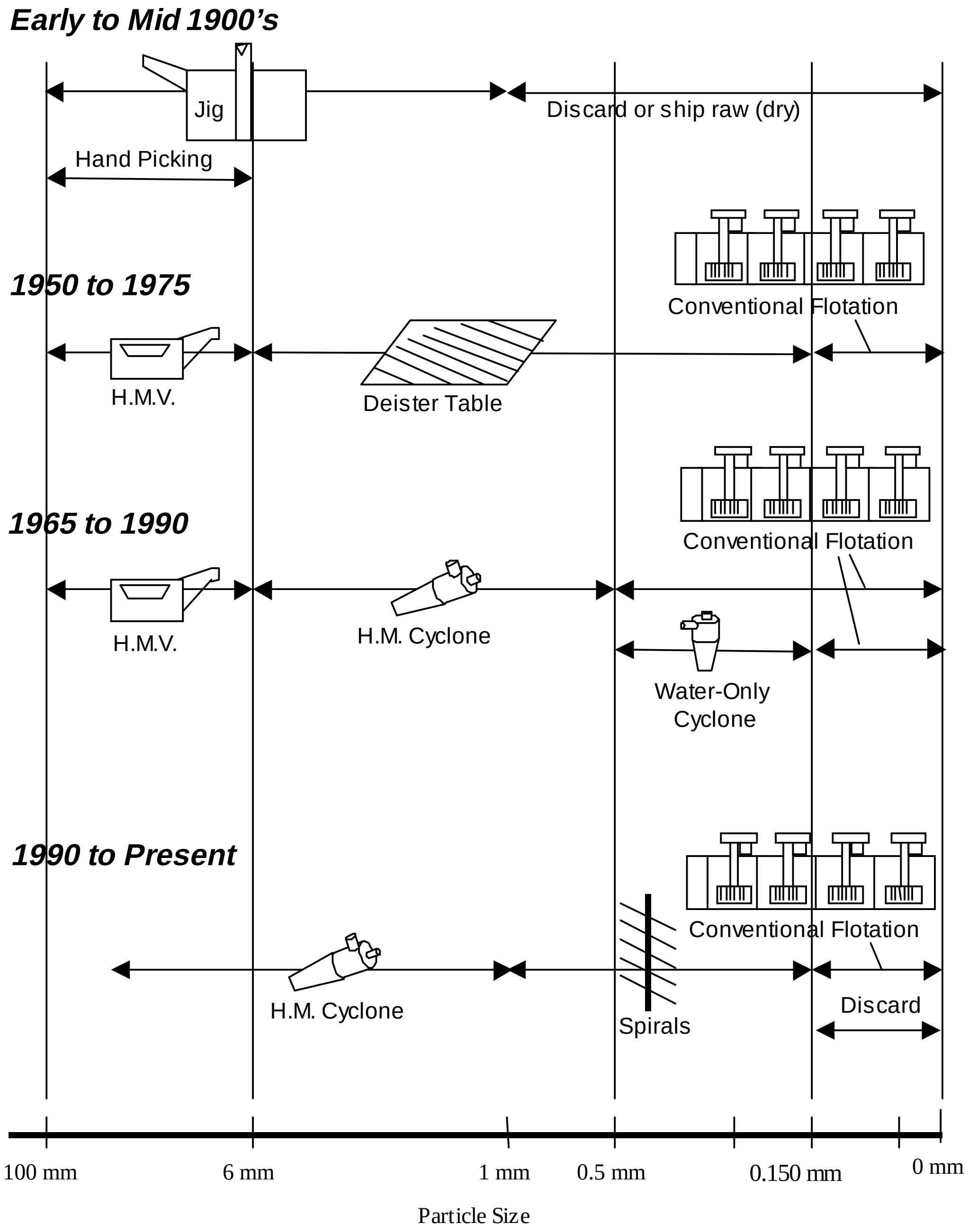

Figure 1-1. Common coal processing methods and sizes.

Coarse coal was formerly often cleaned in a jig but is now most commonly cleaned in a heavy media vessel. It is becoming increasingly popular to simplify circuitry by cleaning the coarse size along with the intermediate size in a heavy media cyclone. The intermediate sizes were formerly cleaned in jigs or Deister tables but are now almost totally processed in heavy media cyclones. In the distant past fine sizes were often discarded and in later years were cleaned on either Deister tables or in water-only cyclones. Since heavy media cyclones replaced Deister tables for the intermediate sizes, it was desirable to use a separate cleaning device for just the fine size in order to reduce magnetite consumption in the HMC circuit. For this reason as well as their low operating costs, spiral concentrators (spirals) were chosen even though their efficiency is similar to that of Deister tables on the minus 1 mm coal (Deurbrouck and Palowitch 1963). The ultrafine sizes were historically shipped raw and dry or were discarded. Today the ultrafines are still discarded in many coal plants producing power plant fuel (steam coal), although some steam coal plants and most metallurgical (coking coal) plants use flotation since it remains the only method suitable for cleaning the ultrafine material. Conventional cell flotation (a trough shaped tank with agitators) is the type of flotation most often employed, although column flotation has gained a few operations in recent years.

1.2 Current Industry Problems

Processing coal smaller than 1 mm in size has always been more difficult than cleaning coarser sizes and remains so today. Even though several devices such as spirals and heavy media cyclones can process these sizes with reasonable efficiencies on the plus 0.150 mm fraction, the overall circuit suffers due to insufficient removal of the ultrafine fraction. A common circuit today consists of sending minus 1 mm material to a classifying cyclone with the overflow (nominal 0.150 mm x 0) going to either flotation or discard. The underflow of the cyclone is then cleaned in spirals. There are three main problems with this circuit. One is that the spirals actually clean only a narrow size range well and perform poorly on anything smaller than 0.25 mm (Martin 1994). The second problem is the high separating gravity at which a spiral makes a separation, usually above a 1.80 SG cut-point. The resulting higher ash product in effect requires compensation by lowering of the separating gravities on the coarser circuits, resulting in lost yield.

Ultrafine material that is misplaced to the underflow of the classifying cyclone creates the third problem with this common circuit. This misplacement can be as much as 20% of the ultrafines (minus 0.150 mm) feeding the cyclone. Nearly all of the ultrafines feeding a spiral report to the clean coal product of the unit. Usually some attempt is made to further deslime the clean coal stream such as the use of sieve bends or even, occasionally, an additional classifying cyclone circuit. Even with this additional desliming, there is often sufficient fines of high ash to cause a considerable rise in the product ash. The author has seen 0.5 x 0.150 mm products have a 14% ash where a 10% ash was expected. The increased ash was due to the presence of minus 0.150 mm material with an ash of over 40%.

Table 1-1 shows the clean coal data from an actual plant test (Phillips 1992) on a relatively new plant having heavy media cyclones and spirals with the minus 0.150 mm (100 mesh) discarded. In this sample, the plus 0.5 mm was 8% ash while the total spiral product (all the minus 0.5 mm) was 20.6% ash. Without the misplaced minus 0.150 mm material, the spiral product would have been 17% ash, indicating the poor performance of the spirals. The misplaced minus 0.150 mm material had an ash of 30.22% and even though it represented only 3.2% of the total product, it contributed enough ash to change the total clean coal from 8.8% ash to 9.5%. This problem is found to be worse with coal feeds having high clay contents which also cause inconsistent product ashes due to the variations in feed clay from hour to hour. Truly proper sizing at 0.150 mm cannot be economically achieved with today's sizing equipment (Firth et al. 1995).

Table 1-1. Beech Plant total clean coal.

Size (mm)	Individual		Cumulative Retained		Cumulative Passing	
	Wt. %	Ash %	Wt. %	Ash %	Wt. %	Ash %
+1	81.3	7.96	81.3	7.96	100.0	9.48
0.5 x 1	6.7	8.12	88.0	7.97	18.7	16.11
0.150 x 0.5	8.8	17.06	96.8	8.80	12.0	20.57
0.045 x 0.150	2.1	26.22	98.9	9.17	3.2	30.22
0 x 0.045	1.1	37.86	100.0	9.48	1.1	37.86

1.3 Potential Improvements to Fine Coal Processing

As previously discussed, cleaning coal finer than 1 mm and especially finer than 0.5 mm is difficult due to poor efficiencies of common devices and due to the high separating gravities for water based devices such as spirals. The complications added by poor desliming (removing fine sized particles) increase the problems and inconsistencies. A new device known as column flotation may provide some or most of the answer to the problems of cleaning fine coal. The limited locations in which columns have been applied in the coal industry, have been almost exclusively on material finer than 0.150 mm. By applying flotation columns to coarser sizes of 0.25 mm, or 0.5 mm, or even 1 mm, many of the previously mentioned problems can be overcome. For instance, if 0.5 mm x 0 material can be processed in flotation columns, there is no need to make any size cuts near 0.150 mm. When properly applied, the wash water in a column is the best desliming system available (Luttrell et al. 1991). Size cuts of 0.5 mm can be made very well on vibrating screens and to a slightly lesser extent on sieve bends. With these screening devices, unlike classifying cyclones, there is very little misplacement of fines to the coarse stream. These devices perform a much more positive size separation.

Since heavy media cyclones perform well and have no equal on coal greater than 0.5 or 1 mm, it remains to optimize and simplify the fines circuits. As mentioned previously, the current standard is the use of spirals for the 1 x 0.150 mm fraction and conventional flotation for the material finer than 0.150 mm. If columns are to be applied for a coarser flotation size, then there are two options to consider for improving the processing of coal finer than 1 mm. Option 1 is to use spirals for the coal from 1 mm down to some size such as 0.5 or 0.25 mm and use column flotation for all material finer than the lower size. Option 2 would involve heavy media cyclones down to 0.75 or 0.5 mm and column flotation on the material finer than that.

Option 1 assumes that the practical upper limit for flotation is 0.5 or 0.25 mm and that spirals will still be utilized above that. This circuit should be much simpler and more efficient than current practice due to the ease with which a coarser classification size cut can be made. By narrowing the size range feeding the spirals, they will also be more efficient.

Option 2 is obviously desirable due to its simplicity. If the amount of coal above 50 mm (2 inch) in size is insufficient to justify a heavy media vessel, then a plant can consist of only heavy media cyclones above some size and column flotation on the finer size. This two circuit plant could conceivably have a very high efficiency on all sizes down to zero.

Coarse Coal Flotation Not New

Although very few plants are currently using flotation for particle sizes up to 0.5 mm, it was actually relatively common to use conventional flotation cells on coal this coarse in the late 1960’s and 1970’s. It lost favor, however, during the 1980’s due to the belief that it was too difficult to float the coarser, plus 0.25 mm, coal. The author believes that the coarser coal was cleaned well in the earlier years, but that during the later 1970’s and early 1980’s the increased use of continuous mining machines and the mining of dirtier coal seams brought two changes to the flotation systems:

1. Increased amounts of fines which literally overloaded the carrying capacity of the cells. As will be shown in later chapters, this overload resulted in the coarser particles being lost from the bubble due to the preferential loading of the finer sizes.

2. Increased fine ash resulted in higher flotation feed ashes. The ultrafine ash is entrained in the froth normally and even more so when the cells are “pulled hard” to recover the coarse particles. Although most operators did not realize that the entrainment was the main problem, they did understand that by using less frother and drying up the froth, the product ash would drop. This actually allowed some of the entrained ash to drain from the froth but it also drastically reduced the froth’s ability

to carry coarse particles due to froth overload and due to a froth that was too dry and immobile.

After a few years of cells operating in this way, a reputation was developed that flotation cells could not really handle the coarser flotation sizes. It is the author's opinion that coarser coal can be floated well in a properly designed system and that with the use of wash water in a flotation column, the problems of earlier years can be corrected.

1.4 Scope of Research

To prove that improvements in coarse coal flotation can be utilized in ways such as those shown as Options 1 and 2 of the previous section, requires more understanding of the fundamentals involved. To be of benefit to industry, these coarse coal flotation improvements and circuit recommendations must be shown to be better than current practice. To further the fundamental understanding of potential processing improvements, this research has the following goals:

1. To understand the fundamentals of coarse particle flotation and detachment.

2. To show that flotation columns can process coal above 0.150 mm as well or better than current methods. This requires actual tests with a column on coarse coal.

3. To investigate and recommend techniques for improving the flotation of coarse coal.

4. To investigate how a column can fit into the overall plant circuit and should include not just the efficiency of cleaning but also the interrelationships of misplaced sizes.

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CHAPTER 2 FUNDAMENTALS OF COARSE PARTICLE DETACHMENT

2.1 Introduction

In recent years much has been learned about the fundamentals of bubble particle attachment (Jowett 1980). From those studies it has been shown that coarse particles have higher probabilities of attachment to bubbles than do finer particles and thus should result in high flotation recovery. This is often not the case since coal particles finer than 0.150 mm (100 mesh) generally have the highest recovery. From observation of a transparent laboratory flotation cell or column cell, one can observe coarse particles, attached to bubbles, rising through the pulp zone and then becoming detached during or shortly after emergence through the interface between the pulp and the froth zone. A concentration of particles can be observed near the interface and falling downward into the pulp zone. The problem with coarse particle flotation is not the ability to attach coarse particles to bubbles, but the particle's difficulty in remaining attached to the bubble throughout the froth zone and thus flow into the flotation cell launder.

What force is required to detach a particle from a bubble? What is the difference in magnitude for this force between fine particles and coarse particles? These questions will be investigated in this chapter.

2.2 Current Detachment Models

There are many dynamic circumstances under which a coarse particle can become detached. A rising bubble has streamlines of liquid flowing past it which exert a stress on the particle. Turbulence within the flotation system can cause liquid flows that may literally rip the particle from the bubble, especially near the agitator of a conventional flotation cell. As the bubble rises through the pulp-froth interface, the energy released from expanding and coalescing bubbles may break the bond of attachment.

Many factors such as hydrodynamics of bubble particle collision, film thinning time, and actual attachment forces are involved with the probability and forces of bubble particle attachment. True measurement of all the attachment forces remains a difficult task. It should however, be possible to measure or in some way quantify the force required to remove a coarse particle from a bubble surface. By looking at the fundamental forces involved in the detachment of a particle from a bubble or other gas interface, one can begin to understand the parameters that cause or prevent detachment. By utilizing models of these detachment mechanisms, one can see the relative importance of each contributing factor.

Yoon and Mao (1996) showed that at the level of basic forces, the probability of detachment (P_d) can be shown as

$$P_d = \exp\left(-\frac{W_a + E_1}{E'_k}\right) \qquad \text{Eq. 2-1}$$

in which a particle can be detached from a bubble if the kinetic energy (E'_k) pulling on the particle is greater than the sum of the work of adhesion (W_a) and the energy barrier (E_1) (Laskowski 1989) (Laskowski, Xu and Yoon 1992). The exponential function was used because the multiple small particles on the bubble surface would have distributed E'_k values.

Morris (1950) was one of the first to attempt to actually measure the particle detachment force on a visual scale. Small paraffin coated cylinders were attached to large bubbles in water. The static contact angle was measured. As long as the particle remained attached, this angle was always less than the maximum contact angle. The heavier particles produced angles closer to the maximum angle and thus were closer to the point of detachment. By equating upward and downward forces at static equilibrium, the following equations allowed for the verification of test measurements.

$$P\frac{d^2\pi}{4} + w = d\pi\gamma\,\mathrm{Sin}\;\theta + (h + R + l)\rho_L\frac{d^2\pi}{4} \qquad \text{Eq. 2-2}$$

From the Laplace equation for pressure difference between the inside and outside of the bubble, one can substitute and rearrange equation Eq. 2-2 to yield:

$$\mathrm{Sin}\;\theta = \frac{d}{4}\left(\frac{1}{R_1} + \frac{1}{R_2}\right) + \frac{w}{d\pi\gamma} - \frac{(r + l)\rho_L d}{4\gamma} \qquad \text{Eq. 2-3}$$

Where: h = distance from surface of liquid to midpoint of bubble
R = radius of bubble
l = length of rod
d = diameter of rod

w = weight of rod
P = internal gas pressure of bubble (force area^{-1} ,e.g. dynes cm^{-1})
θ = static contact angle
γ = surface tension (dynes cm^{-1})
ρ_L = density of liquid (grams cm^{-3})
ρ_p = density of particle (grams cm^{-3})

Morris determined that internal gas pressure of the bubble and thus bubble size was a critical factor in detachment. Smaller bubble sizes provide a higher internal gas pressure and thus a greater detachment force. Morris verified that the internal gas pressure was greater than the static pressure outside the bubble. He did not, however, actually measure any detachment forces but only the change in contact angle with changes in particle weight.

Nearly all other researchers in the area of particle detachment have made use of spherical particles in the tests and models. Huh and Mason (1974) worked with mathematical models of asymmetric spheroids solving complicated equations to determine the effect of rotation, interface deflection, and contact angle on the spheres. The models ignored internal gas pressure of the bubble but did show that the largest particle that can remain attached to an interface is very dependent on contact angle, as suggested by Eq. 2-2.

The use of spheres in modeling requires many assumptions and as shown in Figure 2-1, provides for a complicated solution of any equation. The solid-liquid-gas interface will move up the edge of the particle until equilibrium is reached. As the interface moves, the surface tension force vector, γ, also changes direction which in turn changes the magnitude of the upward force component. In a mathematical model, this change in upward force changes the location of the interface on the particle which again changes the surface force vector direction and also the perimeter distance of the interface. These complicated interrelationships have required the use of extensive equations and heavy computing power for solutions that also involve several assumptions.

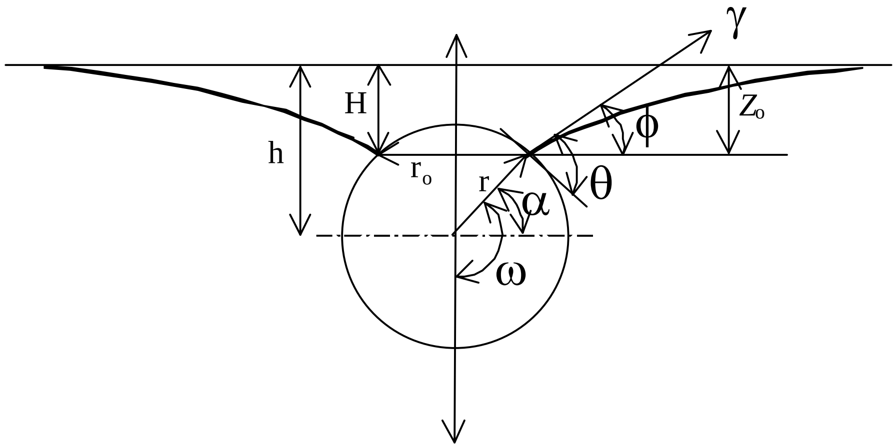

Figure 2-1. Spherical particle at interface.

After completion of this author's work using centrifuges for particle detachment measurements and after extensive literature research, one article was discovered describing some previous work involving centrifuge tests. Nutt (1960) used spherical particles (glass beads) coated with silicone oil placed at the water surface in a tube and rotated in a centrifuge at increasing speeds until the particles detached. His model also involved complicated interrelated geometry for which he claimed an exact analytical solution is not feasible. For detachment to take place at the semi-static interface, the centrifugal force must be just greater than the sum of the surface tension force and the buoyant force, $F_c < F_s + F_b$. Just before detachment of the particle of radius r, and at the critical centrifugal acceleration, a, the sum of all forces is zero.

$$\frac{4}{3}\pi r^3 \rho_P a = 2\pi r\gamma \cos\alpha \cos(\pi-\theta-\alpha) - \rho_L a \left[(2\gamma/\rho_L a)^{1/2} \pi r^2 \cos^2\alpha \{1-\sin(\pi-\theta-\alpha)\}^{1/2} + \frac{4}{3}\pi r^3 - \frac{\pi}{3} r^3 (1-\sin\alpha)^2 (2+\sin\alpha) \right] \quad \text{Eq. 2-4}$$

It was left to Schulze (1977) to provide further analytical solutions.

When considering the particle mass as the only detachment force, Scheludko et al, (1976), showed that the maximum limit of flotation is given by:

$$r_{\max,g} = \sqrt{\frac{3\gamma}{2\rho g}} \sin\theta/2 \qquad \text{Eq. 2-5}$$

where it is estimated that $\alpha = \theta/2$, that the contact angle, θ , is less than 40^{o}, and that the particle is static. In a separate equation they also derived the maximum floatable particle size when the removal is determined not by the weight of the particle but by the kinetic energy of its impact with the bubble. That equation is not considered here since this work concerns particle detachment after complete attachment to the bubble.

By solving Eq. 2-6, Schulze (1977) was also able, for the first time, to calculate the energy of detachment, $E_{ab} = \int_{h_{eq}}^{h_{Crit}} (\sum F) dh$.

$$\sum F = \frac{2}{3}\pi r^3 (\rho_P - \rho_L)\left[\frac{2\rho_P}{\rho_L} - 1 + \cos^3\omega - \frac{3h}{2R_P}\sin^2\omega + \frac{3}{a^2 R_P{}^2}\sin\omega\sin(\omega+\theta)\right] \qquad \text{Eq. 2-6}$$

This equation is for spherical particles and ignores the internal gas pressure of the bubble since it assumes that $R_P << R_B$, which is not the case for coarse particles.

To access the significance of high turbulence zones in impeller flotation cells, Jowett (1980) simplified some of Morris' analysis. Ignoring the capillary pressure and the residual hydrostatic term, the largest floatable particle diameter, *d*, is given by:

$$d_{\max,g} = \sqrt{\frac{4\gamma\sin\theta}{(\rho_P - \rho_L)g}} \qquad \text{Eq. 2-7}$$

where $g = u^2/R$, u = relative bubble velocity and R = radius of gyration of an eddy. He had only limited success in verifying this model with actual plant data, since some plants were floating copper at much larger sizes than others with little explanation of the difference. He did make the point that actual plant data confirmed the very rapid drop in recovery which is typically observed in flotation recovery above some critical particle size. This very rapid drop is predicted by models that allow for variations in detachment due to current eddies caused by variations in impeller turbulence.

Schulze (1984) worked extensively on defining the shape of the meniscus of the liquid at the three phase interface using spherical particles. He also explored the mathematics of many minute points of bubbles and aggregate particle motion. Based on

Figure 2-1, he developed an expression to approximate the maximum floatable particle size by making some approximations to solve for the meniscus height Z_o, and ignoring the hydrostatic and gas pressures:

$$D_{\max,G} = 2\sqrt{-\frac{3}{2}\frac{\gamma \sin\omega \sin(\omega+\theta)}{\Delta\rho g}} \qquad \text{Eq. 2-8}$$

Crawford and Ralston (1988) provided a partial review of work by previous investigators and performed numerous contact angle measurements. Through this work it was concluded again that the contact angle and particle size are the major determinants for particle detachment.

2.3 A Simplified Detachment Model for Cuboidal Particles

As mentioned previously, nearly all detachment models have involved the use of spherical particles and the associated complications. In actual plant practice nearly all particles > 0.100 mm are not at all spherical. This is especially true of coal which has particles that are either cubical or rectangular overall (cuboid) and have edges varying from sharp to rounded.

The major force to be overcome if a particle is to be detached from an interface or a bubble is the capillary or *surface tension force* (and the more fundamental quantity, *surface free energy*). When a liquid and gas interface is in contact with a solid, the surface of the liquid bends at some angle θ relative to the solid. This angle θ, is a function of the hydrophobicity of the solid. The more hydrophobic is the solid, the greater it resists being wetted by the liquid and thus greater is the contact angle. Increasing the contact angle of the solid to improve flotation is the principle on which most minerals flotation is based. For purposes of this work it will be assumed that the liquid is water, since other liquids will have different contact angles with the solid. Water is the flotation liquid in nearly all coal and minerals processing plants.

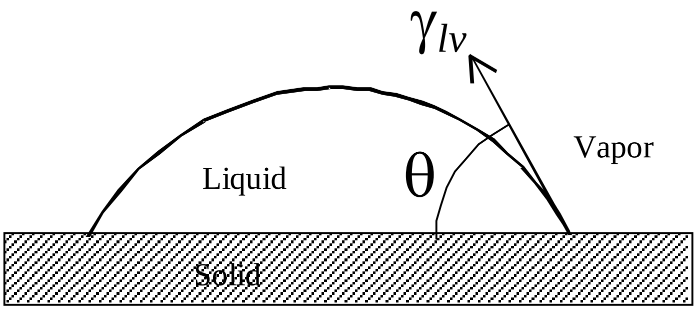

Figure 2-2. Liquid drop on hydrophobic solid.

The surface tension is a contraction or tension force at the liquid-vapor surface. The surface tension force vector γ_{lv} (Figure 2-2), acts at the three-phase contact point and in a direction away from the solid and at angle θ, to the solid.

Consider cuboid particles located at the surface of the liquid much as would be found with a very large bubble. One would expect the liquid to form a meniscus against the side of the particle as in Figure 2-3, similar to that of liquid in a graduated cylinder. However, this cannot be the case since the direction of the surface tension force would actually be downward rather that upward.

Figure 2-3. Common misconception of three-phase contact with cubical particle.

To have an upward component of the surface tension force, the meniscus must bend upward from the particle as in Figure 2-4. If the liquid is against the vertical face of the solid, an upward component of the force would require a contact angle (θ) greater than 90^o. Contact angles above ~70^o are unknown in coal and mineral flotation.

Figure 2-4. A three-phase contact with upward surface tension force.

The only way remaining for the surface tension to support the particle for $\theta < 90^{o}$ is with the three-phase contact point on the top of the cuboid as in Figure 2-5. Close inspection of particles suspended at the surface reveal that the three-phase contact is indeed made at the top edges of particles.

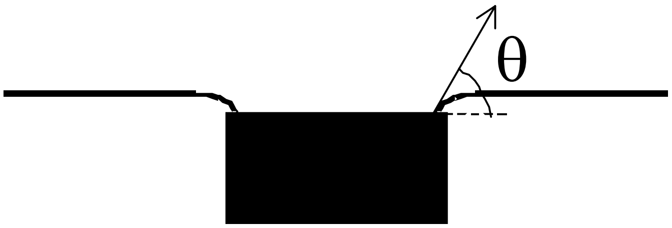

Figure 2-5. Three-phase contact point on top of cuboid particle.

With the surface tension force properly positioned one can consider all relevant forces to derive an expression for detachment force. At static condition with a particle suspended at the surface of a liquid, the upward forces involved are buoyancy and the vertical component of the surface tension force. The downward force is gravity. For the particle to remain attached to the surface, the sum of the buoyancy and the vertical portion of surface tension forces must be greater than the gravitational force.

$$\left[Buoyancy + Surface\,tension\,(vert)\right] \quad > \quad gravity\ force \qquad \text{Eq. 2-9}$$

If the above conditions are met, additional forces must be applied for a particle to become detached. In an actual flotation cell these other forces include, but are not limited to, the turbulence from the impeller of the cell agitator, eddy currents from the bubble moving through the liquid, and the energy release when bubbles break through the pulp/froth interface and coalesce. These detachment forces are difficult to measure in plant practice and it is therefore very difficult to quantify the exact amount of force required to remove a particle from a bubble. One method of measuring the detachment force on an individual particle basis is to place the particle at the surface of a liquid in a glass tube which is then rotated in a centrifuge. Increasing the speed of rotation will eventually dislodge the particle. The increased speed causes an increase in the gravitational force until that force is greater than the sum of the buoyancy and surface tension. To develop an expression for the point at which detachment occurs requires setting the sum of forces equal to zero.

$$\Sigma F = \textit{Buoyancy} + \textit{Capillary surface tension force} - \textit{Gravitational force} = 0$$

or

$$\Sigma F = V_l \rho_l g + \textit{perimeter}\ \gamma \sin\theta - V_p \rho_p g = 0 \qquad \text{Eq. 2-10}$$

where perimeter is simply *2(length + width)* and

V_l = volume of liquid displaced by particle
ρ_l = density of liquid (grams cm^{-3})
g = gravitational force
γ = surface tension of liquid (dynes cm^{-1})
V_p = volume of particle
ρ_p = particle density (grams cm^{-3})

Since the particle is nearly 100% immersed in the liquid then $V_l = V_p$ and

$$V_p g(\rho_l - \rho_p) + 2(l + w)\gamma \sin\theta = 0 \qquad \text{Eq. 2-11}$$

The force required to detach a particle from the interface can be expressed in terms of g and the general model becomes:

$$g = \frac{-2(l + w)\gamma \sin\theta}{V_p(\rho_l - \rho_p)} \qquad \text{Eq. 2-12}$$

A model often has more relevance if g is expressed in multiples of earth's gravitational force G, where G = g / 981 cm/sec/sec.

2.4 Experimental

In order to validate the model in Eq. 2-12, a set of experiments was undertaken involving a laboratory centrifuge capable of holding what are essentially large test tubes.

Sample Preparation and Procedure

A high grade coal from the Pocahontas #3 seam in West Virginia was selected because of it high contact angle and its very slow oxidation rate, which would ensure consistent data over a several week period. The coal was collected from the clean coal stream of an operating preparation plant and was later screened into four sizes covering the range of what is considered coarse coal for flotation.

16 x 20 mesh (1.18 x 0.850 mm)
20 x 30 mesh (0.850 x 0.600 mm)
30 x 40 mesh (0.600 x 0.425 mm)
40 x 50 mesh (0.425 x 0.300 mm)

Initially ten particles were placed in the tube for a given size. Particles were eased onto the surface of the distilled water. If a particle broke through the surface and fell to the bottom, it was re-floated. The tubes were placed into the centrifuge and rotated at slowly increasing speeds in increments of approximately 500 rpm. After each increase in speed the centrifuge was stopped using an electric brake which eased the centrifuge to a stop with minimal disturbance to the particles. The number of particles remaining at the surface were counted and the centrifuge was then restarted to the next higher speed.

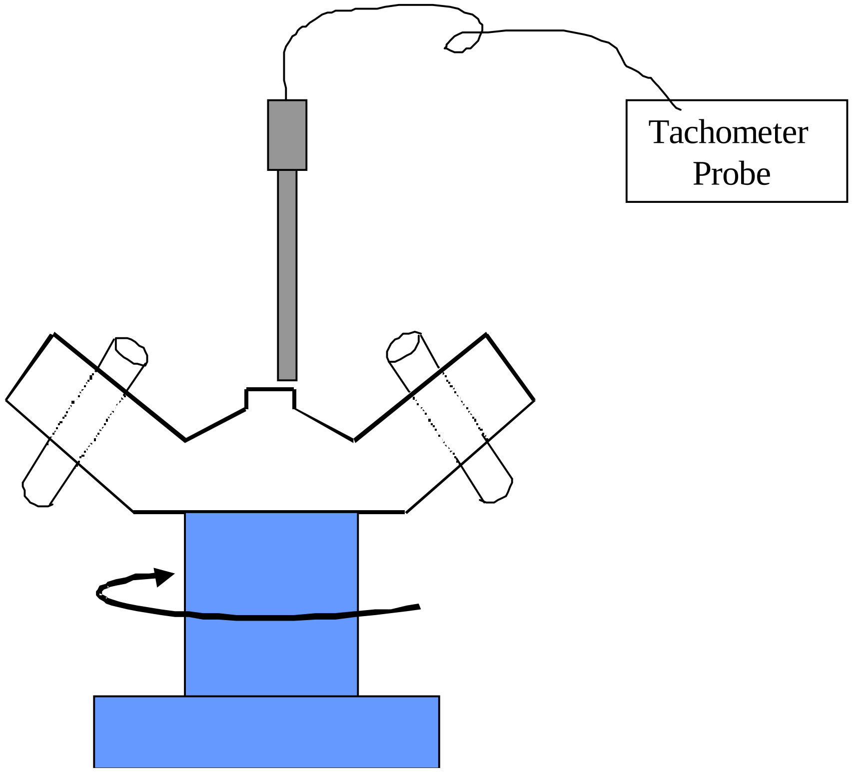

Figure 2-6. Centrifuge test device for particle detachment measurements.

The values for percent remaining at the surface were then plotted versus the G-force at that speed. G-force was calculated from the speed according to the following formula:

$$G = \frac{(rpm)^2 r}{91.19g} \qquad \text{Eq. 2-13}$$

where the radius of gyration r = 7.6 cm for the combination of centrifuge and water level in the tubes (10 ml of distilled water was used for all tests). The value of g was 981 cm/sec^2. Even though the particles were screened over a narrow size range there remained various size and shapes of particles within a given size range. For this reason the point at which 50% of the particles remained at the surface was chosen as the G-force to represent the force required for detachment of that size and condition.

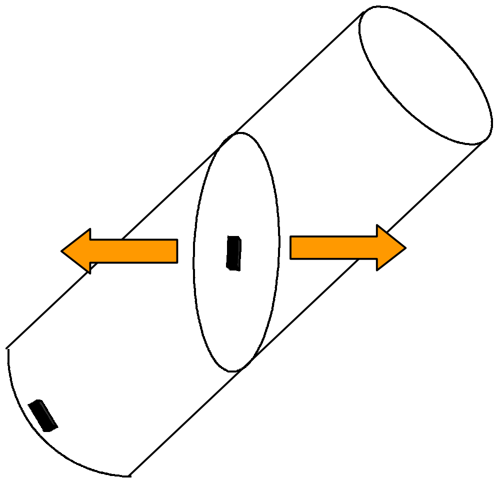

Figure 2-7.   Particles in tube during rotation.

Multiple Particle Results

Although the ten particles used in each tube were from the same size range, they were still of slightly varying sizes and one or two particles would usually drop at a very low speed, while one or two would not detach until a very high speed. The point at which 50% of the particles remained on the surface was thus chosen to represent the average G-force required to detach the particles. As expected, the coarser particle sizes require less force to detach than do the finer particles (Figure 2-8). The addition of frother to lower the surface tension had a small effect on the force required for detachment as seen in Figure 2-9. The drop in the detachment force required of approximately 65% from no frother to a high frother dose corresponds to the drop in surface tension of approximately 69% for a drop of from 72 to 50 dynes/cm. This linear relationship to surface tension γ is predicted from the general detachment force model in Eq. 2-12.

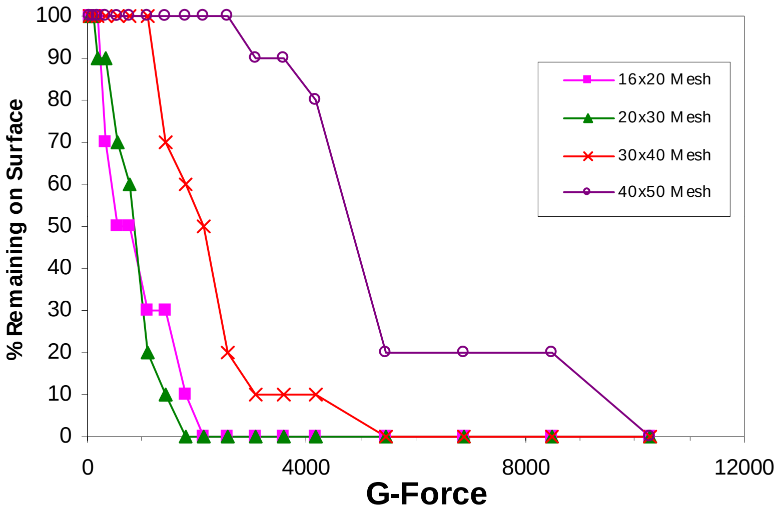

Figure 2-8. **Size by size detachment force, no frother, 10 particles each.**

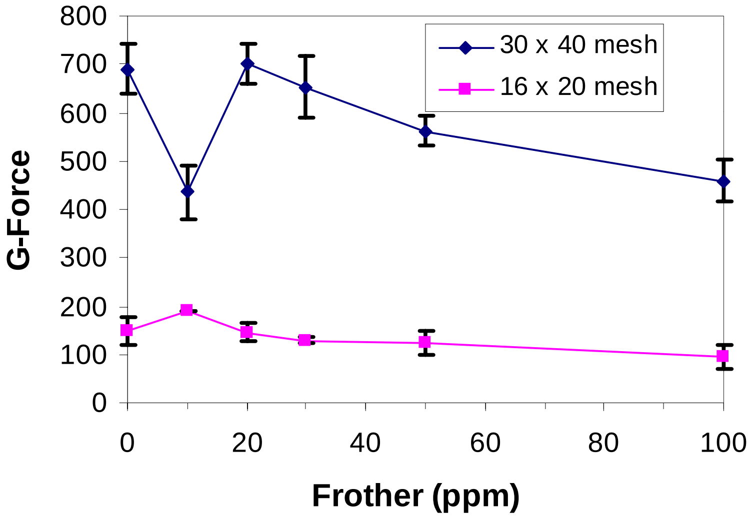

Figure 2-9. **50% detachment point for various frother dosages.**

As discussed above, the results from tests using ten particles were instructive but they also illustrated the problem with having multiple particles together on the surface. Figure 2-8 shows the overlap in the results for the two coarsest sizes while Figure 2-9 displays some variability in results for repeated tests. Nutt (1960) found as much as 50% difference in the detachment force for one particle compared to that of ten mono-sized spherical particles (Figure 2-10). Nutt also found that generally the aggregate of ten particles was detached from the surface as a whole and thus even though the data presented in Figure 2-8 and Figure 2-9 appear as expected on a trending basis, the absolute values may be in question. This concern for accurate reproducibility led to using only one particle per tube in further testing.

Single Particle Results

Selecting typical size single particles from the previously mentioned size classes allowed for more accurate measurements and modeling of the detachment force. Single particles were viewed under a microscope attached to an image analysis system which provided more precise measurements of particle dimensions. As expressed in the simplified model of Eq. 2-12, more than just the particle volume must be known. Actual particle shape and dimensions affect the perimeter length of the particle at the three-phase contact.

For coal and most minerals, one dimension of each particle is much less than the other two dimensions and thus the particles tend to orient themselves in a horizontal position as they are placed on the surface. This is the most stable position from a buoyancy perspective and once in this position, the maximum perimeter length is presented to the surface tension force. The maximum surface tension force is then available to resist any reorientation of the particle.

Several particles of coal were sized, placed at the liquid surface of the centrifuge tubes and then subjected to the centrifugal force at increasing speeds as before. The actual force required to detach the particles in units of G-force was calculated from the speed at which detachment occurred. By using the actual dimensions of each particle, the predicted detachment force was then calculated from the model and plotted for each particle alongside the measured force. Particle mass was chosen as the single number to represent each particle size which was then plotted versus the G-force in Figure 2-11. The plot shows excellent agreement between the measured and predicted detachment forces. A trendline is shown in lieu of a plot of a specific equation since the detachment force is calculated from three characteristics unique to each particle (length, width, and thickness) and mass was the single attribute chosen to characterize the particle in the 2-D plot.

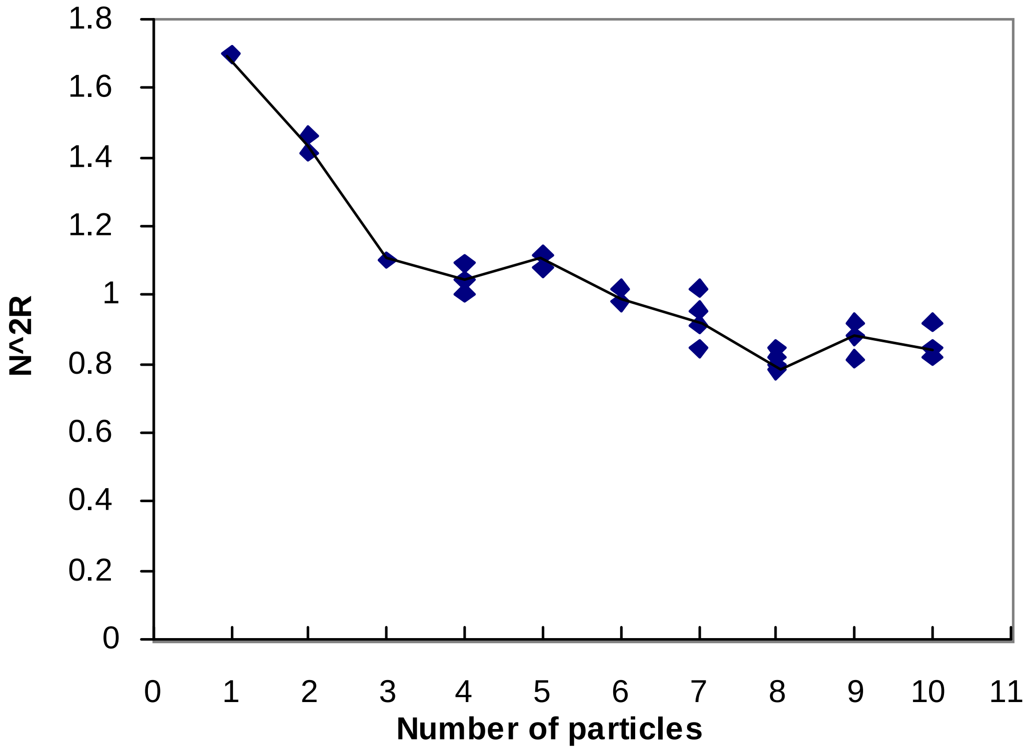

Figure 2-10. Adhesion strength of aggregates of spheres, N^2R (rpm^2 x radius of motion), as a function of the size of the aggregate, from Nutt (1960).

Since the surface tension force functions at the three-phase interface, the perimeter length relative to total particle mass has a very strong effect on particle detachment. This is seen in Figure 2-12, where the flatter particles require a much higher detachment force than the equivalent mass of particle in either a sphere or cube. The detachment force of the sphere was calculated using Eq. 2-5 from Scheludko et al, (1976).

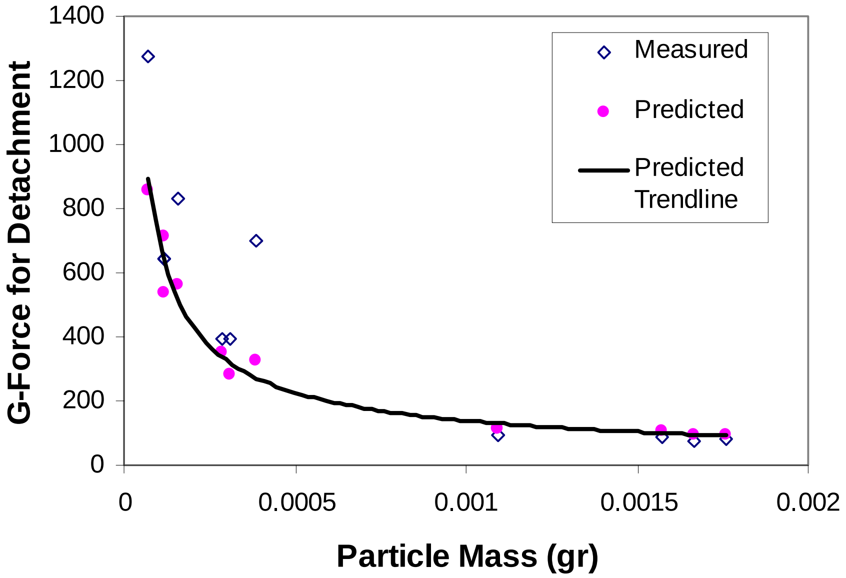

Figure 2-11. Measured and predicted detachment force for each particle weight, ($\theta = 65^o$, $\gamma = 72$, $\rho = 1.35$).

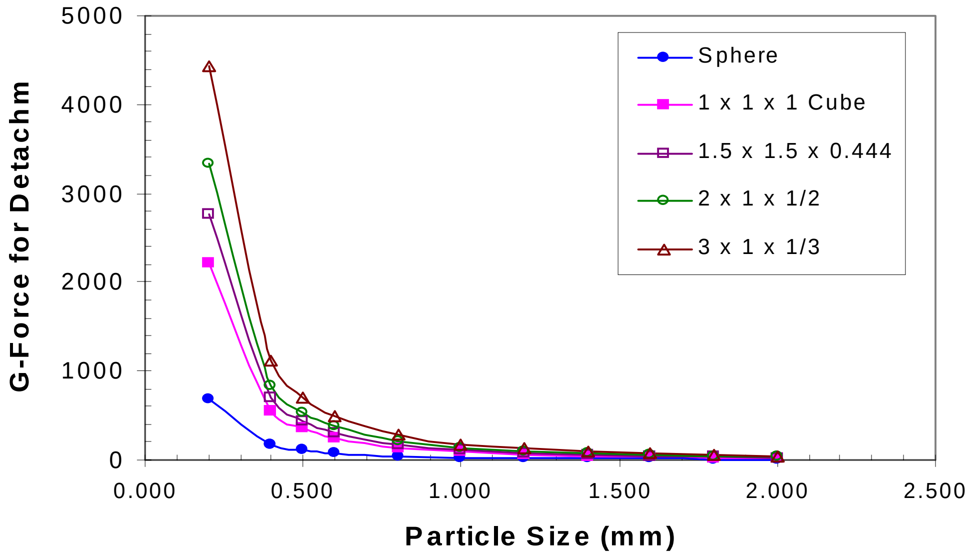

Figure 2-12. Predicted detachment force vs. particle size for various shapes and of equal mass.

Changing the contact angle or the surface tension has a significant effect on the detachment force required for particle removal, but not nearly as much as the effect of flat and elongated particles (Figure 2-13). Since coal in the size range of 0.1 to 1 mm is usually rectangular and somewhat flat shaped, this is good news for coarse coal flotation. Given that detachment is the main reason for poor coarse coal flotation, the model indicates that to improve the flotation of coarse coal requires either an increase in surface tension and/or an increase in contact angle. The physics of bubble size generation require the addition of frother to produce the low surface tension which maintains the small bubbles necessary for high flotation capacity. Thus any increase in surface tension would be counter-productive. Contact angle can be improved and is usually done so for coal by the addition of a hydrocarbon such as kerosene or diesel. Previous work at Virginia Tech has shown that kerosene can increase the contact angle on partially oxidized Pittsburgh #8 seam coal, from 28 to 42 degrees by the addition of 2500 g/tonne. The plant test data will show some results of increased diesel.

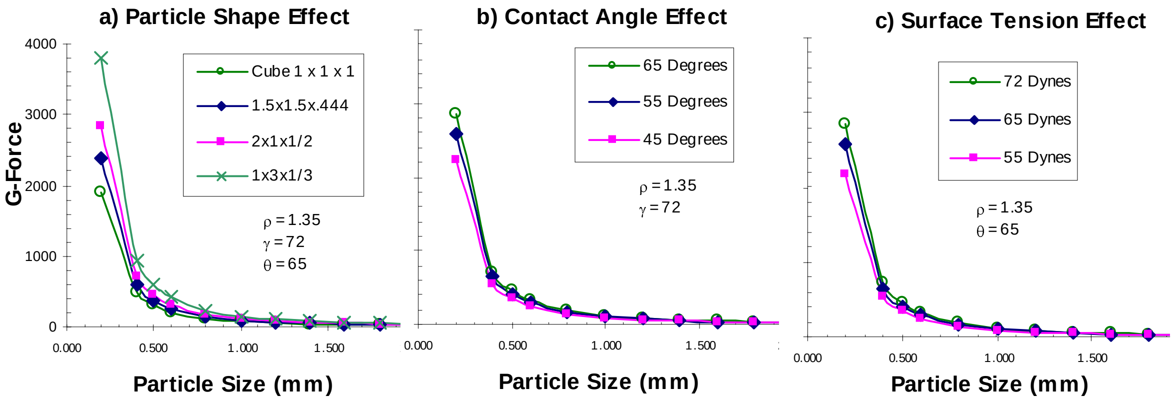

Figure 2-13. Effect on detachment force of changes in a) particle shape, b) contact angle and c) surface tension. A 2 x 1 x ½ shape used for b) and c).

The shape and location of the sharp bend of the curves is very important in Figure 2-12 and is confirmed in the plots of Figure 2-13. When moving from coarse to fine, below 1 mm the detachment force required begins to increase significantly and below 0.5 mm it begins a rapid increase. This follows the results of plant tests discussed later in this document and also matches some of what is known from actual plant experience. The 0.5 x 1 mm coal has been known to float, but only under excellent conditions and then inconsistently. The 0.150 x 0.5 mm material has been known to float in many locations but is still believed by many to be too difficult to undertake commercially. The plant testing chapter will discuss further many of the reasons for the apparent contradictions, but from the rapid change in the slope of these plots it is obvious that particle sizes close to 0.5 mm will be difficult to float under normal conditions.

2.5 Summary and Conclusions

In the past, most theoretical efforts have been spent on quantifying and solving equations concerning spherical particles at the air-water interface. The approach taken here is to consider the particle already attached to the bubble and then to develop a better understanding of the force actually required to remove it from the bubble as well as the factors involved. Rectangular or cuboid type particles were utilized in both testing and modeling, since that size represents the vast majority of minerals and all coal that is subjected to flotation. With the simplified model presented (repeated below for convenience,) one can easily see the direct effect that particle shape, mass, contact angle, and liquid surface tension have on particle detachment.

$$g = \frac{-2(l+w)\gamma \sin\theta}{V_p(\rho_l - \rho_p)} \qquad \text{General Model}$$

The particle perimeter portion of the model points out a surprising fact of detachment that was never mentioned by researchers considering spheres: a much stronger force is required to detach a flat particle than is required to detach either a sphere or cube of comparable mass. This means that coarse coal particles, at or above 0.5 mm, should float well since they are flat and somewhat rectangular.

Since coal particle shape is difficult to control without increasing the amount of ultrafines, the model indicates that the only methods remaining to increase the particle detachment force require either: (i) increasing surface tension γ, which is not practical since flotation recovery generally benefits from small bubbles and a reduced surface tension is needed to maintain the small bubbles or (ii) increasing contact angle θ, which is achievable in a small amount.

The model predicts a sharp drop in the force required for detachment at the coarser particle sizes. The corner of the “L” shaped curve resulting from plotting detachment force vs. particle size tends to be around 0.5 mm for most typical coal conditions. This is the size area at which detachment is much more likely. This area also agrees with most plant practice when serious efforts are made to float the coarser particles.

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CHAPTER 3 Plant Test and Scale-Up

3.1 Introduction

The greatest inefficiencies in coal preparation are in the minus 1 mm particle size range. As discussed in Chapter 1, there are considerable problems with the actual cleaning of the material in this size range as well as problems due to misplaced sizes being sent to a circuit that cannot treat that size. It is common for plants to either discard or treat the minus 0.150 mm material in conventional flotation while the plus 0.150 mm material goes to spirals, water-only cyclones or heavy media cyclones. It is the intent of this work to evaluate the potential for increasing the size range that can be treated in flotation. With today's high ash flotation feeds, conventional flotation cells cannot be used to float coarse coal due to the high amount of entrained ash that comes with the froth product when controlling the cell for high recovery. Today, the best candidate to be evaluated for use in the flotation of coarse coal is column flotation with wash water.

When properly operated, columns tend to have better recovery due to the high volume of consistent sized bubbles which provide better capture of coarse and fine particles and thus increase the flotation rate. Probably the greatest advantage to flotation columns for coarse coal is in their ability to handle high amounts of high ash slimes (clays, etc) through the use of froth washing. This use of wash water not only reduces and stabilizes the product ash content but it allows the coarse coal to be fed to the column without any prior desliming or fine size cuts. This means that the intended top size feeding the column will be the smallest size cut made in the preparation plant. The coarser the size cut, the less material that is misplaced into the cleaning device for the next coarsest size. There is a tremendous difference in the misplacement of fines to coarse when the size cut is changed from 0.150 mm (100 mesh) to 0.5 mm.

A small laboratory size column (2-inch), is the simplest method to evaluate column flotation for a given coal. Since much of the loss of coarse coal is believed to be due to dropback in the froth zone (Finch and Dobby, 1990), it is important that any scale-up testing be performed in a column with sufficient froth travel distance to provide for potential loss of coarse coal. Wall effects in small laboratory columns may actually stabilize the froth zone and reduce dropback compared to that of full-scale columns. When the quantity of coarse coal feed that would be required for proper testing was considered, it became apparent that a lab or pilot plant would be insufficient for testing. Although there are many plants with operating flotation cells, nearly all have a feed that is too fine in size consist for the testing considered. At the time of this project there were only two plants operating flotation columns on coal and they were also utilizing a very fine coal feed. Fortunately, the Center for Coal and Mineral Processing at Virginia Tech was involved in a project with Amax Research and Development that was being funded by the U. S. Department of Energy (DOE) to investigate the production of premium coal fuels (DOE Contract No. DE-AC22-92-PC92208). As will be detailed later, a plant site became available in which a sufficiently large test column, 750 mm (~30-inch) diameter, could be tested on a fine and coarse feed up to 1 mm in size.

This testing provided an opportunity to study the real potential for floating coarser sizes than commonly tried. Size-by-size analysis is provided along with the effects of several parameter changes. Test work also provided scale-up information to be used in designing the world's largest full-scale columns then in use on coarse or fine coal.

3.2 Project Background

A small part of the large Premium Fuels project was Task 3 Development of Near-Term Applications, which was intended to specifically address the use of advanced flotation and selective agglomeration processes for recovering coal lost in existing coal preparation plants. The overall goal of Task 3 was to produce a clean coal product which could be sold in existing markets by one or both of the following strategies:

- Increase the percentage recovery of marketable coal from the ROM coal.
- Improve the quality and value of the marketable coal (heating value, sulfur or ash content, and handling characteristics) in a cost-effective manner.

Under the above task was Subtask 3.2 Engineering Development, which had a primary objective of pilot-scale testing and engineering development of the selected applications. Previous laboratory testing had shown that from an economic and technical point of view, column flotation was a much more viable process and was thus chosen as the most likely candidate for meeting the near-term application criteria.

Plant Selection

The Lady Dunn Preparation Plant, near Montgomery, West Virginia, was volunteered by the Cyprus Amax Company and provided an excellent test site for proving the advanced flotation column. The flotation feed is typically around 40% ash and has a high percentage of minus 325 mesh material in the flotation feed. Also, the plant had existing mechanical conventional flotation cells and therefore results could be directly compared to existing conventional technology. At the time of the pilot testing, the plant was mid-way through an expansion program. The plant flowsheet consisted of heavy-media vessel for coarse coal (+6 mm or + ¼ inch), heavy-media cyclones and Deister tables shared the load for the 6mm x 0.150 mm (¼ inch x 100 mesh), and conventional flotation on the minus 0.150 mm overflow from desliming cyclones (Figure 3-1).

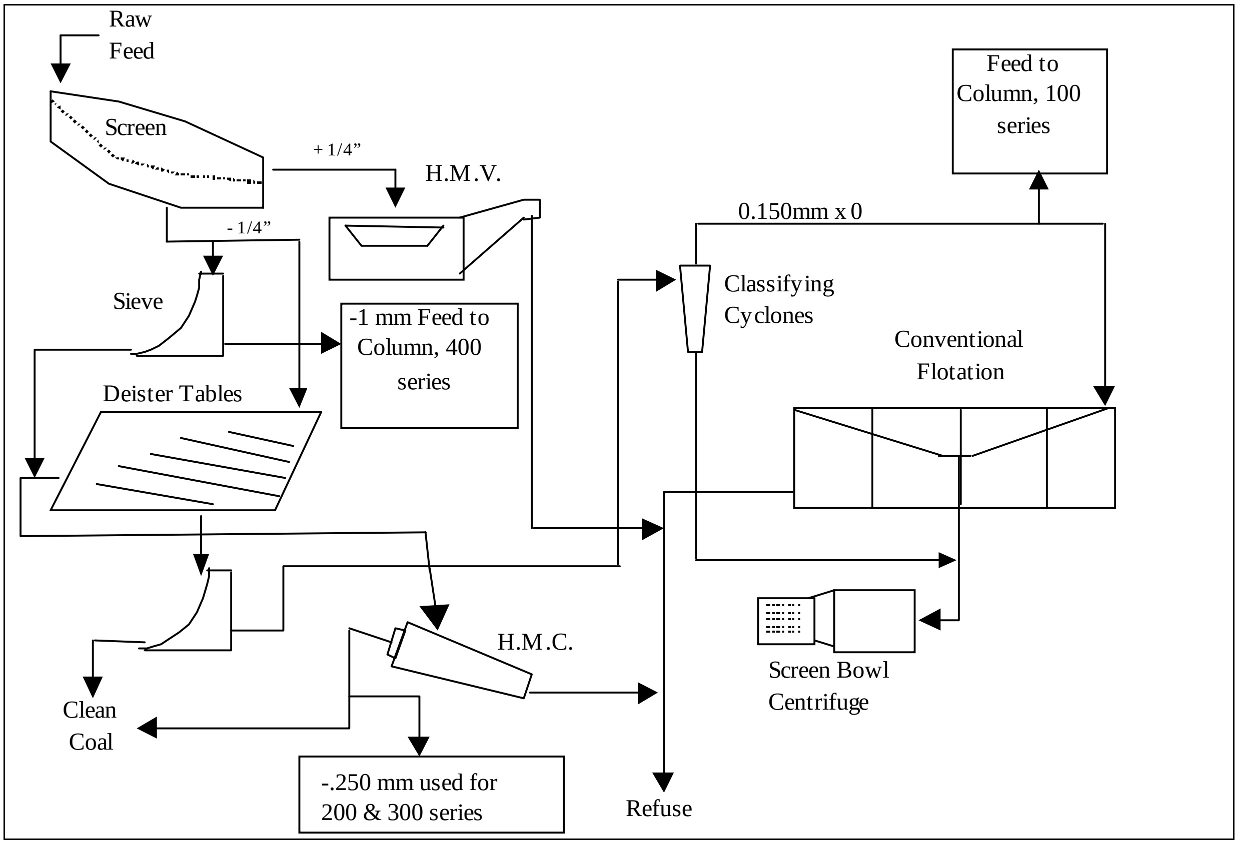


Figure 3-1. Lady Dunn Plant flowsheet during 30-inch column testing showing the two major feed sources.

Phase I of the expansion program involved the addition of the heavy media cyclone circuit while the design of Phase II was not completed but would include a major change to the flotation circuit. The author considered this an excellent opportunity to prove the benefits of increasing the size of feed to flotation by utilizing column flotation and proposed that the pilot-scale test plan be modified to include not only testing of existing flotation feed (-0.150 mm) but to also test up to 1 mm coal. The plan was accepted and the coal company was receptive to the possibility of including changes in the Phase II design.

3.3 Experimental

Objectives Of Plant Testing

Testing of a 30-inch diameter Microcel^{TM} flotation column at the Lady Dunn preparation plant began in June, 1995. The objectives were:

- Determine the benefit of applying the advanced flotation processes to this operating plant.
- Comparison of the flotation column to the existing conventional cells on current flotation feed (minus 100 mesh).
- Determine the optimum size feed for a column at this plant, i.e., emphasis on coarse coal recovery.

Testing of the 30-inch column at this plant could also provide a general research element by furthering the understanding of the relationship of bubble capacity and collector dosage to recovery of the coarser size fractions of coal.

Test Plan and Installation

To meet the stated objectives required in-plant testing with a column of sufficient size to provide reasonable scale-up information and utilize equipment similar to that used in industry. Recovery of various size fractions and product quality (ash) were the major results to be measured. The critical scale-up parameters that would be required for optimum recovery at a reasonable product ash were investigated. Parameters such as feed rate, aeration rate, frother and collector dosage were varied to determine their effects on the recovery of various coal sizes.

Performing process testing in an operating plant required that considerable attention be paid to feed source and control. With the normal variations within a plant process it is sometimes difficult to maintain a proper feed and this was especially true at the Lady Dunn Plant. Coarser material and a very stable froth concentrate were added difficulties that made equipment selection and proper test circuit layout very important.

30-Inch Column Chosen

Past scale-up of the Microcel^{TM} column to full-size units has proven successful even from laboratory size units. Normal in-plant testing could have possibly involved an 8- or 12-inch diameter column, which are usually preferred due to the simplicity of installation.

However, there was concern that a small diameter column would not properly simulate a large column in coarse coal recovery. The concern was that a small column has such a short froth travel distance that the coarse coal would not drop back into the froth zone in equal proportions to that of a larger column. To provide a reasonable froth travel distance and to allow for more froth zone dropback, the largest Microcel test column available was chosen. There was also concern that the feed pumps, pipes and valves associated with a smaller column may have plugging problems due to the misplaced coarser feed sizes found in an operating plant. To accommodate these concerns, a 30-inch diameter column was loaned from the Virginia Tech pilot plant, the 30-inch column having been previously constructed and tested under an earlier DOE project.

To develop preliminary feed and control parameter information for the 30-inch column, a drum of flotation feed was collected from the existing plant and tested in a 2-inch diameter laboratory column at Virginia Tech. Scale-up predictions were made from this preliminary test data and a flotation rate was developed. The scale-up indicated that the total height for the 30-inch column would need to be no more than 21 feet, which suited the height limitations within the plant.

Test Column Feed Sources

At the time of testing, the Lady Dunn Plant flowsheet was in a stage of transition. Prior to an expansion two years earlier, the plant consisted of heavy media vessels for coarse coal (+1/4 inch), Deister tables on 1/4 inch x 100 mesh, and conventional flotation on the minus 100 mesh material. Around 1993, a heavy media cyclone circuit was added to clean the Deister table feed. But, with increasing demands on the plant for increased production, the tables remained in service as well as the HMC circuit, which allowed for a higher plant throughput. The clean fine coal from the HMC traveled across a large sieve, the underflow of which was considered minus 60 mesh and was sent to flotation (Figure 3-1). Thus, two existing flotation feed streams were available. One was a raw minus 0.150 mm (-100 mesh) cyclone overflow and the other was a minus 0.250 mm (-60 mesh) screen underflow containing a 0.250 x 0.150 mm fraction that was mostly clean coal from the HMC circuit.

Four separate feed streams were examined before a final slurry was selected as the feed for parametric testing. The four streams were 1) a fine coal stream of minus -0.15 mm (100 mesh) cyclone overflow, 2) a coarser combined stream of cyclone overflow and partially cleaned sieve screen undersize, 3) a similar stream containing an increased amount of the coarser screen undersize, and 4) a totally raw coal sieve screen undersize stream with some material too coarse for flotation (Figure 3-1).

The raw 0.150 mm x 0 coal from the classifying cyclone overflow was the first material tested in the 30-inch column. A sample thief with a valve was installed in the existing flotation cell distribution box which was fed from the classifying cyclone overflow. This valve was adjusted to provide sufficient flow to the column feed sump on the floor below. The feed sump was allowed to have a slight overflow to ensure a constant flow to the column. The results from testing of the fine material are labeled as the 100 series.

While the results of testing the 0.150 mm x 0 material were excellent (see test series 101-110, Appendix A, testing on a coarser feed was more desirable to all parties since the intent was to prove column flotation on a broader application basis. A valve and collection box were mounted on the side of the sieve underflow pipe which was part of the feed to the flotation cells. This coarse material was blended with the classifying cyclone overflow to simulate the raw coal size consist for a minus 0.25 mm (60 mesh) flotation feed. Several tests were performed on this coarser feed and are labeled as the 200 series (Appendix A). Screen analyses of the products indicated that there was insufficient coarse material in the blend and the coarse feed valve box was modified to accept nearly all of the material in the screen undersize pipe. The flow of this combined feed proved to be unstable and of insufficient volume for parametric testing. The two tests performed on the latter feed were labeled as the 300 series (Appendix A).

The plant had recently installed a test spiral concentrator, the feed for which was taken from the underflow of a temporary fixed sieve receiving raw Deister table feed (6 mm x 0). After completion of the spiral testing, the fixed sieve was changed to one with a smaller opening (1 mm) and the underflow was fed to the column feed sump. This provided a true raw coal feed containing natural minus 1 mm fines. Since the new column feed was coarser than necessary, the plus 0.5-mm fraction was screened from the samples and accounted for separately. Testing of this raw feed was labeled as the 400 series (Appendix A).

Pilot Column Circuit Description and Operation

Although equivalent to a fully functional full-scale commercial unit, the 30-inch test column was considered a pilot-scale column. The major difference was the limited capacity of the test unit due to its 30-inch diameter compared to the 3-meter or larger in diameter of most commercial units. The test column has a capacity of 0.5 to 1 tonne per hour (TPH) of clean coal for most coals.

A general layout of the column testing circuit is shown in Figure 3-2. The column was fed by an 80-gallon feed sump to provide a consistent feed volume. Diesel fuel oil was added as a collector into the stream feeding the feed sump. The plant provided a feed pump with a remotely variable speed controller which was adjusted from the control area (near the top of the column) to maintain or adjust a given volumetric feed flow. The pump and sump were located 2 floors below and over 50 feet horizontally from the column feed area. The column feed piping discharged into a small head box just prior to entering the column. By moving the flexible pipe, a full-stream sample cut could be taken and, by noting the time required to fill a fixed volume container, a positively measured flow could also be taken.

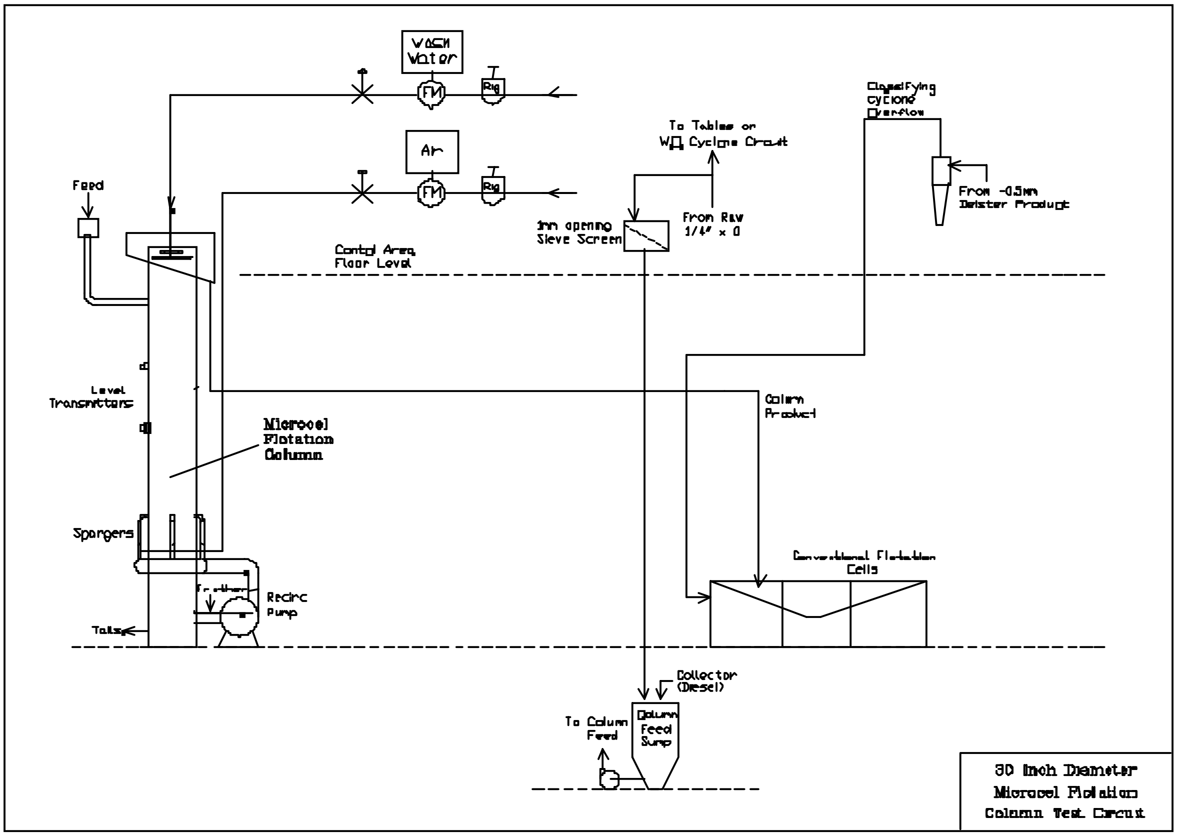


Figure 3-2. Pilot column test circuit arrangement.

The slurry recirculation pump was located on a lower floor at the bottom of the column. By recirculating the slurry through the spargers along with the addition of air and frother, small microbubbles were produced (see schematic in Figure 3-3). The frother was injected into the suction line of this pump and the air was injected just prior to the spargers. A tailings valve was also located in this area of the column and discharged through a section of hose which was maneuvered to provide a full-stream cut for the tailings or refuse sample.

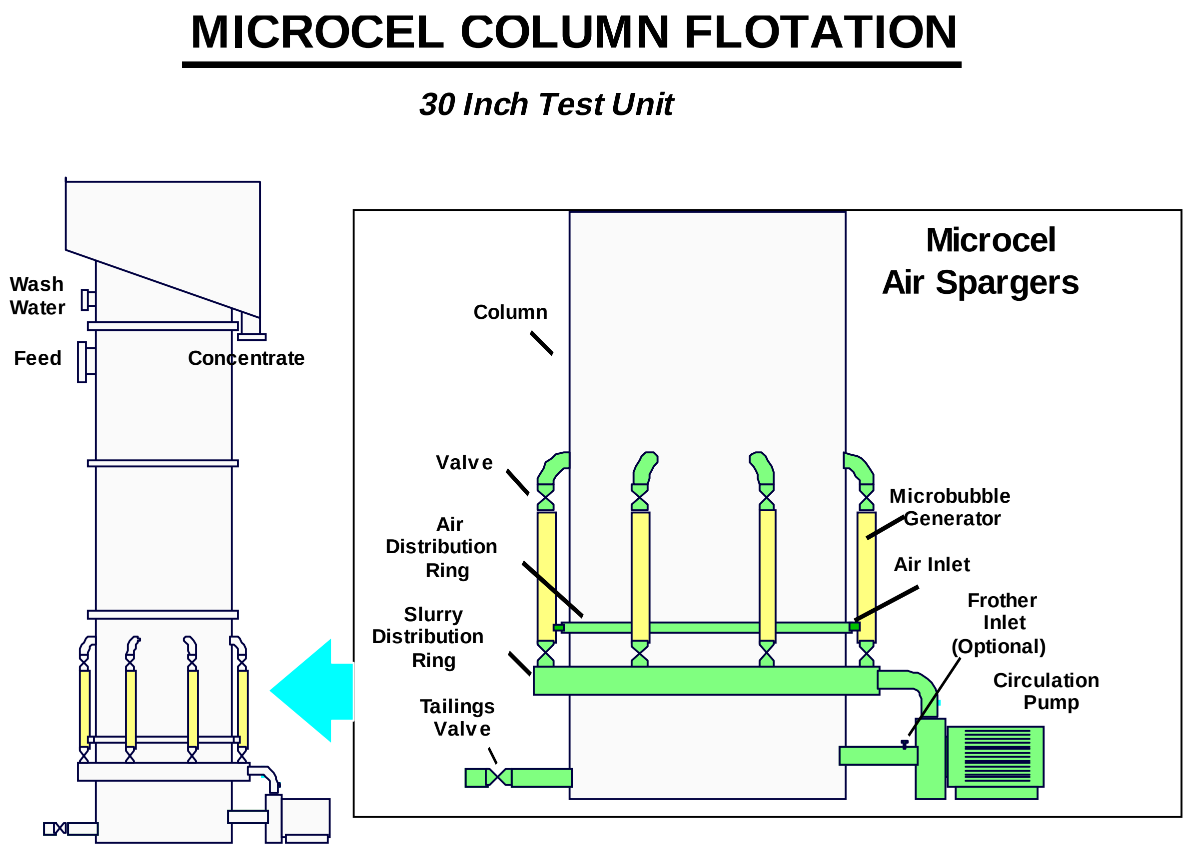


Figure 3-3. Flotation column schematic showing bubble generation system.

The froth product from the column launder discharged through a 6-inch diameter pipe to the existing flotation cell product launders. A minimal height difference between the column and existing cells provided very little slope for the froth concentrate pipe over a long distance. This lack of slope caused the column concentrate launder to back-up frequently and necessitated the use of launder water to help move the froth through the pipe. For this reason many of the concentrate percent solids values are slightly lower than actual values since the sample was taken at the pipe discharge.

Air, water, and pulp level were controlled from the control area at the top of the column. An orifice plate flowmeter with differential pressure transmitter and digital readout was used to measure the air flow to the spargers. The air flow system was equipped with a pressure regulator to provide a constant pressure to the flowmeter. This allowed for accurate flow measurements and air flow recording on a standard temperature and pressure basis. Wash water was measured with a paddlewheel type flowmeter and displayed electronically.

A pressure regulator was also provided for the wash water but low water pressure required the installation of a small in-line booster pump. Manual gate valves were used to adjust the air and wash water flows.

Pulp level in the column was maintained with a PID loop controller which received a signal from an electronic level transmitter on the column and sent a proportional signal to the tailings discharge valve. The signal from a similar transmitter placed at a lower level was also displayed at the control area allowing an air fraction (fraction of air by volume in a given section of pulp) to be calculated from the combination of the two level signals. Air fraction was calculated as:

$$Air\ Fraction = 1 - \frac{lower\ level - upper\ level}{fixed\ distance} \qquad \text{Eq. 3-1}$$

where level was in vertical inches of slurry pressure and the fixed distance between level transmitters was measured by the transmitters with slurry, but no air, in the column (Finch and Dobby, 1990).

A small sight glass near the top of the column provided a means to view the pulp/froth interface area. This was an excellent method for determining column conditions such as turbulence, approximate bubble size, or excessive air flow. Testing involved waiting 30 minutes after any change in operating parameters to allow all conditions to stabilize before sampling. Several full-stream cuts were taken for each sample and were collected in 5-gallon (19 liter) containers with sealed lids. The samples were sent to the laboratory at the end of each day’s testing.

3.4 Results and Discussion

Preliminary information was gathered from 2-inch lab column tests and pilot unit test series 100, 200, and 300. Those results are discussed separate from the two series of parametric tests (400 series) which represent the bulk of the testing data. The detailed results and test parameters are provided in order of series number in Appendix A.

Preliminary Flotation Testing

A 2-inch diameter column was utilized for the initial laboratory testing of the classifying cyclone overflow sample. Normal test procedures for the laboratory column call for maximizing air and washwater input to the column without reaching an air overload or “flooding” condition. Column tests are then run over a range of feed rates. As the feed rate is increased the limit of the froth carrying capacity is eventually reached, resulting in lower ash products but also a lower recovery. This generally produces an ash versus combustibles recovery curve as shown in Figure 3-4. From the lab results it was determined that a 30-inch column would have a capacity of no more that 100 gpm of feed slurry at the percent solids tested.

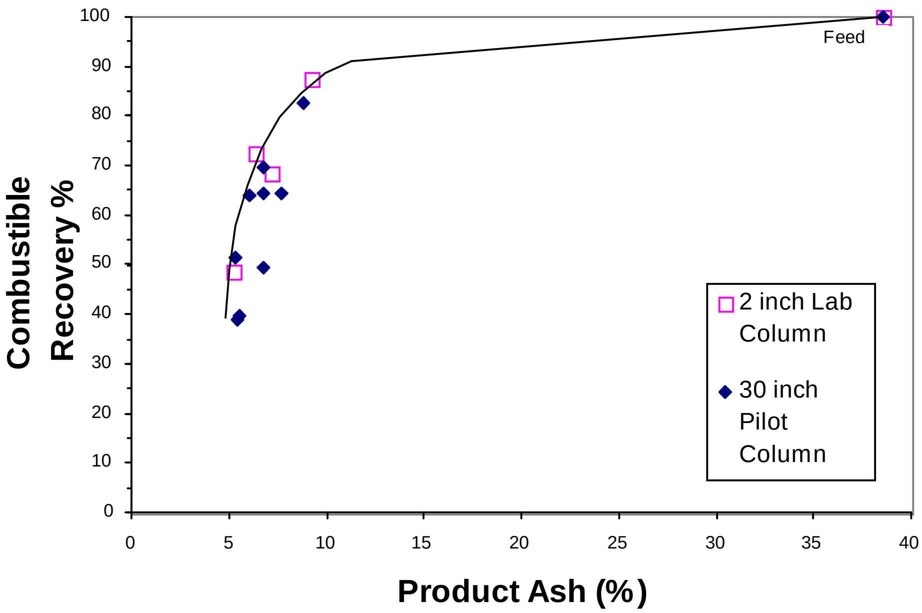


Figure 3-4. Combustible recovery vs. ash for 2-inch Laboratory and 30-inch columns (Tests 101-110).

The initial tests with the 30-inch column in the Lady Dunn Preparation Plant were on the same classifying cyclone overflow stream as tested in the 2-inch column and are labeled as the 100 series in Appendix A (test points 101-110). Results were excellent and compared well to the laboratory tests. Both sets of results are plotted in Figure 3-4 and one can see that the same ash/recovery curve was produced in the two-inch lab column as in the 30-inch diameter pilot-scale column. Figure 3-5 shows that the efficiency of the two units is also comparable since the points lie along the same curve. The purpose of the initial testing was to develop a general “ballpark” for the expected operating parameters. Before this testing was fully developed, however, it was decided to try for a coarser feed as discussed in a previous section.

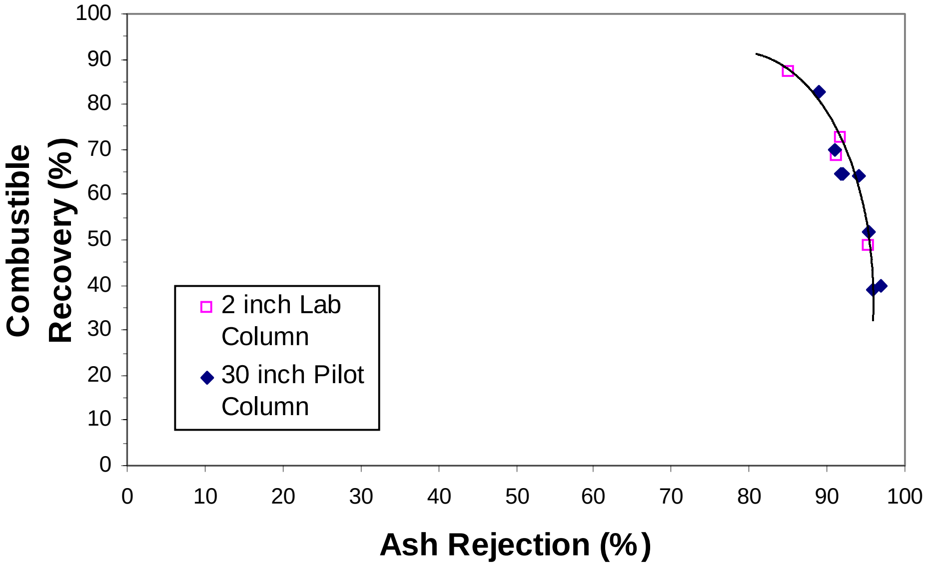


Figure 3-5. Ash rejection and combustible recovery for 2-inch Laboratory and 30-inch column (Tests 101-110).

Testing then began on the coarser feed (200 series) but again this was not fully developed since the stream did not contain as much coarse material as anticipated. The 200 Series Tables in Appendix A give the results obtained with the limited testing performed on this feed. Figure 3-6 shows that the performance was generally well below 10% product ash with a good recovery even though the column had not yet been optimized. The 200 series feed was predominantly classifying cyclone overflow and thus consisted of well liberated fines. An attempt was made to send more coarse material to the column but it was discovered that the available pipe did not carry enough material to provide sufficiently coarse feed to the pilot column. Results from the two tests performed with this attempt are labeled tests 301 and 302 in Appendix A.

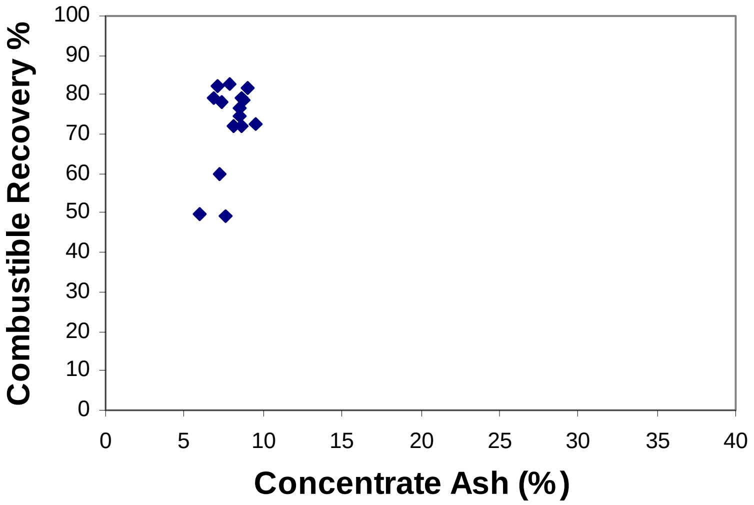


Figure 3-6. Performance of Microcel pilot column on coarsened cyclone overflow (tests 201-215).

At the conclusion of an unrelated test program in the plant, a pipe was installed to the temporary sieve underflow which allowed a true natural sized raw coal stream (1mm x 0) for column feed. Once testing began on the coarse feed there was an urgency to quickly determine the scale-up information for a potential full-scale column installation. Test numbers 401 to 416 are the initial tests to determine the general range of parameters for operation with the coarser coal and were also later considered as a parametric test series (First Series).

Parametric Testing

To best determine the effect of various operating parameters such as air rate, frother and diesel dosage, and feed rate, requires an organized testing method. Tests that produce several combinations and magnitudes of variables are necessary, but to produce all possible combinations of variables and ranges would be a tremendous task. To minimize the number of tests while producing a response surface with statistical meaning, the Box-Behnken experimental design was chosen and a computer program *Design Expert* by Stat Ease was selected for the design and to interpret the data.

Box-Behnken designs are response surface designs, made specifically to require only 3 levels of each variable, coded as -1, 0, and +1, or high, medium, and low. Box-Behnken designs are available for 3 to 7 factors (or variables) and are formed by combining two-level factorial designs with incomplete block designs. Most desirable statistical properties are created by the Box-Behnken, but with only a fraction of the experiments needed for a full three-level factorial. The three factors used in this project required only 15 experiments versus 32 for a full three-level factorial. Of the 15 tests, two are repeats of the midpoint levels to determine statistical variance. A major advantage of Box-Behnken designs is in their nature of requiring only 3 levels. Because there are only three levels, the quadratic predictive model is most appropriate.

The intent of the parametric testing was to determine the effect of various operating variables on the performance of the flotation column, specifically the recovery of the coarser fractions of coal. It was very difficult in an operating plant to produce a consistent feed for a small stream such as that required for this parametric test. For example, during the several days required to run the primary designed parametric test series (tests 451 to 465), there were considerable variations in feed solids to the column. Other problems, such as low wash water pressure, were also experienced. The problems encountered may explain some of the inconsistencies found when evaluating that data set.

The initial set of data for flotation of the raw coarse coal (tests 401 to 416) provided a much more consistent data set. The intent of this initial testing was to vary key operating variables from low to high to determine likely operating points as well as gain scale-up information. After further review of the data it was determined that the variations in control parameters for the initial raw coarse coal testing fit a Box-Behnken experimental test design. The results produce a consistent data set and prediction model that is better than that produced with the main parametric design. For these reasons the results of two separate parametric test series (first and second series parametric tests) will be presented and discussed.

Parametric Tests - First Series

The first set of results (tests 401 to 416) from the coarse raw coal feed provides the most consistent data set. Changes in several of the key operating parameters were performed, as shown in the operating parameter list for the 401 series found in Appendix A, and were meant to cover the range from low to medium to high for several of the key parameters. Even though this series of tests was not a designed parametric test, when the main parameters (i.e. frother dosage, collector dosage, and feed rate solids) were entered into the statistical analysis program as a Box-Behnken experimental design, a good correlation was found and several definite trends were realized. The Box-Behnken design provided a measure of the contribution of each parameter to the given response and also allowed the influence of joint interactions between the various test parameters to be estimated.

Although the test parameters covered a wide range of operating conditions, nearly all of the results for a given particle size fit along a single grade-recovery curve (Figure 3-7). Changes in specific characteristics of the coal particles (i.e., degree of liberation and hydrophobicity) result in a different grade-recovery curve, whereas, column conditions affect a result's location on a grade-recovery curve for a given coal. Conditions that result in a limited carrying capacity provide room on the bubble surface for only the most hydrophobic particles and result in a low ash but low recovery product. The close fit to a common grade-recovery curve indicates that for most of the tests, entrainment of non-floatable material into the froth was not a problem. The wash water flow was sufficient to remove the entrained high ash particles.

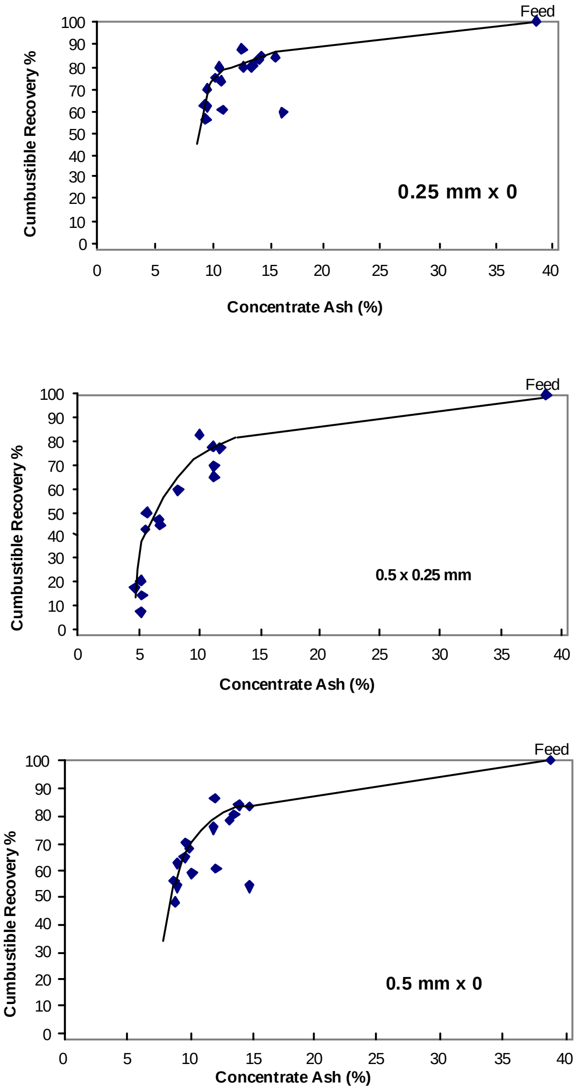


Figure 3-7. Grade-Recovery plots by size for tests 401-416.

The three particle size classes scrutinized were 0.25 mm x 0 (60 mesh x 0), 0.5 x 0.25 mm, and 0.5 mm x 0. These sizes were chosen because one of the main emphases of this test program was to determine the applicability of column flotation to a coarser coal than commonly practiced. As can be seen from the size-by-size recoveries plotted in Figure 3-8, the best recovery during each test was for the 0.150 x 0.045 mm (100 x 325 mesh) size fraction and the 0.045 mm x 0 recovery was always slightly below that. Although this is a "busy" plot, it illustrates well the inconsistencies of coarse particle recovery (plus 0.5 mm) relative to the finer sizes based on the various operation parameters as shown in Table 3-1.

A hump shaped curve is typical for coal, as confirmed by the optimum recovery for the sizes from 0.045 mm to 0.25 mm. Figure 3-8 shows that some tests had a much lower recovery for each size class than other tests, while some dropped off only at the coarser sizes. The 0.5 x 0.25 mm size class was chosen since it was the coarsest size that showed the potential for reasonable recoveries on the Stockton seam coal. The combined 0.25 mm x 0 size class was chosen since it is also a relatively coarse top-size for flotation and also includes the effects of fine coal in the feed. The combination 0.5 mm x 0 was chosen to show the overall results of floating coarse and fine particles together. The variations in recoveries from the 0.5 x 0.25 mm material alone can often be misleading unless the total effect, including the effect of fines, is also considered. For example, high recovery of the coarser coal particles can result in an excessive amount of ash in the finer particle size of the product. To determine the relationships of the operating parameters that may be causing variations in recovery, the test results and parameters were subjected to statistical analysis using the Design Expert package for the computations.

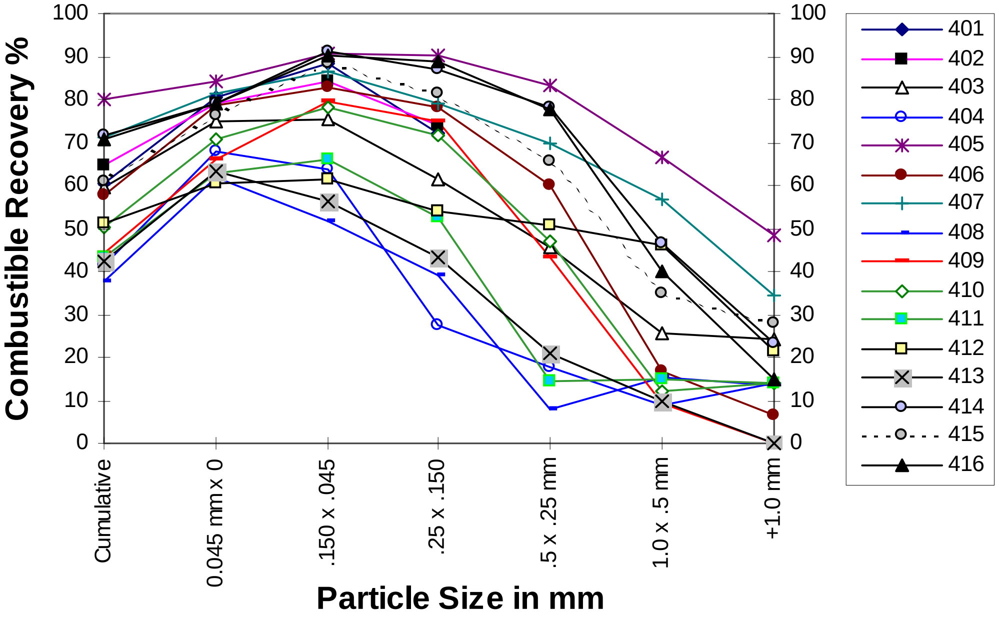


Figure 3-8. First series (401-416) combustible recovery by particle size.

Complete statistical results are found in Appendix B which also develop the predictive models. In most cases the quadratic model provided the best fit. To better grasp the effect of variables, only three variables were considered at one time. Although several of the parameters were investigated, those with the most significant effect on combustibles recovery were feed rate (kg/min), frother dosage (ml/min), and diesel dosage (grams per metric tonne, g/T). The parametric model fits had R-squares of at least 0.94 for all three size classes investigated.

Table 3-1. First Parametric Tests (401-416) results and operating parameters.

Test #	Total Ash % Feed	Conc.	Tails	% Solids Feed	Conc.	Tails	% Wt. Yield	Comb. Rec.%	Ash Rej.%	Comb. Sep Eff. %
401	34.36	12.06	52.65	7.96	13.40	5.60	45.06	60.37	84.18	44.55
402	36.07	9.47	58.54	7.62	15.77	4.88	45.79	64.84	87.98	52.82
403	40.67	9.03	60.75	11.22	11.95	6.62	38.82	59.53	91.38	50.91
404	36.76	8.51	48.11	13.73	12.63	10.00	28.66	41.47	93.36	34.83
405	38.50	10.73	72.58	7.11	16.79	5.02	55.10	79.98	84.64	64.62
406	39.85	10.64	58.34	17.23	17.51	7.73	38.76	57.59	89.65	47.24
407	41.40	11.67	68.07	11.93	21.53	6.93	47.29	71.28	86.67	57.95
408	41.00	8.86	51.37	14.33	13.00	8.70	24.39	37.68	94.73	32.41
409	37.92	8.84	50.42	9.97	12.95	8.52	30.06	44.14	92.99	37.14
410	40.21	9.48	55.41	10.07	15.68	6.46	33.09	50.10	92.20	42.30
411	41.91	8.80	54.59	12.98	12.13	8.40	27.69	43.48	94.19	37.66
412	39.65	9.71	55.22	10.69	13.87	9.04	34.21	51.19	91.62	42.81
413	39.99	13.91	50.96	14.52	9.80	10.88	29.61	42.48	89.70	32.18
414	44.65	12.64	71.30	9.09	16.42	5.62	45.43	71.71	87.14	58.84
415	46.02	11.70	66.51	10.38	18.60	5.00	37.38	61.15	90.50	51.65
416	43.62	12.57	69.22	7.44	17.73	5.85	45.19	70.08	86.98	57.05

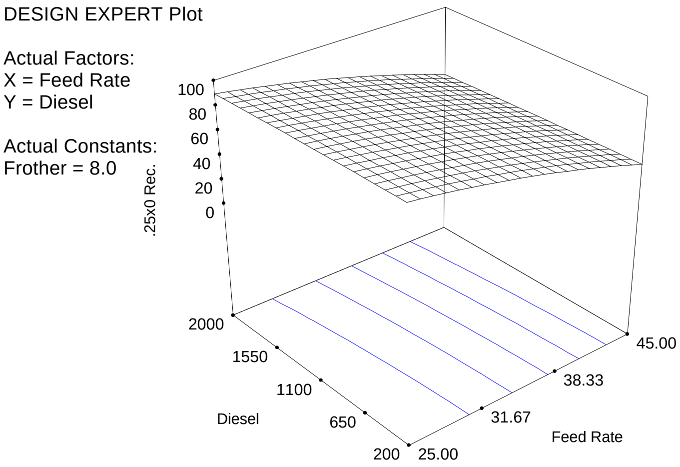
Test #	Feed Rate (gpm)	Feed Rate (kg/min)	Conc Rate (kg/min)	Conc Rate (Tph/m2)	Conc Rate -.5 mm	Wash Water lpm	Air Rate lpm	Air Rate (cms)	Air Fraction %	Frother (ml/min)	Collect (Diesel) (ml/min)	Coll. (g/T)	Coll. -.5 mm (g/T)	Feed Sump (gpm)	Froth Depth (inch)
401	100	31.0	14.0	1.84		48	544	2.0	17	10.5	30	690	920	110	24
402	100	29.7	13.6	1.79		68	465	1.7	15	11.6	30	721	961	110	28
403	100	43.7	17.0	2.23	2.00	68	425	1.6	17	11.6	30	518	696	104	28
404	80	42.8	12.3	1.61	1.55	55	425	1.6	20	11.6	30	423	552	104	24
405	80	22.2	12.2	1.61	1.32	55	425	1.6	13	11.6	48	1307	1805	104	28
406	60	40.3	15.6	2.05	1.78	55	425	1.6	13	24	48	539	753	104	30
407	60	27.9	13.2	1.73	1.32	55	425	1.6	17	10	30	487	707	104	28
408	80	44.7	10.9	1.43	1.41	68	425	1.6	18	10	15	201	254	105	26
409	100	38.8	11.7	1.54	1.49	68	425	1.6	?	11	18	346	445	105	22
410	100	39.2	13.0	1.71	1.60	68	425	1.6	?	11	30	571	745	105	24
411	90	45.5	12.6	1.66	1.61	68	425	1.6	?	11	30	405	530	115	26
412	103	42.9	14.7	1.93	1.74	82	425	1.6	?	10	40	731	1007	103	Turb
413	80	45.2	13.4	1.76	1.63	68	408	1.5	?	10	30	520	758		32
414	80	28.3	12.9	1.69	1.35	67	408	1.5	?	8	30	831	1249		32
415	80	32.3	12.1	1.59	1.29	65	425	1.6	?	8	45	1091	1521		32
416	95	27.5	12.4	1.64	1.38	64	425	1.6	?	8	45	1282	2020		32

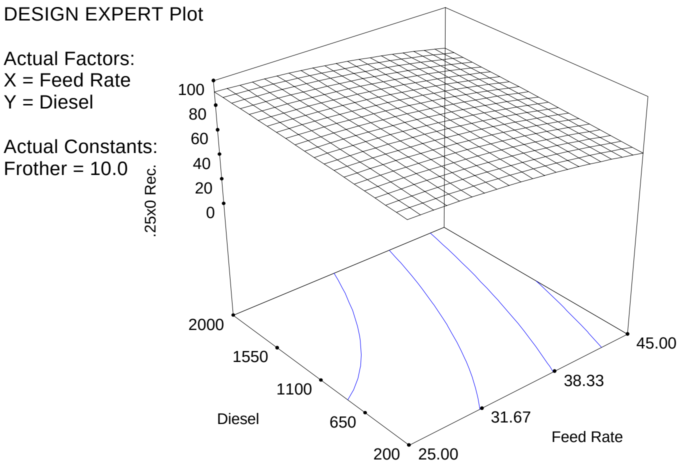
First Series – 0.25 mm x 0

The predictive models in the statistical package help to determine the effect each variable has on the response, which in this case is combustibles recovery. An R-Squared of 0.942 (indicating a relatively good fit) was calculated for the quadratic model whose equation in terms of actual factors is shown below:

$$
\begin{array}{lll}
\text{0.25 mm x 0 Recovery} & = & \\
 & +240.61 & \\
 & -3.48 & * \text{ Feed Rate} \\
 & -15.47 & * \text{ Frother} \\
 & -0.030 & * \text{ Diesel} \\
 & -0.044 & * \text{ Feed Rate}^2 \\
 & -0.27 & * \text{ Frother}^2 \\
 & -6.154E-07 & * \text{ Diesel}^2 \\
 & +0.53 & * \text{ Feed Rate} * \text{Frother} \\
 & -4.646E-05 & * \text{ Feed Rate} * \text{Diesel} \\
 & +3.917E-03 & * \text{ Frother} * \text{Diesel}
\end{array}
$$

The coefficients shown in Table 3-2 are equivalent to those in the above equation except that they are based on the coded factors (i.e. the top feed rate is +1 and the lower is –1. An advantage of using the coded factor coefficients is that the values are directly comparable rather than being related to the units of measure as in the actual factor values of the above equation. The size of the coded coefficients relate directly to the observed change in the response and thus from Table 3-2, one can see that the feed rate has the major effect on recovery. The negative sign (-12.45) for the feed rate coefficient indicates that recovery decreases with increasing feed rate. The Prob>|t| column gives the probability of getting a coefficient as large as that observed, when the true coefficient equals zero. In other words, small values indicate significant coefficients in the model. On that basis the feed rate is the major determinant for recovery and the frother is a distant second within the range of actual factors tested. This corresponds to previous flotation column experience, except that air rate can also play a role if varied significantly.





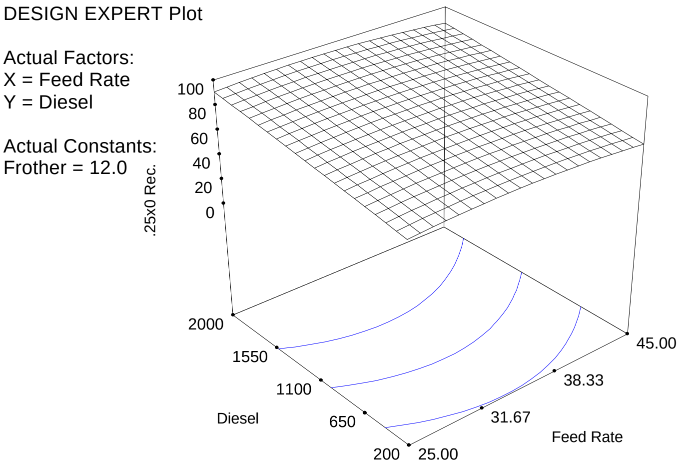


Figure 3-9. 3-D Response Plots, First Series (401-416), 0.25 mm x 0, Frother at 8, 10, and 12 ml/min.

Table 3-2. Response surface model coefficients and significance, First Series, 0.25 mm x 0.

Factor	Coefficient Estimate	DF	Standard Error	T for H_0 Coeff=0	Prob>|t|
Intercept	77.95	1	3.66		
A-Feed Rate	-12.45	1	3.56	-3.49	0.0130
B-Frother	4.27	1	2.80	1.52	0.1787
C-Diesel	5.32	1	5.97	0.89	0.4067
A^2	-4.38	1	5.59	-0.78	0.4637
B^2	-1.09	1	1.24	-0.88	0.4140
C^2	-0.50	1	11.14	-0.045	0.9658
AB	10.70	1	6.78	1.58	0.1655
AC	-0.42	1	9.90	-0.042	0.9677
BC	7.05	1	11.69	0.60	0.5684

In the first plot of the 0.25 mm x 0 size model as illustrated in 3D plots (First Series 401-416, Figure 3-9), one can see that at a low frother dose (8 ml/min) the increased feed rate reduced the recovery. This was as expected since the larger bubbles which occur at a low frother dosage have a limited surface area and are quickly overloaded, restricting their carrying capacity for coal. A medium frother dose indicated the same performance except that at the higher feed rate, recovery improved over that with the lower frother dose. At a high frother dose (12 ml/min) little change in recovery was noticed with changes in feed rate, indicating sufficient bubble surface area to carry the range of coal particles available in the feed slurry. Diesel dosage had basically little effect on flotation of the 0.25 mm x 0 coal, except that some improvement in recovery was predicted for higher frother dosages. This is probably because the smaller bubble size and the increased collector dosage together increased the flotation rate constant and provided extra bubble carrying capacity. The increased rate constant and bubble capacity were sufficient to collect middlings particles previously rejected.

First Series – 0.5 x 0.25 mm

The predictive quadratic model for the coarser size had an R-Squared of 0.969 (indicating an excellent fit, 1.0 being ideal). The predictive equation in terms of actual factors is:

0.5 x 0.25 mm Recovery =
+995.35
-12.56 * Feed Rate
-87.76 * Frother
-0.56 * Diesel

-0.15 * Feed Rate^2
-0.11 * Frother^2
+5.096E-05 * Diesel^2
+1.70 * Feed Rate * Frother
+4.745E-03 * Feed Rate * Diesel
+0.033 * Frother * Diesel

The coded factor coefficients shown in Table 3-3 are directly comparable rather than being related to the units of measure as in the actual factor values in the above equation. From the coefficients, it can be seen that although feed rate and frother dosage had an effect on recovery, diesel dosage definitely had the major effect. The Prob>|t| column (p values) shows that the most significant factors were frother and diesel and the interactions of frother and diesel. At the highest feed rates one would expect increased diesel to have little effect on coarse particle recovery due to insufficient available bubble surface area and the preferential loading of the finer particles, Figure 3-10 however, predicts that at the highest feed and diesel dosages, recovery of the 0.5 x 0.25 mm material increases. This deviation from other experiences may be explained by the higher p value for feed rate in the presented model which indicates that the feed rate is not a good predictor of 0.5 x 0.25 mm recovery in this test series.

When viewing the 3D plots for the predictive model of 0.5 x 0.25 mm size range (Figure 3-10), one notices differences from the plot of the smaller particle size range. At the lower frother dose (8 ml/min) the combustible recovery is highest at the low feed rate just as for flotation of the smaller size range. Unlike the smaller size, however, diesel fuel dosages had a major effect on recovery of the coarse particle size. At low frother and low feed rate, the recovery actually dropped with increased collector addition. This was probably due to the excess diesel, above that needed to coat the coal, encumbering the frother and causing increased bubble size with less bubble surface area.

It is well known that the fines preferentially attach to the bubble surface and that coarse particles remain attached only if there is sufficient bubble surface area available after attachment of the fines. At the medium frother dosage of 10 ml/min and a low diesel dosage, the relationship between feed rate and recovery was similar to that of low frother dosage; that is, increased feed rate meant lower recovery. At the low feed rate, increasing diesel dosage appeared to lower recovery due probably to the decreased effectiveness of the frother as described above. At the highest diesel dosages the recovery increased again due to the increased particle hydrophobicity brought about by the high amounts of collector available in the slurry. Response plots in Figure 3-10 are based totally on the model equation and thus indicate recoveries above 100%, which of course is impossible. The plots are useful, nonetheless, to show the magnitude of the factor effects.

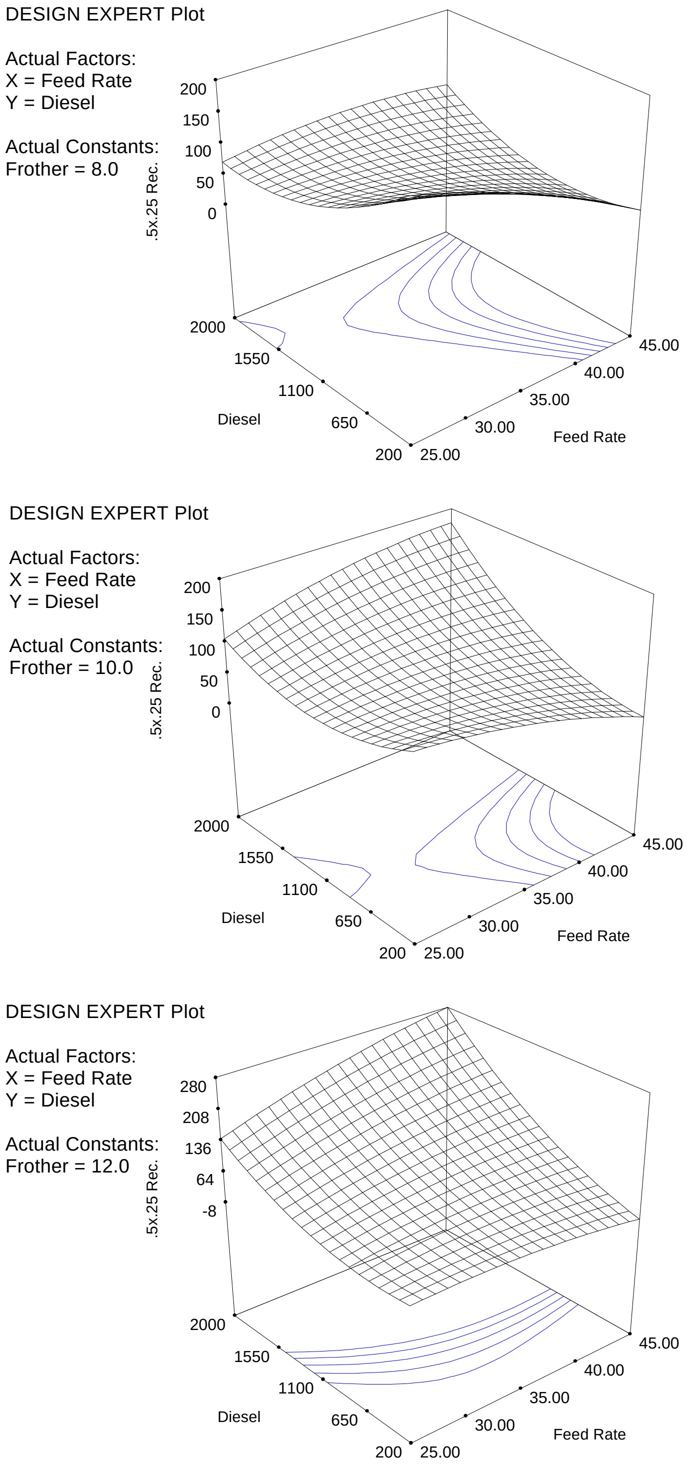


Figure 3-10. 3-D Response Plots, First Series (401-416), 0.5 x 0.25 mm, Frother at 8, 10, and 12 ml/min.

Table 3-3. Response surface model coefficients and significance, First Series, 0.5 x 0.25 mm.

Factor	Coefficient Estimate	DF	Standard Error	T for H_0 Coeff=0	Prob>|t|
Intercept	72.02	1	6.31		
A-Feed Rate	-7.26	1	6.13	-1.18	0.2814
B-Frother	11.01	1	4.83	2.28	0.0628
C-Diesel	42.91	1	10.29	4.17	0.0059
A^2	-14.87	1	9.64	-1.54	0.1738
B^2	-0.42	1	2.13	-0.20	0.8491
C^2	41.28	1	19.21	2.15	0.0752
AB	34.05	1	11.68	2.91	0.0268
AC	42.71	1	17.07	2.50	0.0464
BC	58.59	1	20.15	2.91	0.0271

At a high frother dosage (12 ml/min) combustible recovery appeared to have been affected only by the diesel dosage. At the low diesel dosage the recovery of the coarse coal was depressed, probably due to the excess frother "wetting" the surfaces of the coal particles. At higher diesel dosages the coal surfaces are not "wetted" by the frother and maximum recovery was projected by the 3D model.

Figure 3-8 indicates that for some conditions the 0.5 x 0.25 mm material can be recovered nearly as well as the finer material. The actual size-by-size recoveries shown also illustrate the reason that flotation is seldom, if ever, utilized for coal coarser than 0.5 mm particle size. Even with the best combination of parameters, the combustible recovery began to drop off rapidly above 0.5 mm particle size. However, this plot does show that for most of the tests, the 0.25 x 0.150 mm (60 x 100 mesh) fraction floated as well or better than any size fraction.

First Series – 0.5 mm x 0

Results of the 0.5 mm x 0 combination are, as would be expected, between that of the minus and plus 0.25 mm fractions. The statistical data for the total 0.5 mm x 0 is essentially a combination of that shown for the 0.25 mm x 0 and the 0.5 x 0.25 mm material, and is therefore shown only in Appendix B.

Parametric Tests - Second Series

Enough was learned about the operating parameters from the preliminary testing to determine the most likely parameter settings for further parametric testing. These settings became the midpoints for a Box-Behnken experimental design. The intent of the second series of parametric testing was to determine the effect of bubble size and air fraction on coarse coal recovery. To do this, air volume and frother dosage were varied. Since any slight variation in feed volume could cause the bubbles to be more or less loaded and therefore affect recovery, feed rate was also intentionally used as a variable. Table 3-4 gives the settings and testing order for the Box-Behnken design.

Although earlier testing had shown that the diesel fuel dosage also affected the coarse coal recovery, the intent was to remove it as a variable by holding the diesel dosage relatively constant. This was performed by feeding a different amount of diesel for each of three volumetric slurry feed rates (40, 50, and 60 gallons per minute (gpm)), which would provide a constant g/tonne diesel dosage even as feed flow was varied. The percent solids in the feed slurry was to be held constant at 10%, but due to variations in plant feed, screen wear on the feed system, and raw coal pumping surges (all of which were unique to this test series), the actual percent solids of feed to the pilot column varied considerably. The percent solids variations of from 7 to 14% had a major impact on the diesel dosage as well. Although the volumetric amount was held constant for a given feed flow, the grams per tonne of feed dosage varied with the percent solids changes. The diesel fuel tended to first coat the fine coal particles due to the much higher amount of surface area. Since only the remaining diesel fuel was then available for the coarser particles, any variation in the amount of fine coal caused the amount of diesel available for coarse particles to vary considerably. Since coarse coal recovery was very sensitive to diesel dosage, the inconsistencies in coarse recovery may be due to this unintended variation in the ratio of diesel to solids.

Table 3-4. Test matrix for the Second Series parametric test.

Run Number	Feed Rate	Air	Frother
1	Low	Medium	Low
2	High	High	Medium
3	Medium	Low	High
4	Medium	High	High
5	Medium	High	Low
6	High	Medium	High
7	Medium	Medium	Medium
8	Medium	Medium	Medium
9	Medium	Low	Low
10	High	Low	High
11	Low	Low	Low
12	High	Medium	Low
13	Low	High	Low
14	Low	Medium	Medium
15	Medium	Medium	Medium

The same three major size classes were considered for the second parametric test series as for the First Series, that is, 0.25 mm x 0 (60 mesh x 0), 0.5 x 0.25 mm, and 0.5 mm x 0. Test results were entered into the Design Expert statistical computer program. The initial variables entered into the program were the design parameters, air rate, frother, and feed flow. Although the 0.25 mm x 0 quadratic predictive model had an R-Squared of 0.946, the only significant factor for the recovery response was feed rate, and that was minor.

When the program's predictive quadratic model was used to develop the 3D response plots for the 0.25 mm x 0 fraction (Second Series, Appendix B), variations in feed, air, and frother (within the test ranges) were found to have very little affect on the fine coal recovery. For each of the three frother dosages, the lowest recovery is at maximum feed flow and air. At these conditions the column would be most turbulent, which may explain the poorer recovery.

Although the quadratic model is the best for the 0.5 x 0.25 mm fraction the p-value is 0.11 and the R-Squared is 0.897. The only significant coefficient in the model is frother but it shows a decrease in recovery with increasing frother which is not as would normally be expected. Review of the 3D predictive plots for the coarser coal (0.5 x 0.25 mm) showed the results to be more erratic. At a low frother dosage the model predicted a higher recovery for the higher feed rate. This was contrary to normal flotation results since higher feed rates tend to overload the froth, causing lost recovery. The medium frother dosage showed a similar result although not as pronounced. It is obvious that either these results are unique or that something else was happening that would account for the deviation from predictions based on prior experience. At a low feed rate and high air flow, there was a high recovery of coal as one would typically expect. At the high frother dosage the response plot also looked typical with a much higher recovery of the 0.5 x 0.25 mm coal at the low feed rate, high air corner. This was expected since even though coarse coal can attach easily to a bubble, there are many hydrodynamic situations that can cause detachment of the coarse particles. One the other hand, once attached, fine coal is difficult to detach from a bubble under most conditions.

The predicted results for the combined 0.5 mm x 0 particle size range (Second Series) essentially take the shape of the finer size (0.25 mm x 0) plots since the majority of the coal is in that size range.

The tight grouping of the test points for the 0.25 mm x 0 material in Figure 3-11 show that, as in the First Series, there was little variation in recovery for the finer material in the Second Test Series.

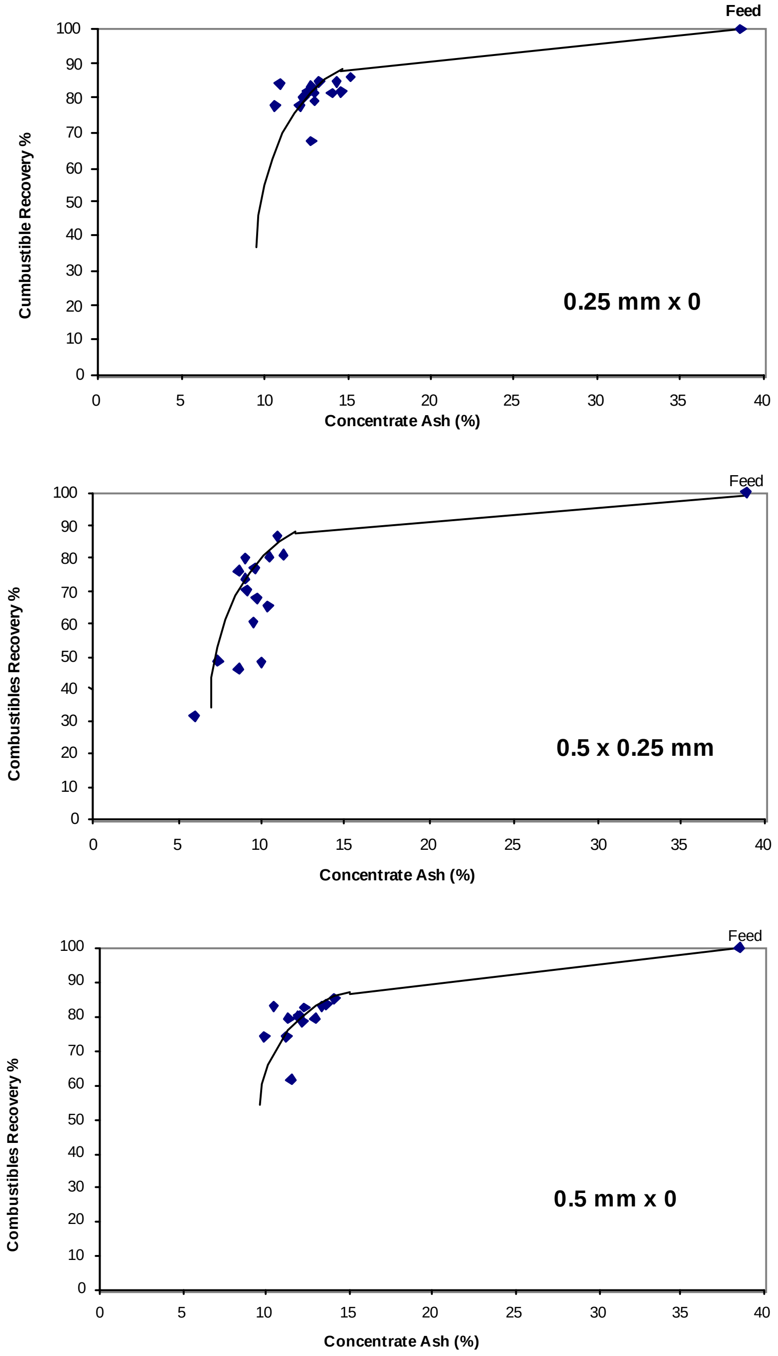


Figure 3-11. Second Series grade-recovery plots by size for tests 451-465.

Figure 3-11, however, shows considerable movement up and down the grade-recovery curve for the 0.5 x 0.25 mm material. The lower section of the figure illustrates the heavy participation of 0.25 mm x 0 material in the overall flotation results for the combined 0.5 mm x 0 fraction.

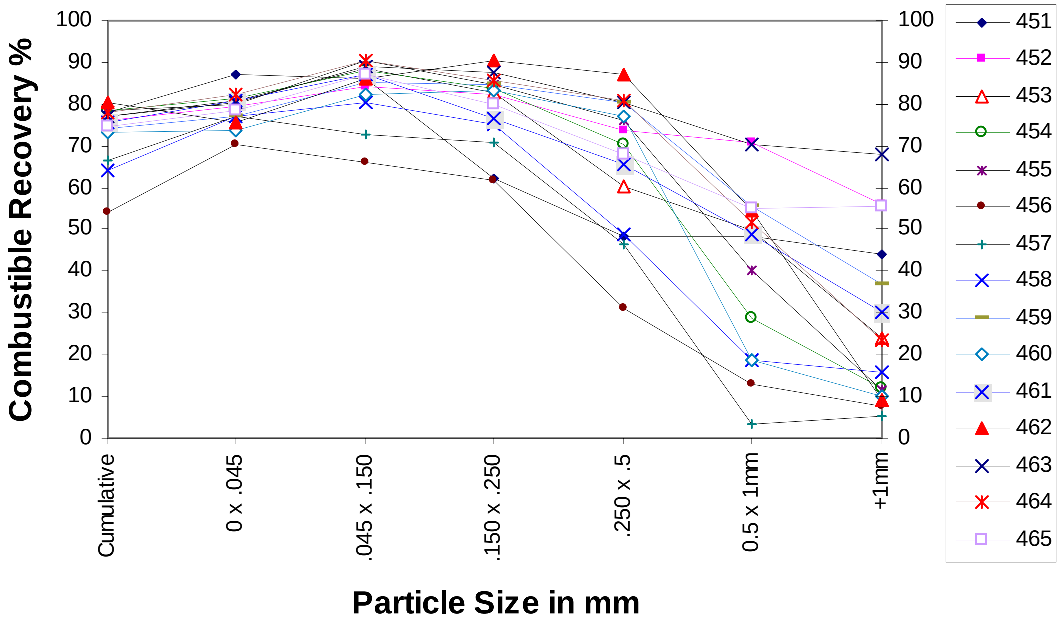


Figure 3-12. Second Series (451-465) combustible recovery by particle size.

Even more than in the First Series of coarse flotation, this Second Series showed that on a size-by-size basis the 0.045 to 0.250 mm material had the best flotation recovery with only a slight drop for the 0.5 x 0.25 mm material (Figure 3-12). The recovery of the 1 x 0.5 mm material varied considerably, although some tests had remarkably high recoveries for material this coarse. Figure 3-12 shows that regardless of significant changes in operating parameters, coal up to 0.250 mm can float as well as up to 0.150 mm. It also shows that the 0.5 x 0.25 mm coal can be expected to float reasonably well if proper attention is paid to column sizing and operation.

Table 3-5. Second Series Parametric Tests (451-465) results and operation parameters.

Test #	Ash %			Solids %			% Wt. Yield	Comb. Rec.%	Ash Rej.%	Comb. Sep Eff. %
	Feed	Conc.	Tails	Feed	Conc.	Tails				
451	36.95	11.18	67.83	7.32	12.35	3.11	54.50	76.79	83.52	60.30
452	38.62	11.24	69.13	7.06	15.40	5.23	52.71	76.22	84.66	60.88
453	36.29	11.23	67.32	8.27	12.73	3.40	55.31	77.08	82.89	59.97
454	34.26	11.43	66.05	6.76	12.31	2.33	58.20	78.42	80.59	59.00
455	33.87	10.08	66.29	7.31	15.90	2.76	57.68	78.43	82.84	61.26
456	35.63	11.12	51.39	7.35	14.55	11.38	39.14	54.04	87.78	41.82
457	42.61	10.60	66.45	14.18	12.32	6.28	42.69	66.50	89.38	55.88
458	34.70	9.33	56.41	13.16	15.57	7.64	46.10	64.02	87.61	51.63
459	38.86	10.66	68.06	13.43	12.01	7.44	50.88	74.34	86.05	60.39
460	40.24	11.52	68.43	10.10	15.51	5.75	49.54	73.35	85.81	59.16
461	42.80	11.78	72.56	9.19	14.17	3.86	48.97	75.52	86.52	62.03
462	36.31	12.63	69.73	12.24	10.47	4.81	58.53	80.29	79.64	59.93
463	45.02	12.08	75.53	13.24	17.25	8.66	48.10	76.90	87.09	63.99
464	42.44	12.60	74.06	10.08	11.50	4.90	51.44	78.11	84.74	62.85
465	42.30	12.68	75.10	13.90	13.77	6.32	52.55	79.52	84.25	63.77

Table 3-5 cont'd.

Test #	Feed Rate (gpm)	Feed Rate (kg/min)	Conc Rate (kg/min)	Conc Rate Tph/m2	Conc Rate -.5 mm	Wash Water lpm	Air Rate lpm	Air Rate cms	Air Fraction %	Frother (ml/min)	Collect (Diesel) (ml/min)	Coll. (g/T)	Coll. -.5 mm (g/T)	Froth Depth (inch)
451	41	11.7	6.4	0.84	0.73	54.6	323	1.2	5.0	4.5	16.5	1107	1358	24
452	60	16.5	8.7	1.14	1.03	47.8	350	1.3	3.5	7.3	24	1141	1383	20
453	50	16.1	8.9	1.17	1.02	40.9	306	1.1	3.5	9.3	20	974	1171	26
454	50	13.2	7.7	1.01	0.87	43.7	350	1.3	2.0	9.3	20	1191	1445	24
455	50	14.2	8.2	1.08	0.98	43.7	350	1.3	3.3	5.5	20	1102	1348	28
456	60	17.2	6.7	0.88	0.83	43.7	329	1.2	3.3	11.0	24	1096	1385	25
457	50	27.6	11.8	1.55	1.38	51.8	329	1.2	11.0	8.0	20	568	699	29
458	50	25.6	11.8	1.55	1.37	51.8	329	1.2	11.9	8.0	20	612	765	30
459	50	26.2	13.3	1.75	1.54	47.8	306	1.1	9.3	6.0	20	600	749	27
460	60	23.6	11.7	1.54	1.37	47.8	306	1.1	10.0	9.0	23	764	1041	28
461	40	14.3	7.0	0.92	0.76	47.8	306	1.1	11.0	6.0	16	876	1128	25
462	60	28.6	16.7	2.20	1.90	54.6	329	1.2	9.5	6.3	23	631	745	27
463	40	20.6	9.9	1.31	1.04	50.5	350	1.3	12.9	6.0	16	608	834	25
464	40	15.7	8.1	1.06	0.86	50.5	329	1.2	13.6	7.5	16	799	1136	25
465	50	27.1	14.2	1.87	1.66	50.5	329	1.2	11.3	7.5	19	550	724	28

Second Series Revisited

After extensive review and cross plotting of the variables and other operating parameters, the question of inconsistent results from tests 451-465 was resolved. The major problem stemmed from the uncontrollable variation in percent solids of the feed. Figure 3-13 is a plot of the effect diesel dosage (on a gram per tonne basis, g/T) had on the air fraction. It indicated that above a threshold value of diesel (~1200 g/T for this system), the air fraction dropped rapidly. A decrease in air fraction from the 10 to 13% range to below 4% indicated much larger air bubbles were being formed and thus a much lower bubble surface area was available for the attachment of coarse coal. Since fine coal was more strongly attached to the bubble surfaces than coarse coal, the coarser particles were the first to be lost when particle loading on the bubbles became high. The larger bubbles may also have caused increased turbulence that also resulted in detachment of the coarse particles. Feed solids versus diesel fuel dosage has been plotted on the right axis of Figure 3-13. Since the intent was to hold a constant diesel dosage on the assumption of a constant percent feed solids, it is no surprise that there is an unintended linear correlation between feed solids and diesel dosage.

The statistical analysis was re-evaluated using diesel fuel dosage, frother dosage, and feed rate as variables. In the previous analysis of this series, air flow was found to have a very small effect and could be dropped to allow room for diesel dosage to be evaluated in the statistical model. With only 15 test points, 3 variables are the most that can be evaluated using the Box-Behnken design.

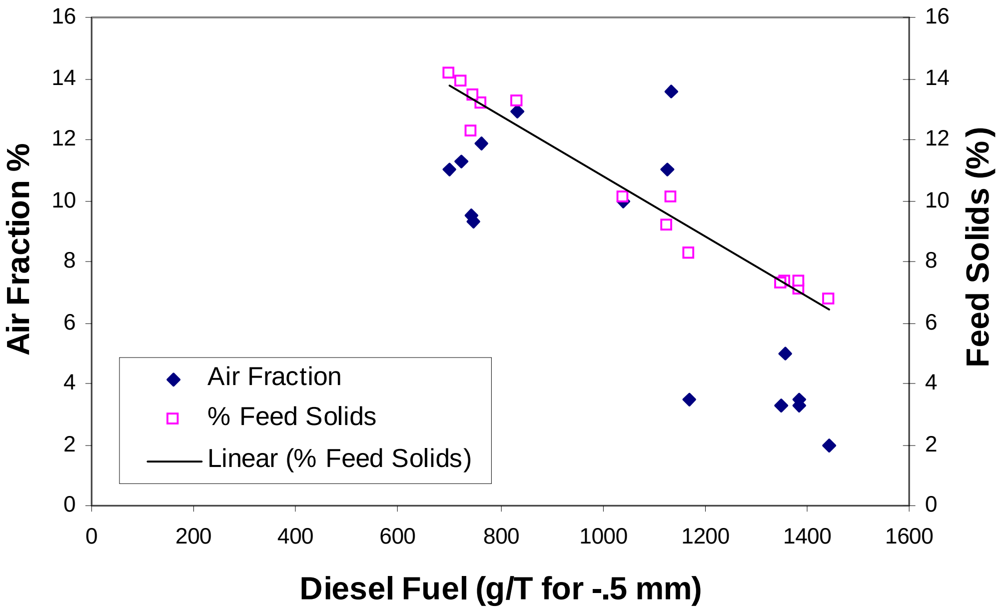


Figure 3-13. Relationship of air fraction, diesel dosage, and % feed solids for the Second Series (451-465).

Second Series Revisited – 0.25 mm x 0

The quadratic model for the 0.25 mm x 0 fraction had a R-Squared of 0.867 while the predicted equation in terms of actual factors is:

$$
\begin{aligned}
\text{0.25mm x 0 Recovery} = \; & +37.98 \\
& -0.53 \quad * \text{ Feed GPM} \\
& +0.044 \quad * \text{ Diesel} \\
& +13.05 \quad * \text{ Frother} \\
& +6.636\text{E-}03 \quad * \text{ Feed GPM}^2 \\
& -1.501\text{E-}05 \quad * \text{ Diesel}^2 \\
& -1.54 \quad * \text{ Frother}^2 \\
& -9.466\text{E-}04 \quad * \text{ Feed GPM} * \text{Diesel} \\
& +0.086 \quad * \text{ Feed GPM} * \text{Frother} \\
& +4.426\text{E-}03 \quad * \text{ Diesel} * \text{Frother}
\end{aligned}
$$

Table 3-6. Response surface model coefficients and significance, Second Series Revisited, 0.25 mm x 0.

Factor	Coefficient Estimate	DF	Standard Error	T for H_0 Coeff=0	Prob>|t|
Intercept	86.29	1	3.38		
A-Feed GPM	-2.19	1	1.68	-1.30	0.2643
B-Diesel	-0.63	1	2.67	-0.24	0.8237
C-Frother	-1.90	1	1.75	-1.09	0.3379
A^2	0.66	1	2.70	0.25	0.8180
B^2	-1.84	1	2.78	-0.66	0.5442
C^2	-5.29	1	3.67	-1.44	0.2236
AB	-3.31	1	2.85	-1.16	0.3094
AC	1.59	1	4.94	0.32	0.7639
BC	2.87	1	2.50	1.15	0.3150

The 3D response plots for this evaluation are relatively flat and thus are shown only in Appendix B. From the 0.25 mm x 0 plots of the predicted coal recovery, the changes in recovery due to differences in frother dosage were small. The best performance was at the medium frother dosage while the lowest recovery was found at the extremes of high diesel dosage, high feed rate, and low frother dosage. At a low feed rate, diesel dosage accounted for a slight recovery increase at all but the lowest frother dosage. However, differences in recovery for this fine coal fraction are small and thus it is unreliable to predict a recovery response within the range of operating parameters tested.

Second Series Revisited – 0.5 x 0.25 mm

A much broader range of response in the prediction model was seen in the combustible recovery of the 0.5 x 0.25 mm fraction 3-D plots (Figure 3-14). A change in recovery at low diesel dosages was the most significant variation observed. Recovery dropped considerably at all feed rates with increasing frother dosage. The decrease in recovery at high frother dosage was possibly due to "wetting" of the coarse coal by the excess frother which reduced their hydrophobicity sufficient to allow the particles to drop back into the pulp from the froth. At the higher frother dosages, increasing the diesel dosage improved the recovery by overcoming the effects of excess frother. An unexpected response was the increase in recovery with increasing feed flow and low diesel. Increased feed rate normally decreases recovery due to bubble surface overload, but in this case the increased feed flow may have diluted the frother and reduced its negative effect.

The p values of Table 3-7 indicate that the only significant factor in the model for the 0.5 x 0.25 mm material was frother and it is only weakly so. As discussed earlier, the Second Series results were not nearly as reliable as those of the First Series. The quadratic model for the 0.5 x 0.25 mm fraction had a R-Squared of 0.762 while the predicted equation in terms of actual factors is:

$$\begin{aligned}
\text{0.5 x 0.25 Recovery} = \\
& -30.42 \\
& +0.18 && * \text{ Feed GPM} \\
& +0.14 && * \text{ Diesel} \\
& +12.71 && * \text{ Frother} \\
& +2.429\text{E-}03 && * \text{ Feed GPM}^2 \\
& -9.049\text{E-}05 && * \text{ Diesel}^2 \\
& -4.58 && * \text{ Frother}^2 \\
& -2.579\text{E-}03 && * \text{ Feed GPM} * \text{Diesel} \\
& +0.39 && * \text{ Feed GPM} * \text{Frother} \\
& +0.026 && * \text{ Diesel} * \text{Frother}
\end{aligned}$$

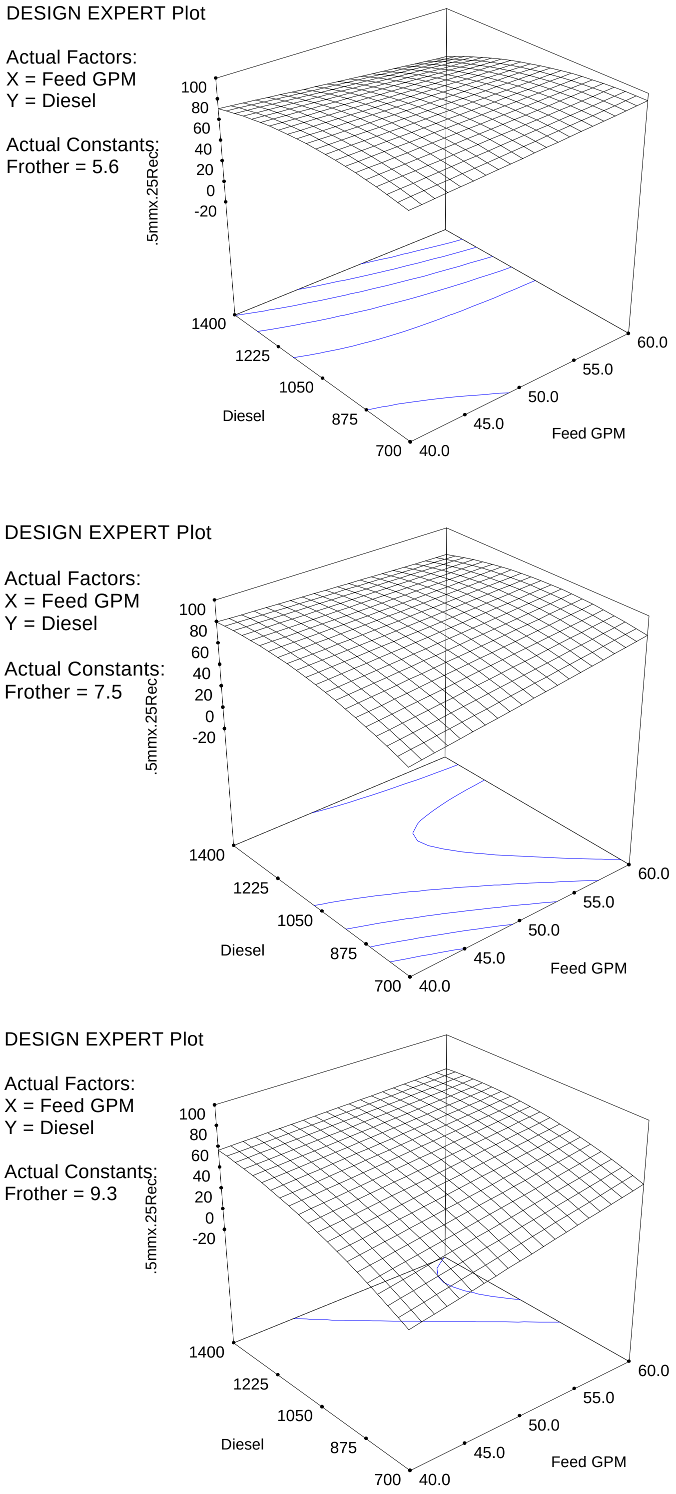


Figure 3-14. 3-D Response Plots, Second Series Revisited (401-416), 0.5 x 0.25 mm, Frother at 5.6, 7.5, 9.3 ml/min.

Table 3-7. Response surface model coefficients and significance, Second Series Revisited, 0.5 x 0.25 mm.

Factor	Coefficient Estimate	DF	Standard Error	T for H_0 Coeff=0	Prob>|t|
Intercept	83.31	1	15.24		
A-Feed GPM	5.94	1	7.59	0.78	0.4772
B-Diesel	4.77	1	12.01	0.40	0.7112
C-Frother	-16.63	1	7.88	-2.11	0.1023
A^2	0.24	1	12.16	0.020	0.9850
B^2	-11.09	1	15.51	-0.89	0.4255
C^2	-15.67	1	16.55	-0.95	0.3973
AB	-9.03	1	12.83	-0.70	0.5204
AC	7.14	1	22.24	0.32	0.7642
BC	16.78	1	11.24	1.49	0.2099

Second Series Revisited – 0.5 mm x 0

The predicted flotation response of the 0.5 mm x 0 composite was obviously a combination of the two previous size ranges, but it does indicate very well the detrimental effect of excess frother on coarse coal recovery. The statistical data for the total 0.5 mm x 0 is essentially a combination of that shown for the 0.25 mm x 0 and the 0.5 x 0.25 mm material, and is therefore shown only in Appendix B.

All of the test work presented here showed that coarse coal can be floated successfully when proper attention is paid to control parameters such as air, frother, diesel, and feed rate. Laboratory test work on coarse particles is always difficult due to particle settling, differences in samples, etc. Producing a coarse slip stream in an operating plant is also difficult, as discussed previously. Replication of the major parametric test for this project at the Lady Dunn Preparation Plant was not possible due to the onset of construction for a major plant upgrade. Therefore, the 30-inch pilot-scale column was removed from the plant shortly after completion of the last test series.

3.5 Full Size Flotation Column Scale-Up

The successful testing of the 30-inch pilot column coincided with an expansion project at the Lady Dunn Plant which prompted interest in a full-scale column that would clean coal coarser in size than was currently being sent to flotation.

Lady Dunn Plant Circuit

The evolution of the Lady Dunn Preparation Plant flowsheet parallels the classic history of coal preparation. The original plant was designed and built in 1951 for an easily cleaned metallurgical coal (Cedar Grove seam) produced by old conventional mining methods which also provided coarser feed coals. Newer continuous mining methods brought increased amounts of fines, including high ash clays, which overloaded the fine coal circuits. A change in coal feed to predominantly steam coal (Stockton seam) brought a more difficult-to-clean high-middlings coal along with more reject material. The 550 tph flowsheet of heavy media vessel, Deister tables, and conventional flotation was no longer adequate for the new plant feed material. A heavy media cyclone circuit was installed as part of a recent plant upgrade, but more capacity was needed to handle the planned increase in raw feed tonnage to 1,200 tph.

Plans for the new circuit included a new heavy media vessel for the coarser sizes and an additional heavy media cyclone circuit for the intermediate sizes. As usual, the selection of an appropriate circuit(s) to treat the fine sizes required more study. A two-stage spiral circuit was tested for the minus 1 mm material, but performance of the spiral deteriorated rapidly on the finer sizes, especially below 0.25 mm (60 mesh). Therefore, alternative methods for treating this size fraction were considered.

The original flotation circuit at the Lady Dunn Plant consisted of four 5-cell banks of mechanical flotation cells. The primary flotation feed was the raw minus 100 mesh classifying cyclone overflow with a secondary feed coming from a clean coal sieve effluent. The ash content of the flotation product was relatively high (i.e., 14-16%) and the combustible recovery seldom exceeded 20%. Test work with the pilot-scale column showed that a substantially lower clean coal ash (i.e., 8-10%) and higher recovery (70-80%) was possible by column flotation. High ash slimes normally entrained in flotation froths were easily rejected by the column wash water system. In addition, the ability of the column to treat material from a coarser size-cut of 0.25 mm instead of the normal 0.15 mm (100 mesh) meant that a much more efficient circuit was possible. For these reasons, spirals were installed to treat the 1 x 0.25 mm material and all of the minus 0.25 mm (60 mesh) was sent to column flotation. A simplified version of the new plant flowsheet is shown in Figure 3-15.

Applicable Pilot-Scale Testing

Testing of the pilot-scale unit indicated that the column is capable of recovering coal in all sizes up to at least 0.5 mm. The testwork also showed that the recovery of the coarser particles is sensitive to operating parameters such as air, frother, and feed rate, all of which needed to be addressed in full-scale design. Previous test work conducted at Virginia Tech (Jha et al. 1997) showed that a column froth becomes overloaded when the amount of floated solids per unit of available bubble surface area becomes too high. This condition can occur when the amount of floatable feed solids increases or the bubble surface area is reduced (by reducing frother dosage or air rate). As the bubble surface area is reduced, the recovery and clean coal ash decreases as middlings particles are rejected back into the flotation pulp. Unfortunately, finer particles tend to remain preferentially attached to the bubbles and coarse particles are lost.

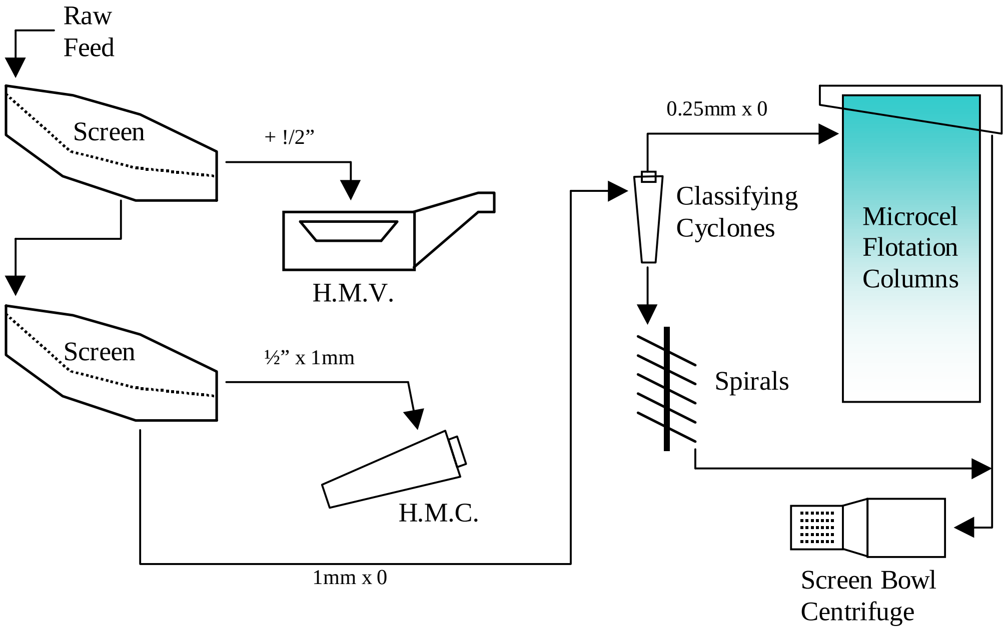


Figure 3-15. Simplified flowsheet (new circuit) for Lady Dunn preparation plant.

As shown in Figure 3-16 feed rate has a major effect on column performance. The upper two (overlapping) curves show that recovery drops with increasing feed rate in nearly identical fashion for both the minus 0.15 mm (100 mesh) and the minus 0.25 mm fractions. These two curves show that column flotation should be considered for minus 0.25 mm sized feeds rather than the minus 0.15 mm feed (100 mesh) traditionally used in coal flotation circuits. The curve for the 0.5 x 0.25 mm material displays a rapid drop in recovery with increasing feed rate for the coarser material. This plot shows that the coarse particles can be recovered by column flotation, but these particles are lost much faster than the fine particles as the column becomes overloaded.

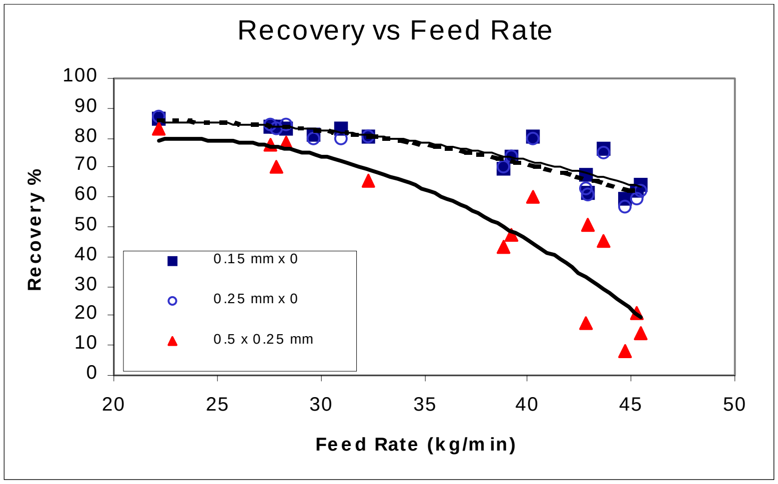


Figure 3-16. Effect of feed rate increase on recovery by particle size for 30-inch pilot-scale column.

The other variable that has a significant influence on coarse coal recovery is the hydrophobicity of the particles. As shown in Chapter 2, the contact angle has a significant effect on the amount of force required to detach a particle and the contact angle is essentially a measure of the hydrophobicity of a surface. The addition of a collector will often increase hydrophobicity and for coal the most economical collector is common diesel fuel. Figure 3-17 shows that recovery increases with increasing diesel dosage (g/tonne) for both the fine (0.25 mm x 0) and coarse (0.5 x 0.25 mm) material. However, the rate of increase in recovery is much higher for the coarser material. Three separate feed rate groupings are shown, i.e., low (20-30 kg/min), medium (30-40 kg/min) and high (40-50 kg/min). The recovery improved very little as the diesel dosage was increased at the lower feed rates, while diesel dosage had a much larger impact on recovery for the higher feed rates. The increased hydrophobicity imparted by the diesel allows for a much stronger bond between the coarse particles and bubbles. This reduces the probability of coarse particle detachment which would otherwise occur at the higher feed rates where the froth carrying capacity may be exceeded.

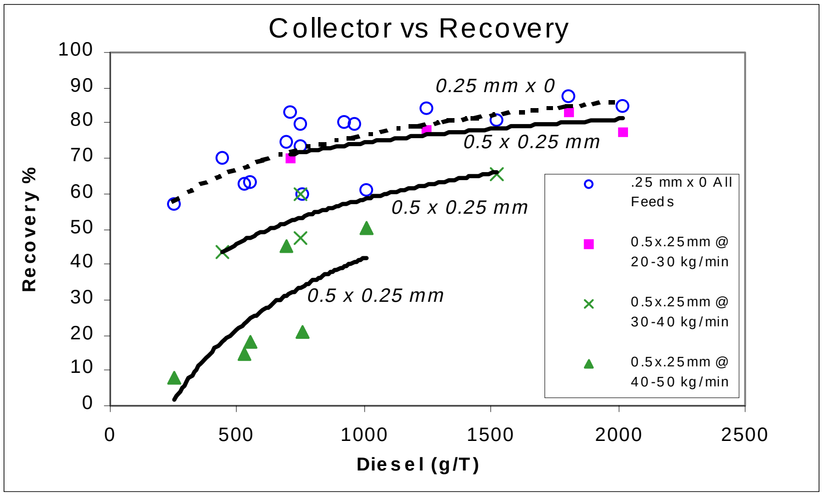


Figure 3-17. Effect of collector on combustibles recovery (pilot-scale).

Full-Scale Column Design

The coal company made the decision for the author to proceed with design of full-scale Microcel columns based on the pilot-scale test work. The first major design concern was the unit capacity since it affected the column diameter and number of columns. From the pilot-scale testwork, clean coal capacity of the froth appeared to be in the range of 1.5 metric tonnes (1.65 short tons) per hour per square meter. This production capacity was in good agreement with the test values reported elsewhere (Luttrell et al. 1993). Equations developed by others, however, are only valid for smaller particle sizes. Eq. 3-2 by Finch and Dobby (1990) would indicate an unrealistic value of 23 Tph/m^2 for froth carrying capacity (C_a) when using a typical d_{80} particle size of 400 µm and a particle S.G. (ρ_p) of 1.40. The d_{80} size refers to the 80% passing particle size of the floatable material rather than the feed size. Although often overused, this equation, as well as the plots by Espinosa-Gomez et al. (1988), have not been validated for particle sizes much above 40 µm. The apparent leveling off of carrying capacity at the larger particle sizes is probably due to the skewedness of the particle size distributions for coal. Although the d_{80} may be relatively coarse, a considerable amount of material and particle surface area is usually found at the smaller diameters.

$$C_a = 0.041 d_{80} \rho_p \qquad \text{Eq. 3-2}$$

Using the 1.5 Tph/m^2, the maximum projected clean coal tonnage of 55 tonne/hr (60 tph) would require 37 square meters of surface area. This area requirement could be achieved with six standard 3-meter diameter columns. In the interest of reducing costs (as well as space limitations), an alternative was sought which led to the design and development of the largest diameter columns known to exist for coal processing. Three 4-meter diameter columns would provide 12.5 square meters each or 37.5 total square meters of surface area. This would be adequate to handle the maximum projected feed to the column circuit. In addition, an internal launder was used to minimize the possibility of immobile froth in the center of such a large column since keeping the froth mobile would help to prevent coarse particle dropback. The internal launder made the maximum froth travel distance less than 1 meter, which is less than that of successful 3-meter diameter columns currently operating in the field. The 4-meter diameter columns provide 77% more area and capacity than 3-meter units at only 25% more cost. Further savings are realized on structural steel, piping, and other costs associated with fewer units. Also, the pump motor horsepower is only 100 hp compared to 75 hp for the 3-meter unit.

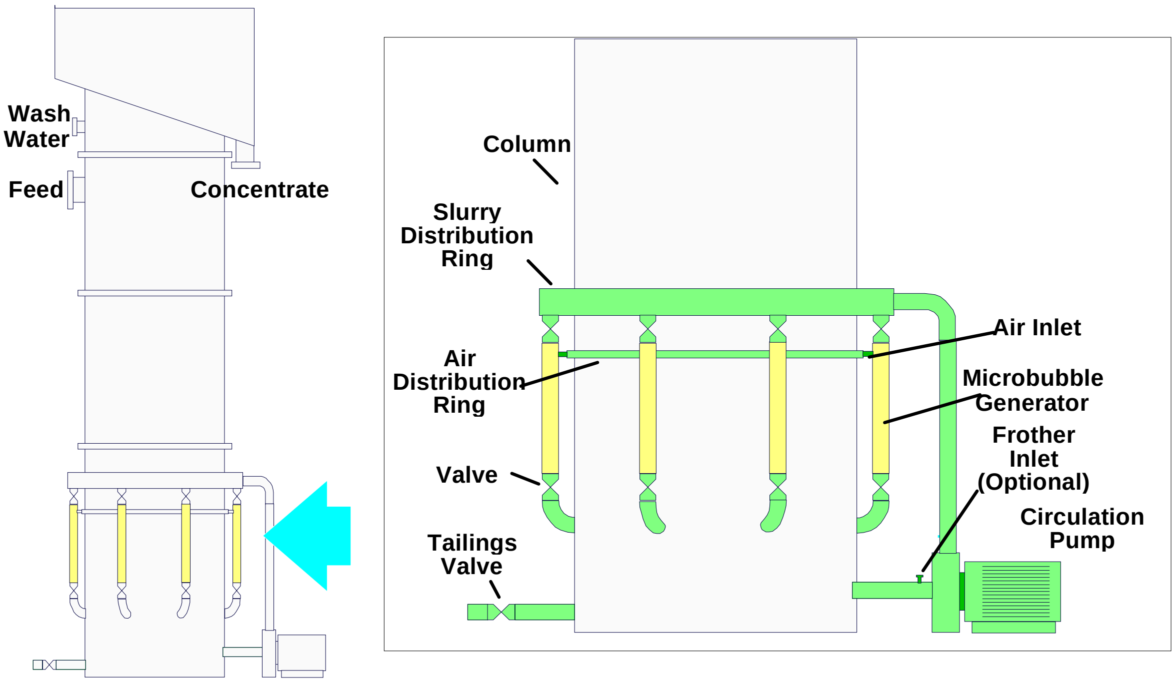


Figure 3-18. Full-scale Microcel$^{\text{TM}}$ column and bubble generation system.

Figure 3-18 depicts the operation of the patented bubble generation system for the Microcel column. The full-scale system is similar to that of the 30-inch unit except that the flow is downward through the spargers rather than upward. The recirculation of column slurry through a sparger with the addition of air and frother provides for a very consistent bubble size in any quantity needed. Level is controlled by a proportional tailings valve via a PID loop controller using a signal from a level transmitter. Detailed descriptions of the Microcel technology have been provided elsewhere (Davis et al. 1992, Jha et al. 1996, Luttrell et al. 1993, Phillips 1997).

Full-Scale Column Results

The column circuit (Figure 3-19) was commissioned in August of 1996 and immediately began producing a product within specifications. The major limitation on column production at the Lady Dunn Plant was froth handling capacity. Initially, the columns had the capability of producing more froth at a satisfactory ash than could be fed to the dewatering circuit. Plans were underway to enlarge the froth handling system as well as modify the screen bowl centrifuges with larger inlets.

Figure 3-19. Full-scale 4-meter diameter column installation at the Lady Dunn Preparation Plant.

Results from initial testing of the 4-meter units show that the performance is similar to that predicted from the pilot-scale unit (Figure 3-20). The data points fall on essentially the same grade-recovery curve with a product ash of less than 10% (well below the 12% ash target). Combustible recoveries of up to 80% have been obtained with the expected normal being 75% at a 10-12 % product ash. These are excellent results for this relatively high middlings coal. In fact, washability data indicates that a 9% cumulative ash would be obtained at 1.70 S.G. float for the 0.25 x 0.15 mm (60 mesh) material. Diesel consumption is

500-800 g/tonne (1.0-1.5 lb/ton), while frother dosage is 270 g/tonne (~0.6 lb/ton) at peak demand. The size-by-size recoveries shown in Figure 3-21 indicate excellent recoveries up to and including particle sizes of 0.5 mm. However, the columns are currently heavily loaded and a high bubble surface area must be maintained if a high combustible recovery is to be obtained. Figure 3-21 illustrates this fact by showing higher recoveries for all sizes at the higher frother dosage.

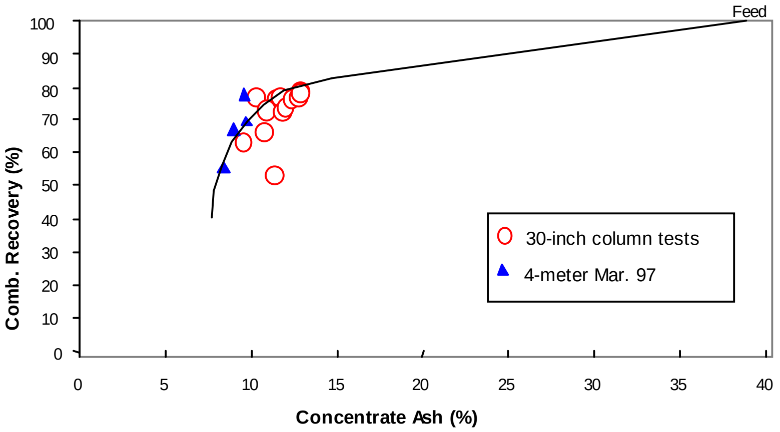


Figure 3-20. Results of 4-meter and pilot-scale column tests.

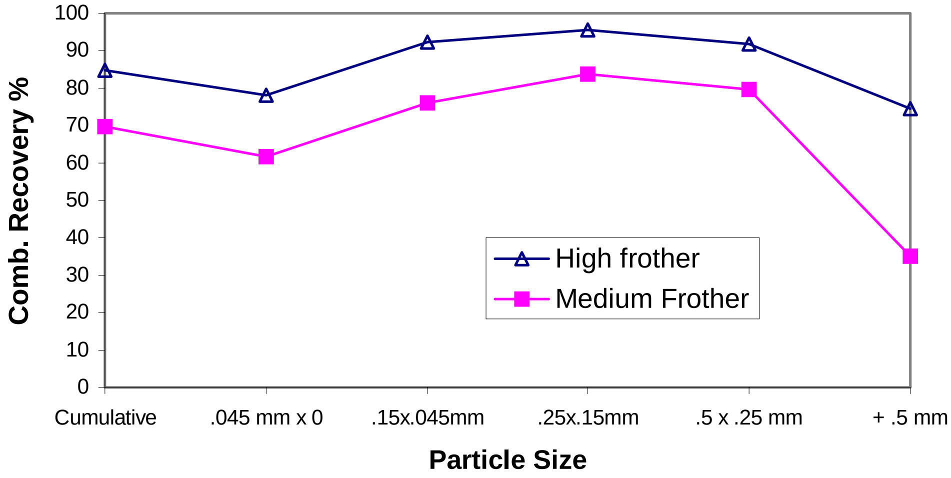


Figure 3-21. Size-by-size recovery in the 4-meter Microcel column.

3.6 Summary and Conclusions

Coarse coal flotation is alive and well. The testwork presented here illustrates well the potential for coarse coal flotation in a properly operated flotation column. Figure 3-12 shows that particle sizes up to 0.25 mm can be floated consistently in a column. It has shown that coarse coal up to 0.5 mm can also perform well in a column, but coal recovery drops off rapidly above this size. Since it is difficult to separate fine particles accurately by size, with little misplaced material, making a nominal 0.25 mm cut and sending the minus 0.25 mm to a flotation column should work well in most coal processing plants. As long as the misplaced +0.5 mm material in the column feed is minimal, a flotation column should provide very good recoveries at a low product ash.

The high flotation feed ash containing much clay was a major indicator that a flotation column would perform much better at the Lady Dunn Plant than conventional flotation cells. In fact, the original mechanical flotation cells produced an average of 14-16% ash clean coal at only a 20% combustibles recovery. Results from testing the 30-inch diameter column indicated that a product of 10-11% ash could be obtained from the 0.25 mm x 0 material at a combustibles recovery of 75%.

The success of this test work was made tangible by the installation of three Microcel^{TM} flotation columns, each four meters in diameter. The 4-meter Microcel^{TM} columns at the Lady Dunn Preparation Plant produced 10-12% clean coal ash from a 35-40% feed ash at combustible recoveries of 70-80% (weight yields of 50-60%). Results matched well with those predicted from the 30-inch pilot-scale test unit and illustrate the successful scale-up that is possible with proper testing and design.

The decision to treat minus 0.25 mm (60 mesh) material by the columns rather than a traditional minus 0.15 mm (100 mesh) feed was validated by the performance data obtained using the full-scale columns. The columns can produce a 6-9% ash product at over 70% combustible recovery for the 0.25 x 0.15 mm material. In comparison, material of the same size (0.25 x 0.15 mm) that makes it to the spiral circuit reports to clean coal at 20-25% ash. The pilot-scale test data also suggests that any of the coarser 0.5 x 0.25 mm material that may be misplaced to the columns would also be recovered at values close to that of the smaller material. At the time of installation these were the largest known flotation columns processing coal anywhere in the world. This project proceeded successfully from an idea, to laboratory tests, to pilot-scale testing, and then to full-scale commercial operation.

3.7 References

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CHAPTER 4 YIELD ADVANTAGES OF IMPROVED DEWATERING

4.1 Introduction

With the advent of column flotation and its use of wash water, any fine coal amenable to flotation can now be cleaned to a reasonably low ash. In many instances however, all of the fine coal cannot be tolerated in the product due to its excessively high moisture content. Many operations consider the extra effort required to perform better dewatering on the fines to be not worth it. If the overall plant performance is reconsidered in terms of the potential yield gain, more effort may be expended on recovering more fine coal and its associated dewatering.

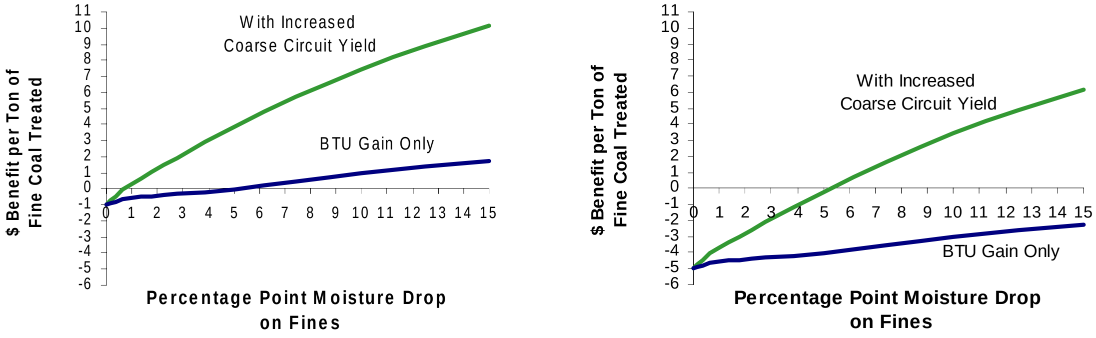
The importance of successful dewatering of fine coal has been recognized since water was first used as a medium for coal cleaning. Although the coal industry has long been interested in methods for removing more surface water from fine coal, it appears that often little additional thought is given to it once the mechanical systems are operating properly. For this reason, many advantages to even a small improvement in dewatering are often overlooked. For metallurgical (coking) coal markets there is little incentive to reduce moisture below that required by the contract since the contracts are on a dry ash basis. With steam or utility coals, however, any excess moisture is a reduction in the thermal value of the coal and thus of some importance to the utility.

Removing some of the water from the coal results in an increase in its thermal (BTU or calorie) value, which usually brings a premium or an increase in sales price. However, the amount of saleable coal is reduced by the amount of moisture reduction. Premiums paid are usually only a little more than the value of the coal tonnage reduction and thus there is little revenue remaining to cover the cost of the additional dewatering.

4.2 Ash for Moisture Simulations

To coal, water is just like rock in that it has no heating value. In fact, if water is removed from coal, it can be replaced with an equivalent amount of pure ash and maintain the same heating value per ton. If combustible material comes with this ash as in the case of adding middlings (middle gravity material) by increasing media gravities, then the original heating value is maintained but the total tons have increased. Improvements in the fine coal circuit often benefit the coarse circuit more than the fine circuit. At Peak Downs Plant in Australia, a drop in flotation product ash from 9.5 to 6% by the addition of new flotation columns allowed the coarse circuit ash to be raised such the total plant saw more than 4 points yield increase (Brake and Eldrige 1996).

The two ways of realizing the benefit of fines moisture reduction on steam coal at a typical plant are shown in Figure 4-1. In Figure 4-1a the cost of additional dewatering is $1 per ton of fines dewatered. If the fines moisture drops by at least 5 percentage points, the cost of additional dewatering is recovered on the basis of premiums for the higher heating value coal. If, on the other hand, the original heating value is maintained by raising the media gravities on the coarse coal circuits, then the value of the additional yield will result in a net benefit after only a 1 point drop in fines moisture (upper line on graph). Any moisture drop on the fines above 1 point then brings considerable revenue.



a) b)

Figure 4-1. Total effect of moisture reduction on fines portion at dewatering cost of: a) $1 per ton of fines treated, b) $5 per ton of fines treated. Base case is $20/ton sales, 12,500 BTU, $0.25/100 BTU premium, 30% fines moisture.

If the cost of dewatering the fines is $5 per ton of fines treated, the slope of the benefit line is the same but a greater moisture drop is required to break-even on the dewatering costs (Figure 4-1b) than was required at $1 per treated ton. If only the BTU premiums are sought, a dewatering cost of $5 per ton can never be recovered. Maintaining a constant BTU and increasing coarse circuit yield will produce a revenue gain for any moisture drop greater than approximately 5 points. For the example used, a 10 point moisture drop in the fines (from 30 to 20%) produces a gain of $3.40 per ton of fines treated, even after the $5 per ton of dewatering cost. For the 1000 tph plant example this is $200 per hour or approximately $1.2 million of additional revenue.

4.3 Base Case and Modifications

The plant example used includes common cleaning devices as shown in the circuit description in Table 4-1. Froth flotation is the source of the high moisture fines but the concepts presented here apply to any high moisture portion that has potential for further dewatering. A common mistake made when evaluating moisture reductions is failing to reduce plant product tonnage by the amount of moisture reduction. By showing the results in table form, one can see the effect of all changes on total plant results.

Assumptions used include a base sales price of $20 per ton for 12,500 BTU currently being produced by the plant (Table 4-1). BTU premiums are at $0.25 per 100 BTU and the initial fines moisture is 30%. The only change for Table 4-2 is a drop in fines moisture from 30 to 25% which results in increased product BTU and a lower product tonnage. Rather than collect the BTU premiums, it is more profitable to increase the separating gravity as in Table 4-3 while maintaining the same BTU as in the base case of Table 4-1. The simulations show that this increased yield produced an additional 15.1 tph for the example in Table 4-3. Only the media gravities on the bath and cyclone circuits are considered truly variable and thus were the only ones changed. To optimize yield for each simulation the gravities were adjusted on the principle of maintaining a constant incremental ash plus moisture (Abbot 1981) on the media circuits.

Table 4-1. Base case (original) plant settings, 30% fines moisture.

Circuit Description	Feed Wt%	Cut SG	Ep	Feed Rate	Clean Ash	Clean Yield	Comb. Rec.	Moist	AR BTU	AR Tons
D M Bath	*36.09*	*1.52*	*0.02*	360.9	7.66	56.65	81.37	*4.00*	13251	212.9
D M Cyclone	*48.57*	*1.49*	*0.04*	485.7	7.88	56.88	77.80	*6.00*	12918	293.9
Coal Spirals	*5.34*	*1.85*	*0.15*	53.4	19.84	64.73	85.27	*12.00*	10224	39.9
Froth Flotation	*10.00*	**	**	100.0	*10.00*	41.09	64.48	*30.00*	9000	58.7
Plant Totals	100.00	**	**	*1000.0*	8.65	55.71	78.41	8.01	12500	604.9

Table 4-2. Plant settings for 25% fines moisture with BTU gain only.

Circuit Description	**Feed Wt%**	**Cut SG**	**Ep**	**Feed Rate**	**Clean Ash**	**Clean Yield**	**Comb. Rec.**	**Moist**	**AR** *BTU*	**AR Tons**
D M Bath	*36.09*	*1.52*	*0.02*	360.9	7.66	56.65	81.37	*4.00*	13251	213.0
D M Cyclone	*48.57*	*1.49*	*0.04*	485.7	7.88	56.88	77.80	*6.00*	12918	293.9
Coal Spirals	*5.34*	*1.85*	*0.15*	53.4	19.84	64.73	85.27	*12.00*	10224	39.3
Froth Flotation	*10.00*	**	**	100.0	*10.00*	41.09	64.48	*25.00*	9750	54.8
Plant Totals	100.00	**	**	*1000.0*	8.65	55.71	78.41	7.42	12590	601.0

Table 4-3. Plant settings for 25% fines moisture with increased coarse circuit yields.

Circuit Description	**Feed Wt%**	**Cut SG**	**Ep**	**Feed Rate**	**Clean Ash**	**Clean Yield**	**Comb. Rec.**	**Moist**	**AR BTU**	**AR Tons**
D M Bath	*36.09*	*1.56*	*0.02*	360.9	8.36	58.27	83.06	*4.00*	13147	219.0
D M Cyclone	*48.57*	*1.53*	*0.04*	485.7	8.72	59.21	80.23	*6.00*	12791	305.9
Coal Spirals	*5.34*	*1.85*	*0.15*	53.4	19.84	64.73	85.27	*12.00*	10224	39.3
Froth Flotation	*10.00*	**	**	100.0	*10.00*	41.09	64.48	*25.00*	9750	54.8
Plant Totals	100.00	**	**	*1000.0*	9.31	57.45	80.28	7.35	12500	619.0

4.4 Conclusions

Any improvements in the ash or moisture of the fine coal circuit can have tremendous benefits in the total plant product. It is often overlooked or forgotten that normal plant operations include compensations made in the coarse circuit due to higher ash or moisture of the fines. The value of the yield increases possible in the coarse coal circuits due to improvements in the fine circuit can often easily justify these improvements even though the apparent benefit to the fines circuit is small.

4.5 References

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CHAPTER 5 QUANTITATIVE EVALUATION OF THE APPLICATION OF COARSE COAL FLOTATION TO PLANT CIRCUITRY

5.1 Introduction

The model discussed in previous sections showed that fundamentally, relatively coarse coal should be floatable under the proper conditions. The pilot-scale testing and later full-scale operations at the Lady Dunn Preparation Plant showed that coarse coal can indeed be floated quite successfully in a column. Not only can it be floated, but it can also be deslimed of high ash fines in the column. The ability to successfully clean fine particle size coal (minus 0.150 mm) previously considered too high in feed ash for treatment, is of a significant benefit by itself (Davis 1993). But what are the benefits of treating coarser particle material (i.e. 0.150 to 0.5 mm) that is currently already being processed by other devices? Can coarse coal flotation benefit plant operations and what are the real advantages?

Processing coal smaller than 1 mm in size has always been more difficult than processing the coarser sizes and remains so today. Even though several devices such as spirals and heavy media cyclones can process the 1 x 0.5 mm fraction with reasonable efficiencies, the size range below 0.5 mm remains a difficult area. Heavy media cyclones can process particles as small as 0.150 mm in size, but those circuits are expensive and difficult to operate. Spirals are often the current choice for material down to 0.150 mm but detailed test data shows that spirals perform very poorly on particles below 0.25 mm in size. Flotation is the only choice for cleaning material finer than 0.150 mm. However, high ash fines are often discarded to refuse with no flotation due to the difficulty in treating with conventional flotation cells.

It is proposed that in all cases where the coal is amenable to flotation (i.e. sufficiently hydrophobic and not oxidized) that column flotation be extended to much coarser sizes than that of current practice. The advantages of coarser particle flotation are two-fold: 1) the combustible recovery or cleaning efficiency for column flotation may be greater than other devices, 2) the ability to make a much coarser size cut as the smallest size cut made in the plant, provides for much less misplacement of high ash fines to clean coal. Anytime that high ash fines can be removed from the product, they can be replaced with middlings particles from the coarser coal circuit. By raising the separating cut-point on the coarser or intermediate sized coal, the yield can be increased and the overall product will be coarser.

Since most of the advantages of improved performance for a given particle size range may actually be realized in another circuit, this work evaluated the use of column flotation in four different plant circuit configurations. Various models for sizing and gravity separation were employed for the predictions of total plant product. The effects of misplaced particle sizes on total performance was also considered. Some comparisons were made to actual plant performance where that information was available.

5.2 Case Circuits and Evaluation Methods

Four different plant configurations were utilized for the evaluations. In addition, the same four case circuits were evaluated on two separate coal seams to determine the influence of coal type on the results. A common circuit today consists of sending minus 1 mm material to a classifying cyclone with the overflow (nominal 0.150 mm x 0) going to either flotation or discard. The underflow of the cyclone is then cleaned in spirals. The base case (case A), therefore utilizes column flotation for the minus 0.150 mm fraction, spirals for 0.150 to 1 mm and heavy media cyclones (HMC) for the plus 1 mm material (Figure 5-1).

Case B is similar to case A except that the top size going to column flotation is 0.25 mm. Case C provides for the column to handle everything minus 0.5 mm in size and the spirals then see a very narrow size range of 0.5 to 1 mm. In case C and D the smallest size cut made in the plant is 0.5 mm, which results in very little misplacement of fines to the oversize portion. In case D the spirals are eliminated and the heavy media cyclones treat everything above 0.5 mm, which was common in the coal industry prior to the utilization of spirals. Column flotation treats all material below 0.5 mm.

Development of the case studies involved the understanding and modeling of several devices. Each device of the A, B, C, and D circuits was evaluated and modeled separately. A discussion of the rationale and modeling of each cleaning or sizing device follows.

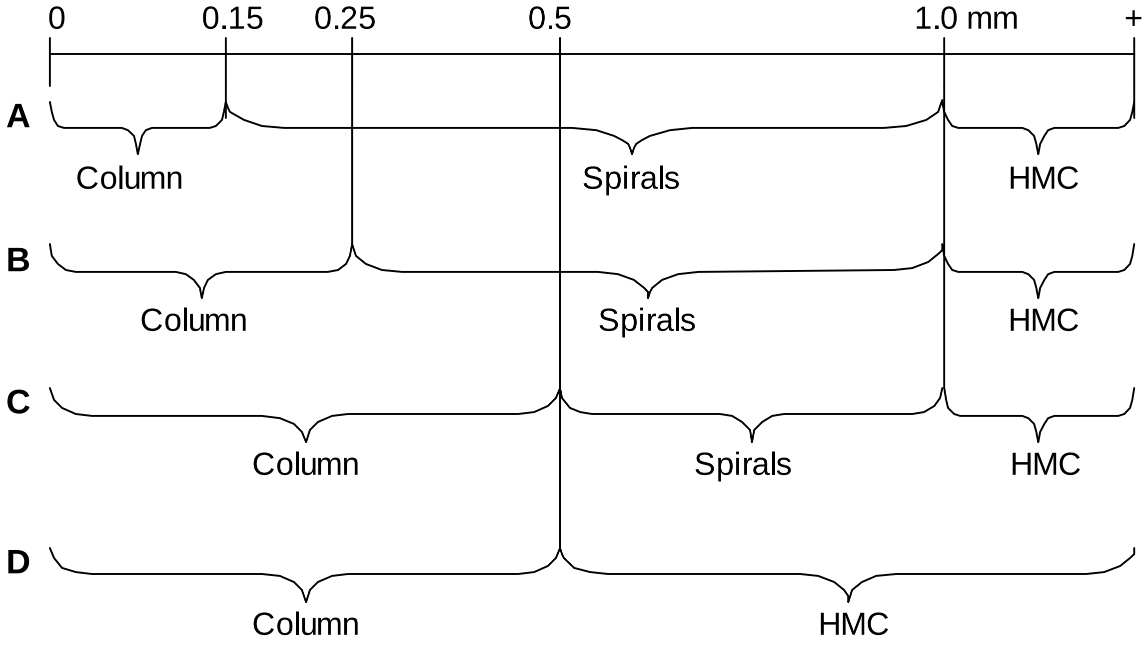

Figure 5-1. Particle size and devices for the case circuits.

Column Flotation

Several authors, test work conducted for the USDOE, and a few plant installations have shown the clear advantage of flotation columns over conventional flotation cells when treating a high ash, minus 0.150 mm feed. For this reason it was taken as a given that if flotation is to be used on the minus 0.150 mm material in these case circuits, that a column is the best choice and is state of the art.

Columns are thought by many in the industry (Fonseca 1995) to be effective only on the ultrafine material and that material coarser than 0.150 mm cannot be recovered well in a column. The author's belief that coarser material can be floated well in a column led to the test work at the Lady Dunn Preparation Plant which along with the eventual full-scale installation there showed that coarser particles can be floated successfully in a column. Detachment is a major concern for floating coarse particles and the fundamental model presented previously shows that it should be possible to float coal up to at least 0.5 mm.

Stockton Seam

The data used to predict column flotation results in the case evaluations came from actual test data of the 30-inch and 4-meter diameter columns as tested on the Stockton seam coal and is presented in detail in Chapter 3. A 12% ash product was a common result and was near the "elbow" of the grade-recovery curves during testing for both the minus 0.150 mm and minus 0.25 mm material. The 12% ash product was thus used for both Case A at minus 0.150 mm feed and Case B with minus 0.25 mm feed.

5-Block seam

The 5-Block seam predictions were based on laboratory column tests of that seam from Central West Virginia. The sample had a top size of 0.5 mm and thus predicted performance on both fine and coarse sizes. The recent work in a 2-inch column provides further support for the control of operating parameters necessary to produce high recoveries on the coarser coal particles. Figure 5-2 illustrates that in the case of a high volatile coal, the proper amount of diesel collector can produce high recoveries on particle sizes up to at least 0.5 mm.

The effect of feed rate is also shown on the two low diesel dosage tests in which a slightly lower feed rate (80 vs. 93 g/min) produced higher recoveries. The small feed rate effect was overcome at the higher diesel dosage however. In all four tests air fraction was held constant by controlling frother to a relatively constant ppm basis.

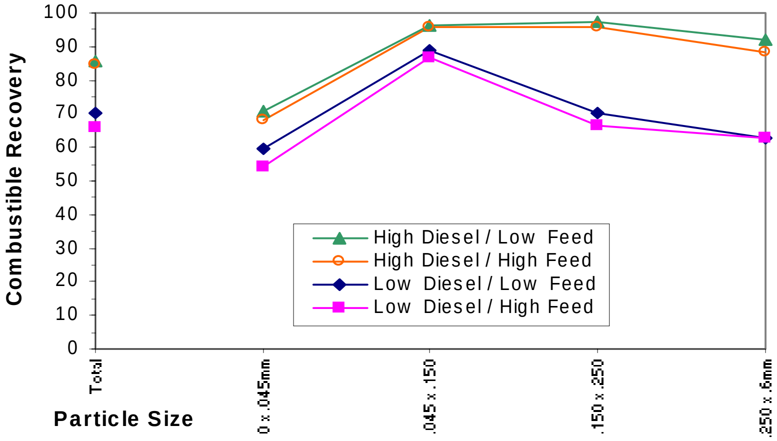

Figure 5-2. Recovery by size from 2-inch column tests on high-vol 5-Block seam coal, feed rate range 80 and 93 g/min, diesel rates 250 & 500 g/Tonne.

Increased recovery through the use of higher diesel or frother dosages or lower feed will bring higher product ashes as expected from a grade-recovery curve. With conventional flotation cells any change made to increase coarse recovery will usually produce a major increase in fine particle ash. However, with a column using wash water in a deep froth, this is not the case. Figure 5-3 shows that a drastic increase in fine particle ash does not happen, but rather the product ash increases a comparable amount for all sizes. If anything, the product ash increases less for the finest sizes than for the coarser. The full size-by-size results from the 5-Block seam tests are given in Table 5-1.

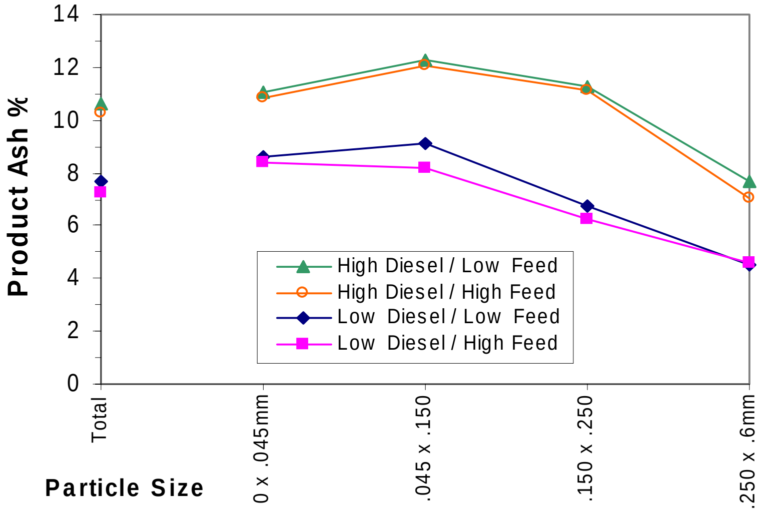

Figure 5-3. **Product ash by size from 2-inch column tests on high-vol 5-Block seam coal, feed rate range 80 and 93 g/min, diesel rates 250 and 500 g/Tonne.**

Table 5-1. **Size-by-size results of 2-inch column flotation tests on 5-Block seam coal.**

BR-5	**FEED**			**CONC**			**TAILS**		
Size	**% Wt.**	**Feed Ash %**	**Cum Flt.Ash**	**% Wt.**	**Conc Ash %**	**Cum Flt.Ash**	**% Wt.**	**Tails Ash %**	**Cum Flt.Ash**
.6 x .250mm	16.32	12.39	36.00	19.31	4.49	7.71	13.97	23.17	62.75
.250 x .150	9.09	23.66	40.61	14.77	6.74	8.49	4.69	46.42	69.18
.150 x .045	23.12	23.07	42.67	34.13	9.10	8.88	11.20	65.10	70.50
.045mm x 0	51.46	51.48	51.48	31.79	8.64	8.64	70.13	71.36	71.36
Total	100.00	36.00		100.00	7.71		100.00	62.75	

BR-5	**Individual by Size**				**Cum. From Bottom**			
Size	**Wt. Yield**	**Comb. Rec.**	**Ash Rej.**	**Sep Eff.**	**Wt. Yield**	**Comb. Rec.**	**Ash Rej.**	**Sep Eff.**
.6 x .250mm	57.71	62.91	79.09	42.00	48.61	70.09	89.58	59.67
.250 x .150	57.37	70.08	83.65	53.74	47.08	72.55	90.16	62.70
.150 x .045	75.05	88.68	70.39	59.08	45.16	71.78	90.60	62.38
.045mm x 0	31.70	59.68	94.68	54.36	31.70	59.68	94.68	54.36
Total	48.61	70.09	89.58	59.67				

Column Case Values

The potential advantages of Case B over Case A are 1) size classification is better at a 0.25 mm cut than at 0.150 mm and should result in less misplaced high ash fines to the spiral circuit and 2) the column may perform better than spirals on material smaller than 0.25 mm.

Case C and D look at utilizing a column on material up to 0.5 mm. The major advantages of these circuits are 1) much better sizing since a 0.5 mm cut can be made very economically on conventional vibrating screens and 2) a column can make the equivalent of a much lower gravity cut than can spirals. Expected column performance values for the 0.25 x 0.5 mm material came from the same test data as for the smaller sizes. There was no split feeding of sizes to the columns. All of the raw minus 0.5 mm material was fed to the column together in all tests and the products were then screened to determine size-by-size results. For the Stockton seam a value of 8% ash and 60% yield was used for this coarse material, while an 8% ash and 61% yield value was used for the 5-Block seam. The yields for the coarser material are higher than the finer (-0.150 mm) material due to the lower feed ash on the 0.25 x 0.5 mm material. Flotation yields for the coarse material vary considerably depending on the operating parameters used. The values chosen here are those easily achievable if sufficient froth capacity and collector is used. As discussed elsewhere, the coarse is always the most difficult to float and thus results in a lower product ash than that of the finer material.

Sizing Devices (Classifying Cyclones and Sieve Bends)

Circuits A and B require a fine size cut at either 0.150 mm or 0.25 mm, while all circuits require a coarser cut at either 0.5 or 1 mm. The coarser cut of 0.5 and 1 mm is common to all circuit configurations and can be performed in actual plant practice at relatively high efficiencies with little misplacement of fines. For these reasons, no misplacement of sizes is considered for this coarse size cut. The inefficiencies of the finer size cuts (0.150 and 0.25 mm) are the cause of many, often unacknowledged, problems. Misplaced fine slimes in circuits such as spirals, usually report to the clean coal stream and if not removed downstream, contribute significantly to a higher product ash. The two devices nearly always used for making the fines size cuts are classifying (hydro) cyclones or fine wire sieve bends.

A partition curve best describes the performance of a separating device. The partition curve displays the probability of a particle reporting to either the oversize or undersize flowstream. As used herein, the partition value or coefficient, Pc, will refer to the probability of the given size reporting to the oversize of a sieve bend or the underflow of a classifying cyclone. As shown in Figure 5-4 the probability or partition coefficient is plotted on the Y-axis while the particle size is plotted on the X-axis. A common partition curve is 'S' shaped and is generally characterized by several parameters:

- The d50 is the particle size which has a 50% chance of reporting to either the oversize or undersize portion of the separating device and is found at the vertical middle of the plot.
- The sharpness of separation is determined by the slope of the line on both sides of the d50 and is represented by a slope factor, α.
- Some water always reports to the oversize flowstream. Very little if any separation is performed on the ultrafine material finer than about 0.045 mm (325 mesh) and thus the ultrafines travel with the water. A factor, R_f , represents the percentage of the ultrafine material that reports to the oversize stream. This is illustrated on the partition curve by the lower tail which levels out at some partition coefficient greater than zero (R_f). Water that reports to the oversize stream is often referred to as 'by-pass'.

To correct for the fines that bypasses to the coarse stream, Eq. 5-1 is used to calculate the partition coefficients Pc for the *corrected efficiency curve*. By removing the effect of the bypass (R_f), one can better see the fundamental classification behavior of a device. The bypass amount, however, is real and varies for different geometrical designs and circumstances and thus must be accounted for at some point in calculations and comparisons.

$$Pc_{ci} = 100\frac{P_i - R_f}{1 - R_f} \qquad \text{Eq. 5-1}$$

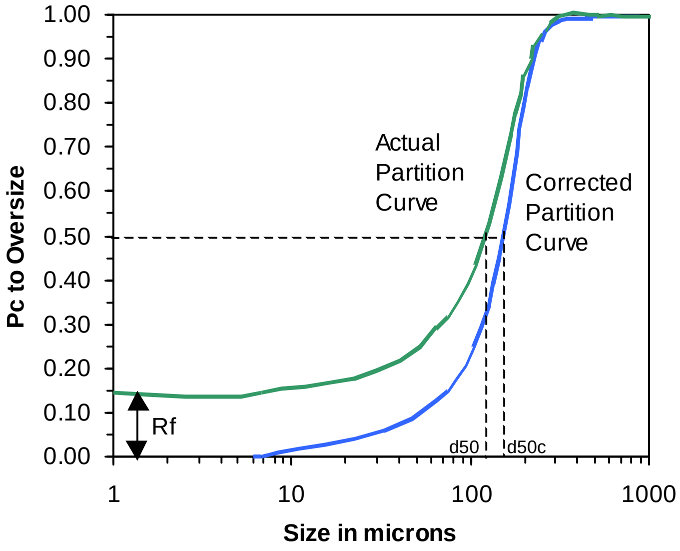

Figure 5-4. Characteristic partition curve for classification.

One of the better models developed in recent years for defining fine particle classification is the Lynch Rao model (Lynch and Rao 1975) of Eq. 5-2, where α is the slope factor, d is the particle diameter, and d50c is the corrected d50.

$$Pc_c = \frac{\left[\exp\left(\alpha \frac{d}{d_{50c}}\right) - 1\right]}{\left[\exp\left(\alpha \frac{d}{d_{50c}}\right) + \exp(\alpha) - 2\right]} \qquad \text{Eq. 5-2}$$

By combining Eq. 5-1 and Eq. 5-2, the true partition efficiency curve can be calculated as in Eq. 5-3 (Heiskanen 1993). If actual data is collected on a classifying device this equation can be fitted to the data by variations in the slope factor and the d50c.

$$Pc = (1 - R_f)\left[\frac{\exp\left(\alpha \dfrac{d}{d_{50c}}\right) - 1}{\exp\left(\alpha \dfrac{d}{d_{50c}}\right) + \exp(\alpha) - 2}\right] + R_f \qquad \text{Eq. 5-3}$$

An α value of 2.5 to 4 is common for a classifying cyclone making a cut near 0.150mm while a value of 4 to 6 is common for fine wire sieve bends (Firth et al. 1995) (Moorehead 1998). The lower values represent a flatter slope of the partition curve and thus more misplacement and poorer performance. For simulation of the 0.150 mm size cut in the case A circuit, a 15-inch diameter classifying cyclone was utilized with an α value of 2.7. This value is given by Moorehead as the most common for a 0.150 mm cut in cyclones. A value of 4.0 was chosen for the sieve bends making the 0.25 mm cut in the case B circuit. The value of 4.0 was taken from actual plant data collected by Moorehead, and although it is at the low end of the 4 to 6 range given previously, it provides for the deterioration in performance that takes place as sieve screens wear.

Many plants that have considerable amounts of high ash material in the ultrafine size have installed additional classifying devices on the clean coal stream of the spirals. This combined with the classifiers on the feed to spirals provide what is essentially a two stage classifying circuit. Although this comes at additional capital and operating costs there may be considerable benefits in the further removal of fines from the coarser streams. In order to evaluate the benefit of just adding more classification circuits, two-stage cycloning was considered for Case A (labeled Case A-Improved) and two-stage sieving for Case B (labeled Case B- Improved). Misplaced fines across a sieve will have an ash value similar to that of that size in the plant feed. On the other hand, classifying cyclone underflow fines may actually have a slightly higher ash due to the higher density of the higher ash particles. For evaluation purposes, the misplaced fine material kept the ash value of that size in the feed.

Gravity Separations – Spirals and Heavy Media Cyclone (HMC)

Spiral and heavy media cyclone performance was predicted from the use of partition curve modeling. A computer program jointly developed by Eric Yan and the author was utilized to quickly predict ash and yield results from the various size feed washabilities. In order to have sufficient and consistent detail in the feed washability, the program expands the washability to 360 gravity points by the use a modified clamped spline fit. The program then has three models available for determining the actual partition coefficients. Partition curve modeling for gravity separations is identical to that of size classification except that the particle density parameter replaces that of the size parameter on the X-axis. A portion of the gravity partition curve between the partition coefficients (Pc) 25% and 75% is commonly used to determine the sharpness of separation. A slope index, Ecart Probable Moyen (Ep), is defined as the difference of the S.G. of the 75% Pc and the S.G. of the 25% Pc divided by 2 (Eq. 5-4).

$$Ep = \frac{d75 - d25}{2} \qquad \text{Eq. 5-4}$$

A lower value means a steeper slope and a sharper separation.

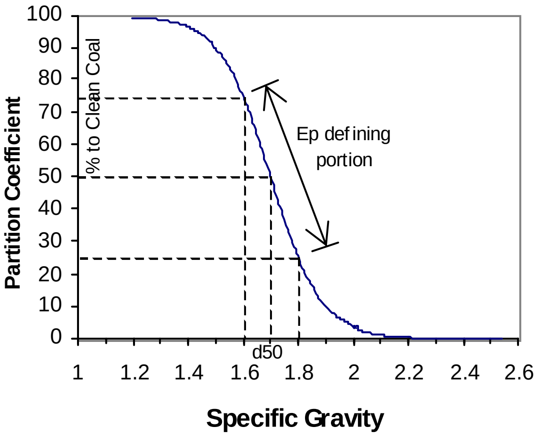

Figure 5-5. Characteristic partition curve for gravity separation.

Spirals

The spirals were modeled using an Ep of 0.15 and a cut-point of 1.95 S.G. for the material coarser than 0.25 mm. This is considered good performance for the overall average of a large bank of spirals. For the processing of the 0.5 by 1 mm size range as in Case C and D, the same Ep of 0.15 and a lower gravity cut-point of 1.90 S.G. was chosen due to the narrow size range which is optimum for a spiral. As recommended by David Chedgy of CLI Corp., an Ep of 0.30 and a gravity cut-point of 2.10 S.G. was chosen for the 0.25 x 0.15 mm material since this size is known to perform poorly in a spiral. Even with this poor performance the spirals produced 15.54% ash product on Stockton coal, which is better than the actual 22% ash reported on this size fraction at the Lady Dunn plant.

Heavy Media Cyclone (HMC)

Predictions for the heavy media cyclone performance assume that all particle sizes of interest are combined into only one size of cyclone. Although changes in operating parameters will affect cyclone performance the same Ep was used for all cases and thus HMC performance was held constant. An Ep of 0.06 was chosen as a typical value for all modeling of the plus 1 mm material in heavy media cyclones. The cut-point was then varied on the HMC to produce a constant total product ash for all case circuits. For the 0.5 x 1 mm to HMC in Case D, an Ep of 0.09 was chosen due to the smaller particle size. This is conservative compared to the 0.075 Ep predicted in Eq. 5-5 where ρ_{50c} is 1.60 S.G., the cut-point for the coarser particles, and *d*, the mean particle size, is 0.71 mm. Eq. 5-5 is from a HMC handbook developed by Chris Wood (1990a) at the JKMRC center in Australia.

$$Ep_d = \frac{0.0333\rho_{50c}}{d} \qquad \text{Eq. 5-5}$$

The Ep increases rapidly below a particle size of 1mm and a table of size-by-size Eps for full scale installations found in Wood (1990b), indicates that both the 0.06 Ep for the coarser sizes and the 0.09 Ep for the 0.5 x 1 mm is typical or somewhat conservative.

Yield Optimization

It has been shown by Abbott (1981) that for yield to be maximized in a multi-circuit plant, all units should be operated at the same incremental ash value. For a perfect separating device, similar to laboratory float/sink, the incremental ash is defined as the ash of the next lump to float if the separating cut-point is raised by a very small amount. However, with a non-perfect separator, a small increase in separating gravity not only brings the next near-gravity particle but also brings material that is several gravity points away from the cut-point. The higher the Ep, the more this occurs. For actual plant separating devices the incremental ash can be obtained by samples at two slightly different cut-point gravities or by simulation. The effective incremental ash is then calculated by dividing the difference in the two yield ash products by the difference in yields (Eq. 5-6).

$$IncAsh_{Eff} = \frac{(yield_2)(ash_2) - (yield_1)(ash_1)}{yield_2 - yield_1} \qquad \text{Eq. 5-6}$$

The incremental ash concept as defined above applies to any plant in which dry ash is the controlling parameter. However, for a plant that is limited by the BTU value of the product, incremental inerts must be held constant. The term incremental inerts is defined as the incremental ash plus the incremental moisture. For different particle sizes the moisture will obviously vary considerably and results in the need for the finer sized coal to have a much lower incremental ash to allow for the higher moisture values.

For the evaluations undertaken in this work, the heavy media cyclone was the only unit that had significant control of the separating gravity. For this reason, as well as to maintain the ability for direct comparisons between cases, the constant incremental ash concept could not be followed in Cases A through D. If, however, the plus 1 mm material were separated in a heavy media bath for the coarser portions and in a HMC for the remaining material, the incremental ash concept could then be applied between those two devices.

In this evaluation the conditions of Case A were taken as the base case and represent an existing operation that is just meeting a target ash spec. The additional substitutions of column or spirals were set up with the most likely performance results. It was then left to the heavy media cyclone circuit to be adjusted to bring the total plant ash (always dry basis ash) back to the same product spec as in the base case. Just as in most operating plants, the resulting yield was just that – a result and not an optimization.

5.3 Results of Case Evaluations

After numerous simulation runs the results were compiled into Table 5-2 and Table 5-3 which show the tons, yield, and ash results by size and circuit for each case. A target ash of 13% was used for the Stockton seam which compares with that of an existing plant processing this seam. For the 5-Block seam a target ash of 12% was used, as that is a typical ash for this seam. The evaluation results are discussed case by case.

Case A

One of the most significant differences was due to improving the classification at 0.150 mm (100 mesh). By using a two-stage classifying cyclone circuit the amount of misplaced raw fines in the clean coal product dropped from 19.2 tons per hour (tph) to 5.9 tph for the Stockton and a similar amount for the 5-Block seam. Reducing the amount of this high ash material in the product caused a drastic drop in the total product ash which allowed the yield to increase considerably on the HMC circuit from Case A to Case A-Improved. This points out the importance of good classification especially when processing coal with high ash fines. In practice, many plants that have a high ash feed are starting to realize the importance of effective desliming and have put additional sieves and or cyclones on the spiral product stream to remove more of the high ash slimes from the product.

Case B

Case B illustrates the advantages of making a coarser size cut for sizing the feed to flotation. By making a 0.25 mm cut rather than 0.150 mm the amount of misplaced fine material dropped from 19.2 to 9.2 tons for a one-stage sieve (Case B) and from 5.9 to 1.4 tons for the two-stage sieve (Case B-Improved). Due to similar feed sizes and ashes the results were similar for the Stockton and 5-Block seams. Even with two-stage classification, simply sending material up to 0.25 mm to flotation rather than only 0.150 mm results in an overall yield increase of 0.85% for the Stockton and 1.0% for the 5-Block seam.

Case C

Allowing the flotation column to treat the 0.25 x 0.5 mm material rather than spirals caused a decrease in the yield on this size fraction but also dropped the ash. Apparently the incremental ash was lower on the HMC circuit and thus the lower flotation ash allowed the HMC cut-point to be raised such that the HMC yield increased by roughly 1.7 percentage points on both seams. Since the HMC represents over 80% of the feed, the total yield for both seams increased by approximately 0.5% on Case C over Case B-Improved.

Case D

Removing the 0.5 x 1.0 mm material from spirals and putting it into the HMC circuit provided an overall yield increase of ~0.25 percentage points for both seams. Although relatively small compared to the increases for the other Cases, this is still 2.5 tph of additional clean coal. The main cons of this circuit is the increased difficulty of rinsing the magnetite from this finer size material. The 0.5 x 1 mm material is being treated by HMC quite successfully in many plants but it does require more screens and an increased media recovery circuit. The major advantages to Case D circuit is (i) the ability to separate at a lower gravity cut-point (ii) having only two cleaning circuits in the plant and (iii) no fine slimes problem since the smallest size cut made is 0.5 mm.

Table 5-2. Case evaluation results, Stockton seam.

Stockton Seam							
Case A	**Size**	**Feed Tons**	**Clean Tons**	**Ep**	**d50**	**Yield %**	**Ash %**
Column	-0.150 mm	59.5	30.9			52.00	12.00
Spirals	0.150 to .25mm	16.3	12.7	0.30	2.10	77.86	15.54
Spirals	0.250 to 1 mm	88.8	67.3	0.15	1.95	75.82	13.90
Misplaced	-0.150 mm	19.2	19.2			100.00	42.00
HMC	+1 mm	816.2	468.4	0.06	1.536	57.39	11.68
	Total	1000.0	598.6			59.86	13.00
A-Improved (Less Misplacement)		**Feed Tons**	**Clean Tons**	**Ep**	**d50**	**Yield %**	**Ash %**
Column	-0.150 mm	72.8	37.9			52.00	12.00
Spirals	0.150 to .25 mm	16.3	12.7	0.30	2.10	77.86	15.54
Spirals	0.250 to 1 mm	88.8	67.3	0.15	1.95	75.82	13.90
Misplaced	-0.150 mm	5.9	5.9			100.00	42.00
HMC	+1 mm	816.2	498.5	0.06	1.574	61.08	12.55
	Total	1000.0	622.3			62.23	13.00
Case B	**Size**	**Feed Tons**	**Clean Tons**	**Ep**	**d50**	**Yield %**	**Ash %**
Column	-0.250 mm	85.8	46.6			54.32	12.00
Spirals	0.250 to 1 mm	88.8	67.3	0.15	1.95	75.82	13.90
Misplaced	-0.150 mm	9.2	9.2			100.00	42.00
HMC	+1 mm	816.2	495.4	0.06	1.572	60.70	12.43
	Total	1000	618.6			61.86	13.00
B-Improved (Less Misplacement)		**Feed Tons**	**Clean Tons**	**Ep**	**d50**	**Yield %**	**Ash %**
Column	-0.250 mm	93.6	50.8			54.32	12.00
Spirals	0.250 to 1 mm	88.8	67.3	0.15	1.95	75.82	13.90
Misplaced	-0.150 mm	1.4	1.4			100.00	42.00
HMC	+1 mm	816.2	511.2	0.06	1.597	62.63	12.90
	Total	1000	630.8			63.08	13.00
Case C	**Size**	**Feed Tons**	**Clean Tons**	**Ep**	**d50**	**Yield %**	**Ash %**
Column	-0.25 mm	95	51.6			54.32	12.00
Column	0.25 x 0.5 mm	43.6	26.2			60.00	8.00
Spirals	0.5 to 1 mm	45.2	33.7	0.15	1.90	74.44	13.51
Misplaced	-0.5 mm	0	0.0			0.00	0.00
HMC	+1 mm	816.2	524.8	0.06	1.623	64.30	13.32
	Total	1000	636.2			63.62	13.00
Case D	**Size**	**Feed Tons**	**Clean Tons**	**Ep**	**d50**	**Yield %**	**Ash %**
Column	-0.25 mm	95	51.6			54.32	12.00
Column	0.25 x 0.5 mm	43.6	26.2			60.00	8.00
Misplaced	-0.5 mm	0	0.0			0.00	0.00
HMC	0.5 x 1 mm	45.2	32.2	0.09	1.75	71.26	11.12
HMC	+1 mm	816.2	529.1	0.06	1.632	64.82	13.46
	Total	1000	639.0			63.90	13.00

Table 5-3. Case evaluation results, 5-Block seam.

5-Block Seam							
Case A	**Size**	**Feed Tons**	**Clean Tons**	**Ep**	**d50**	**Yield %**	**Ash %**
Column	-0.150 mm	64.9	34.0			52.40	11.47
Spirals	0.150 to .25 mm	21.1	16.0	0.30	2.100	75.75	19.69
Spirals	0.250 to 1 mm	94	71.0	0.15	1.950	75.55	15.43
Misplaced	-0.150 mm	19.1	19.1			100.00	41.70
HMC	+1 mm	800.9	427.8	0.06	1.549	53.42	9.86
	Total	1000	567.9			56.79	12.00
A-Improved (Less Misplacement)		**Feed Tons**	**Clean Tons**	**Ep**	**d50**	**Yield %**	**Ash %**
Column	-0.150 mm	78.6	41.2			52.40	11.47
Spirals	0.150 to .25 mm	21.1	16.0	0.30	2.100	75.75	19.69
Spirals	0.250 to 1 mm	94	71.0	0.15	1.950	75.55	15.43
Misplaced	-0.150 mm	5.4	5.4			100.00	41.70
HMC	+1 mm	800.9	451.4	0.06	1.586	56.36	10.88
	Total	1000	585.0			58.50	12.00
Case B	**Size**	**Feed Tons**	**Clean Tons**	**Ep**	**d50**	**Yield %**	**Ash %**
Column	-0.250 mm	96	54.1			56.40	11.41
Spirals	0.250 to 1 mm	94	71.0	0.15	1.950	75.55	15.43
Misplaced	-0.150 mm	9.1	9.1			100.00	41.70
HMC	+1 mm	800.9	453.4	0.06	1.588	56.61	10.94
	Total	1000	587.7			58.77	12.00
B-Improved (Less Misplacement)		**Feed Tons**	**Clean Tons**	**Ep**	**d50**	**Yield %**	**Ash %**
Column	-0.250 mm	103.8	58.5			56.40	11.41
Spirals	0.250 to 1 mm	94	71.0	0.15	1.950	75.55	15.43
Misplaced	-0.150 mm	1.3	1.3			100.00	41.70
HMC	+1 mm	800.9	464.1	0.06	1.607	57.95	11.47
	Total	1000	595.0			59.50	12.00
Case C	**Size**	**Feed Tons**	**Clean Tons**	**Ep**	**d50**	**Yield %**	**Ash %**
Column	-0.25 mm	105.1	59.3			56.40	11.41
Column	0.25 x 0.5 mm	51.3	31.3			61.00	8.00
Spirals	0.5 to 1 mm	42.7	30.9	0.15	1.900	72.32	14.85
Misplaced	-0.5 mm	0	0.0			0.00	0.00
HMC	+1 mm	800.9	478.1	0.06	1.631	59.70	12.15
	Total	1000	599.6			59.96	12.00
Case D	**Size**	**Feed Tons**	**Clean Tons**	**Ep**	**d50**	**Yield %**	**Ash %**
Column	-0.25 mm	105.1	59.3			56.40	11.41
Column	0.25 x 0.5 mm	51.3	31.3			61.00	8.00
Misplaced	-0.5 mm	0	0.0			0.00	0.00
HMC	0.5 x 1 mm	42.7	28.6	0.09	1.750	66.96	11.16
HMC	+1 mm	800.9	482.8	0.06	1.639	60.28	12.38
	Total	1000	601.9			60.19	12.00

General

The gain from changing from the base case (A) to each of the case circuits is shown in Table 5-4 and Table 5-5. Since the first step taken by a plant with considerable high ash fines should be to improve the size classification methods for 0.150 mm desliming, the gains of using coarser size column flotation is also compared to the Case A-Improved. The results of the comparisons are very similar for both seams and thus to reduce confusion only the Stockton seam will be discussed here.

Compared to the gain over the base case, the yield improved with each case up to a yield gain of 4% for the Case D circuit (Stockton seam). The exception to this is the 0.37 point drop in yield from Case A-Improved to Case B. This shows that a good classification (two-stage cyclones) at 0.150 mm is better than a poor classification at 0.25 mm. The additional revenue for the additional tons produced by each circuit change was calculated assuming a $20 per ton net realization and 6000 hours per year of actual production. Compared to the poor classification of Case A the circuit changes could produce up to $4.8 million in additional revenue.

Since most plants with serious fine desliming problems would first make an effort to make a better cut at 0.150 mm, the Case A-Improved is probably the best case to use as a basis for comparison. As shown in Table 5-4 this change alone is worth a projected $2.8 million. Compared to Case A-Improved sending a well sized 0.25 mm topsize to columns (Case B-Improved) rather than a 0.150 mm topsize results in a yield gain of 0.85 points or over $1 million. Sending up to 0.5 mm to columns rather than spirals creates a total of $1.67 million additional. Although sending the 0.5 x 1 mm material to HMC (Case D) will cause some additional capital costs and additional magnetite consumption, there is $336,000 per year available to support it plus the capital savings of not building the spiral circuit.

Table 5-4. Case evaluation summary, Stockton seam.

Case	**Clean Tons Per Hour**	**Yield %**	**Gain Over Case A**			**Gain Over Case A-Improved (Less misplacement)**		
			Add'l Tons	**Add'l Yield**	**Additional Annual Revenue @ $20/ton (6000 hrs)**	**Add'l Tons**	**Add'l Yield**	**Additional Annual Revenue @ $20/ton (6000 hrs)**
A	598.6	59.86	-	-	-	-	-	-
A-Improved	622.3	62.23	23.7	2.37 %	$2,844,000	-	-	-
B	618.6	61.86	20.0	2.00 %	$2,400,000	-3.7	-0.37%	-$444,000
B-Improved	630.8	63.08	32.2	3.22 %	$3,864,000	8.5	0.85 %	$1,020,000
C	636.2	63.62	37.6	3.76 %	$4,512,000	13.9	1.39 %	$1,668,000
D	639.0	63.90	40.4	4.06 %	$4,848,000	16.7	1.67 %	$2,004,000

Table 5-5. Case evaluation summary, 5-Block seam.

Case	Clean Tons Per Hour	Yield %	Gain Over Case A			Gain Over Case A-Improved (Less misplacement)		
			Add'l Tons	Add'l Yield	Additional Annual Revenue @ $20/ton (6000 hrs)	Add'l Tons	Add'l Yield	Additional Annual Revenue @ $20/ton (6000 hrs)
A	567.9	56.79	-	-	-	-	-	-
A-Improved	585.0	58.50	17.1	1.71 %	$2,052,000	-	-	-
B	587.7	58.77	19.8	1.98 %	$2,376,000	2.7	0.27 %	$324,000
B-Improved	595.0	59.50	27.1	2.71 %	$3,252,000	10.0	1.00 %	$1,200,000
C	599.6	59.96	31.7	3.17 %	$3,804,000	14.6	1.46 %	$1,752,000
D	601.9	60.19	34.0	3.40 %	$4,080,000	16.9	1.69 %	$2,028,000

Since the cost of a basic flotation circuit is already in the base case (Case A), the cost of adding coarser material to the flotation circuit would be relatively small. The coarser sizes have a very high carrying capacity in columns and thus only a few additional columns would be needed. The dewatering capacity for the additional floated coarse material is already there in the spiral circuit product stream.

5.4 Conclusions

The simulations and the resulting evaluation summary has shown that nearly each level of circuit change can make a significant improvement in coal yield. This additional coal comes with no additional cost for the coal since it was already mined. Any revenue realized from the sale of this additional coal is pure profit. Although construction cost estimates are not given here, the author's experience indicates that the payback should be less than 1-2 years for any of the case circuits.

The simulation results presented here represent a realistic and somewhat conservative approach to show the advantages of making changes in the traditional plant flowsheet. The two major advantages are:

1. a coarser size cut for separating the fines from the next coarser size cleaning device provides additional yield by eliminating most of the misplaced high ash fines and

2. the use of a flotation column on coarser sizes than traditionally thought (+0.150 mm) allows the column to perform the desliming and ash rejection while simplifying the circuit.

5.5 References

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CHAPTER 6 CONCLUSIONS

The work presented here has significantly increased the understanding of coal processing. The major findings of the present investigations and their potential contributions to obtaining optimum processing of 1 mm x 0 coal are:

1. With proper attention to operating parameters, coarse coal up to at least 0.5 mm in size can be processed successfully in a column flotation cell. A column can float coarse coal as well or better than conventional flotation cells due mainly to the use of wash water in a column. This is especially true at the higher feed ashes due to the need to float coarse coal with high air and frother rates which, in conventional cells, results in excessive entrainment of high ash fines.

2. Detailed pilot-scale work showed that successful coarse coal recovery in a flotation column is sensitive to several parameters:

    - The foremost factor is the loading of floatable material. The bubble surface area only builds one layer of particles and then the froth is considered overloaded and any additional coal is dropped back into the pulp zone. Any truly non-floatable material has no effect on the froth loading or column capacity.
    - Factors that affect the amount of bubble surface area such as air rate and frother dosage. Smaller bubbles have more surface area and also produce a wetter froth which is less likely to collapse from the coarse particles.
    - Proper dosage of collector (diesel) can increase coarse particle recovery but at a slight increase in fine particle ash. The addition of diesel for increased recovery is not new information but when combined with the use of wash water in a column, one can minimize what is normally a considerable increase in product ash.

3. A flotation column properly utilizing wash water is the best desliming system for removing high ash clays. The size-by-size analyses of the pilot- and full-scale test results show good recovery of the minus 0.045 mm (325 mesh) particles while also indicating a low ash comparable to that of the slightly coarser sizes in the froth concentrate. Together these two results indicate excellent removal of only the high ash clays, and not fine coal, from the froth concentrate.

4. The top size for coal flotation in any system should be at least 0.25 mm (60 mesh) rather than 0.150 mm(100 mesh). The size-by-size recovery plots from the pilot-scale and full-scale testwork (Figure 3-12 and Figure 3-21) show that the 0.150 x 0.25 mm fraction generally floats as well or better than the minus 0.150 mm material. A major benefit of making a coarser cut is less misplaced high ash slimes to affect the other circuits.

5. Through the work of Chapter 2 it was concluded that the contact angle, particle size, and particle shape are the major determinants for particle detachment. Most models are for spherical particles but in actual plant practice nearly all particles > 0.1 mm are not at all spherical. This is especially true for coal which has particles that are either cubical or rectangular overall and have edges varying from sharp to rounded. The simplified model developed in this work shows that particle shape has more affect on coarse particle detachment than was previously thought. Contact angle is also important and can be altered to a limited extent. The force required for detachment varies in direct proportion to the perimeter of the flat projection of the particle but varies only by the sine of the contact angle. The good news is that flats remain attached better than cubes or spheres and most coal particles in the 0.25 to 1 mm range are relatively flat with considerable perimeter length.

6. The original concept of this work was to improve fine coal processing by making a coarser top-size cut for the feed to column flotation. This undertaking was fortunate in that it was able to proceed from the concept through lab testing to pilot-scale and then to full-scale commercial operation. Scale-up to a large flotation column has proven successful.

7. Considerable economic benefits can be realized by increasing the size at which the finest size cut in a plant is made. From the case examples it was shown that an increase of almost 1 percentage point in yield ($1 million additional revenue on a 1000 tph plant) can be realized by increasing the top size of flotation feed from 0.150 to 0.25 mm and thereby reducing the amount of high ash fines misplaced to clean coal. If the flotation feed upper size cut were increased from 0.150 to 0.5 mm, misplacement of high ash fines to clean coal would be nearly eliminated and an increase of up to 1.7 percentage points in yield ($2 million) could be realized.

   It is acknowledged that the ultrafine coal recovered when using flotation for all the fines can result in a high product moisture and therefore many plants prefer to discard the ultrafine material rather than process it. However, due to the difficulties in making an accurate ultrafine size cut, fine ash reports to the plant product and relatively coarse low ash coal is discarded unnecessarily. It is the author's belief that it is better to process the fines in column flotation and then set up a screen bowl centrifuge to classify out the amount of ultrafines necessary to meet the moisture specification. A screen bowl centrifuge can classify ultrafines better than a cyclone due to the high G-force as well as there being only particles of similar density. Although not adjustable on-line, the centrifuge overflow ports are relatively easy to adjust and thus recovery can be optimized frequently. It is site-specific but the increased cost of more flotation should be offset by the yield gain from reduced misplacement and reduced classifying costs.

CHAPTER 7 RECOMMENDATIONS FOR FUTURE RESEARCH

On the basis of the present study, further research in the following areas is recommended:

1. The coarse coal flotation undertaken in this study concentrated on floating the material up to 0.5 mm but some results showed potential up to 1 mm. It is suggested that the control parameters highlighted in this work be utilized to explore column flotation up to at least 1 or even 2 mm. Although floating material this coarse will, at best, be much more delicate, there could be many things learned that could be applied to full-scale flotation of smaller sizes such as minus 0.5 mm.

2. It is known that there is a relationship between diesel and frother in flotation and it was noticed in this study that excess diesel reduced the air fraction in the column due to its affect on frother and thus bubble size. An in-depth in-plant study of the relationship and thresholds of the diesel frother relationship and the resulting effect on coarse coal recovery would be very helpful to operators planning to float coarser coal sizes.

3. The ability to more accurately predict carrying capacity and coarse size recoveries from 2-inch column tests would be helpful. The opportunities to compare 2-inch and full-scale columns are improved as more full-scale columns are installed to process coarser coal.

4. The release analysis technique of flotation prediction is much less time, material, and labor intensive than continuous column testing. Is it possible to develop a similar technique that also looks at flotation by size and thus allows prediction of full-scale column carrying capacity?

5. An improved understanding of the classifying capabilities of screen bowl centrifuges would be beneficial to those who want the product moisture reduction by discarding ultrafines through the centrifuge and also to those wanting maximum solids recovery. It would also be helpful to study the frothing capabilities of the centrifuge main effluent in terms of residual frother and solids.

6. Last but not least is the need for affordable means of breaking the strong downstream froths created due to the use of the stronger frothers in columns. The ultrafine coal that is in the main effluent of the fine coal centrifuges only adds to the stability of the froth due to its fine particle size.

APPENDIX A TABULATED PLANT TEST DATA

Test Series 101-110 (Cyclone Overflow)
Test Results

Test #	Ash %			% Solids			% Wt. Yield	Comb. Rec.%	Ash Rej.%	Comb. Sep Eff. %
	Feed	Conc.	Tails	Feed	Conc.	Tails				
101	41.72	8.79	78.64	4.10	18.00	1.92	52.86	82.72	88.86	71.59
102	40.28	7.67	63.59	2.73	21.48	2.18	41.68	64.45	92.06	56.51
103	42.45	5.58	54.22	4.54	18.01	4.84	24.20	39.70	96.82	36.52
104	41.10	6.08	64.65	4.83	17.29	4.66	40.21	64.11	94.05	58.17
105	38.76	5.37	55.49	4.59	20.76	3.62	33.38	51.58	95.38	46.96
106	36.22	6.79	63.13	5.13	13.79	2.41	47.76	69.80	91.05	60.85
107	35.82	6.72	59.07	4.95	11.54	2.31	44.41	64.55	91.67	56.22
108	35.15	5.44	46.00	3.44	10.70	2.93	26.75	39.01	95.86	34.87
109	No Samples Analyzed									
110	39.89	6.73	55.45	3.75	11.43	2.69	31.94	49.56	94.61	44.17

Test Series 101-110 (Cyclone Overflow)
Operating Parameters

Test #	Feed Rate (gpm)	Feed Rate (kg/min)	Conc Rate (kg/min)	Conc Rate (Tph/m2)	Wash Water	Air Meter in H2O	Air Rate lpm	Air Rate (cms)	Air Fraction %	Frother (ml/min)	Collector (Diesel) (ml/min)	Collector (g/T)	Froth Depth (inch)
101	60	9.6	5.1	0.67	14.0	24.0	416	1.5	32	?	0	0	
102	60	6.4	2.7	0.35	14.0	27.5	446	1.6	19	?	0	0	
103	60	10.6	2.6	0.34	14.0	25.8	432	1.6	23	?	0	0	
104	60	11.3	4.5	0.60	14.0	27.0	442	1.6	24	?	0	0	
105	44	7.9	2.6	0.35	14.0	27.0	442	1.6	22	?	0	0	
106	80	16.0	7.6	1.00	12.0	32.0	481	1.8	18	8.4	0	0	
107	110	21.2	9.4	1.24	12.0	42.0	551	2.0	19	9	0	0	
108	100	13.4	3.6	0.47	24.8	41.0	544	2.0	15	9.2	0	0	
109	100	0.0	0.0	0.00	19.8	41.0	544	2.0	?	10	0	0	
110	100	14.6	4.7	0.61	18.0	41.0	544	2.0	15	8	0	0	

Results by Size
Test 104

	Feed			Concentrate			Tails			Individual by Size				Cum. From Bottom			
	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.
+.5 mm	0.05		40.49	0.03		5.72	0.09		61.93								
.5 x .300 mm	0.44	5.70	40.51	0.47	3.46	5.72	0.38	9.24	61.98	61.25	62.70	62.82	25.52	38.16	60.48	94.61	55.09
.300 x .150	3.34	5.04	40.67	3.77	3.09	5.73	2.93	7.75	62.18	58.15	59.35	64.35	23.69	38.12	60.56	94.63	55.19
.150 x .045	17.25	10.44	41.90	22.43	4.75	5.84	13.22	17.38	63.83	54.95	58.44	75.00	33.44	37.81	61.29	94.73	56.02
0.045 mm x 0	78.92	48.78	48.78	73.30	6.17	6.17	83.38	71.20	71.20	34.48	63.16	95.64	58.80	34.48	63.16	95.64	58.80
Cumulative	100.00	40.49		100.00	5.72		100.00	61.93		38.14	60.43	94.61	55.04				

Test 110

	Feed			Concentrate			Tails			Individual by Size				Cum. From Bottom			
	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.
.5 x .300 mm	1.16	18.99	39.86	0.20	3.95	6.69	1.36	20.24	54.66	7.67	9.10	98.40	7.50	30.85	47.87	94.82	42.69
.300 x .150	3.83	13.11	40.11	1.88	3.79	6.70	4.40	17.53	55.14	32.17	35.62	90.70	26.32	31.03	48.34	94.82	43.15
.150 x .045	17.77	12.05	41.19	19.61	5.22	6.76	17.50	16.16	56.89	37.57	40.49	83.73	24.21	31.31	49.64	94.87	44.51
0.045 mm x 0	77.24	47.90	47.90	78.31	7.14	7.14	76.74	66.18	66.18	30.96	55.18	95.38	50.57	30.96	55.18	95.38	50.57
Cumulative	100.00	39.86		100.00	6.69		100.00	54.66		30.85	47.87	94.82	42.69				

Test Series 201 - 215
Test Results

Test #	Ash %			% Solids				% Wt. Yield	Comb. Rec.%	Ash Rej.%	Comb. Sep Eff. %
	Feed	Conc.	Tails	Feed	Conc.	Add Push Water	Tails				
201	46.66	7.63	62.15	5.56	9.09	Y	3.87	28.41	49.20	95.35	44.55
202	46.55	9.55	74.28	4.99	7.98	Y	2.66	42.84	72.49	91.21	63.71
203	42.52	7.19	63.47	4.61	9.14	Y	3.00	37.22	60.10	93.71	53.81
204	33.46	5.97	48.49	6.91	11.06	Y	4.50	35.35	49.95	93.69	43.64
205	33.43	8.69	66.99	7.75	15.31	Y	3.27	57.56	78.96	85.04	63.99
206	39.37	9.02	75.91	4.92	13.44	Y	2.20	54.63	81.97	87.48	69.46
207	40.10	8.62	68.41	5.96	10.58	Y	3.42	47.35	72.23	89.82	62.05
208	36.61	8.55	68.37	5.05	8.88	Y	2.72	53.09	76.59	87.60	64.20
209	35.51	8.82	69.19	5.59	6.94	Y	1.96	55.79	78.88	86.14	65.02
210	37.00	8.45	67.27	5.70	9.76	Y	3.04	51.46	74.78	88.25	63.03
211	33.67	6.83	68.30	4.71	10.31	Y	2.22	56.34	79.13	88.57	67.70
212	34.40	7.08	72.03	4.78	10.10	Y	2.08	57.94	82.07	88.08	70.14
213	35.33	7.85	73.65	4.44	9.52	Y	1.75	58.24	82.98	87.06	70.04
214	34.65	7.34	68.27	5.54	9.48	Y	2.31	55.18	78.24	88.31	66.55
215	35.50	8.14	63.68	5.05	9.96	Y	2.40	50.74	72.26	88.37	60.63

Test Series 201 - 215
Operating Parameters

Test #	Feed Rate (gpm)	Feed Rate kg/min	Conc Rate kg/min	Wash Water gpm	Air Rate (cms)	Air Fraction %	Frother (ml/min)	Collector (Diesel) (ml/min)	Collector (g/T)	Froth Depth (inch)
201	100	21.7	6.2	12.2	2	16	8.7	0	0	25
202	100	19.4	8.3	12.2	2	23	11.6	0	0	25
203	120	21.5	8.0	15.4	2	19	11.6	0	0	25
204	80	21.5	7.6	13.9	2	17	8.8	0	0	27
205	60	18.1	10.4	14.4	2	23	7.5	0	0	27
206	80	15.3	8.4	18.0	2	19	9.6	18	641	24
207	100	23.2	11.0	18.0	2	17	9.6	18	405	22
208	115	22.6	12.0	18.0	2	18	11.2	18	624	24
209	100	21.8	12.1	18.0	2	17	10.8	18	564	22
210	100	22.2	11.4	18.0	1.5	12	10.8	18	553	25
211	100	18.3	10.3	20.0	2	17	7.9	18	641	29
212	100	18.6	10.8	20.0	2	17	9.2	27	948	29
213	100	17.3	10.1	20.0	2	17	9.2	36	1361	28
214	100	21.6	11.9	20.0	2	17	7.9	18	545	31
215	100	19.7	10.0	20.0	2	17	9.2	18	598	19

Results by Size

Test 203

	Feed			Concentrate			Tails			Individual by Size				Cum. From Bottom			
	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.
+.150 mm	6.02	17.04	42.19	8.36	4.16	6.99	4.61	30.09	63.33	50.33	58.14	87.71	45.85	37.52	60.37	93.78	54.15
.150 x .075	9.85	16.06	43.80	14.29	4.98	7.25	7.88	30.66	64.93	56.85	64.36	82.37	46.73	36.64	60.46	93.93	54.40
.075 x .045	7.91	18.69	47.05	11.87	6.30	7.67	6.67	34.07	68.02	55.38	63.82	81.33	45.15	34.75	60.59	94.33	54.93
.045 mm x 0	76.22	49.99	49.99	65.48	7.92	7.92	80.84	70.82	70.82	33.12	60.97	94.75	55.73	33.12	60.97	94.75	55.73
Cumulative	100.00	42.19		100.00	6.99		100.00	63.33		37.52	60.37	93.78	54.15				
Head Ash		42.52			7.19			63.47		37.22	60.10	93.71	53.81				

Test 204

	Feed			Concentrate			Tails			Individual by Size				Cum. From Bottom			
	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.
+.150 mm	30.46	9.46	32.46	23.35	3.57	5.78	29.28	13.66	48.00	41.63	44.33	84.29	28.62	36.80	51.34	93.45	44.80
.150 x .075	11.02	17.62	42.54	14.58	5.36	6.45	9.92	28.34	62.22	46.65	53.59	85.81	39.40	35.29	57.46	94.65	52.11
.075 x .045	6.31	25.15	47.23	9.45	6.38	6.70	6.09	40.55	67.75	45.07	56.37	88.57	44.94	33.61	59.43	95.23	54.66
.045 mm x 0	52.21	49.90	49.90	52.62	6.76	6.76	54.71	70.78	70.78	32.61	60.70	95.58	56.28	32.61	60.70	95.58	56.28
Cumulative	100.00	32.46		100.00	5.78		100.00	48.00		36.81	51.35	93.45	44.79				
Head Ash		33.46			5.97			48.49		35.35	49.95	93.69	43.64				

Test 205

	Feed			Concentrate			Tails			Individual by Size				Cum. From Bottom			
	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.
+.150 mm	32.93	10.75	33.49	43.73	6.11	8.46	22.35	25.02	67.00	75.46	79.39	57.11	36.50	57.25	78.79	85.53	64.32
.150 x .075	10.23	21.20	44.65	13.72	10.98	10.29	5.99	60.49	79.09	79.36	89.65	58.90	48.55	50.05	81.13	88.46	69.59
.075 x .045	6.07	27.91	48.87	9.32	10.82	10.07	4.65	67.38	80.64	69.78	86.33	72.95	59.27	45.01	79.18	90.72	69.90
.045 mm x 0	50.77	51.38	51.38	33.23	9.86	9.86	67.01	81.56	81.56	42.09	78.04	91.92	69.96	42.09	78.04	91.92	69.96
Cumulative	100.00	33.49		100.00	8.46		100.00	67.00		57.24	78.79	85.54	64.33				
Head Ash		33.43			8.69			66.99		57.56	78.96	85.04	63.99				

Test 206

	Feed			Concentrate			Tails			Individual by Size				Cum. From Bottom			
	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.
+.300 mm	0.86	12.78	39.55	1.61	6.78	8.72	0.64	31.48	75.99	75.71	80.92	59.84	40.75	54.17	81.80	88.06	69.85
.300 x .150	4.24	14.64	39.78	8.39	7.59	8.75	1.32	56.51	76.27	85.59	92.66	55.63	48.28	54.05	81.90	88.11	70.01
.150 x 0.045	15.91	15.93	40.90	30.97	8.38	8.86	6.27	55.76	76.54	84.07	91.61	55.78	47.39	52.66	81.21	88.59	69.80
0.045 mm x 0	78.99	45.93	45.93	59.03	9.11	9.11	91.77	77.96	77.96	46.52	78.20	90.77	68.97	46.52	78.20	90.77	68.97
Cumulative	100.00	39.55		100.00	8.72		100.00	75.99		54.17	81.80	88.06	69.85				
Head Ash		39.37			9.02			75.91		54.63	81.97	87.48	69.46				

Test 207

	Feed			Concentrate			Tails			Individual by Size				Cum. From Bottom			
	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.
+.300 mm	1.14	11.76	39.66	1.25	4.77	8.48	2.63	21.65	69.25	58.59	63.23	76.24	39.47	48.69	73.85	89.59	63.45
.300 x .150	4.45	13.02	39.98	7.71	5.62	8.53	2.16	32.77	70.54	72.74	78.93	68.60	47.53	49.27	75.10	89.49	64.59
.150 x 0.045	17.92	13.32	41.26	31.70	7.07	8.77	7.82	40.18	71.40	81.12	86.97	56.94	43.91	48.13	74.74	89.77	64.51
0.045 mm x 0	76.49	47.80	47.80	59.34	9.68	9.68	87.39	74.19	74.19	40.91	70.78	91.72	62.50	40.91	70.78	91.72	62.50
Cumulative	100.00	39.66		100.00	8.48		100.00	69.25		48.69	73.85	89.59	63.44				
Head Ash		40.10			8.62			68.41		47.35	72.23	89.82	62.05				

Test 208

	Feed			Concentrate			Tails			Individual by Size				Cum. From Bottom			
	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.
+.300 mm	1.27	12.11	37.88	1.59	6.11	8.41	0.89	21.37	67.86	60.68	64.82	69.38	34.21	50.43	74.36	88.80	63.16
.300 x .150	4.85	13.04	38.21	9.17	5.92	8.45	1.96	36.64	68.28	76.82	83.11	65.12	48.24	50.26	74.46	88.89	63.35
.150 x 0.045	19.21	14.52	39.51	33.06	7.04	8.71	8.28	39.92	68.92	77.25	84.01	62.55	46.56	48.84	73.71	89.24	62.95
0.045 mm x 0	74.67	45.94	45.94	56.18	9.69	9.69	88.87	71.62	71.62	41.47	69.27	91.25	60.53	41.47	69.27	91.25	60.53
Cumulative	100.00	37.88		100.00	8.41		100.00	67.86		50.43	74.35	88.80	63.16				
Head Ash		36.61			8.55			68.37		53.09	76.59	87.60	64.20				

Test 209

	Feed			Concentrate			Tails			Individual by Size				Cum. From Bottom			
	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.
+.300 mm	1.54	13.33	34.43	1.94	8.12	8.85	1.01	22.87	69.99	64.68	68.57	60.60	29.17	58.16	80.85	85.05	65.90
.300 x .150	5.54	13.69	34.76	9.48	6.51	8.87	1.59	43.20	70.47	80.43	87.12	61.75	48.87	57.97	80.98	85.22	66.19
.150 x 0.045	21.31	13.20	36.02	34.05	7.50	9.12	7.75	43.72	70.92	84.26	89.80	52.12	41.92	56.47	80.22	85.70	65.92
0.045 mm x 0	71.61	42.81	42.81	54.53	10.13	10.13	89.65	73.27	73.27	48.24	75.81	88.58	64.39	48.24	75.81	88.58	64.39
Cumulative	100.00	34.43		100.00	8.85		100.00	69.99		58.16	80.85	85.05	65.90				
Head Ash		35.51			8.82			69.19		55.79	78.88	86.14	65.02				

Test 210

	Feed			Concentrate			Tails			Individual by Size				Cum. From Bottom			
	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.
+.300 mm	1.36	12.78	37.87	2.22	6.14	8.28	0.90	27.68	68.07	69.17	74.44	66.77	41.21	50.51	74.56	88.96	63.52
.300 x .150	5.30	13.49	38.22	10.28	7.32	8.33	1.84	48.35	68.44	84.96	91.02	53.90	44.92	50.27	74.60	89.05	63.64
.150 x 0.045	20.20	14.65	39.62	33.26	7.54	8.45	7.92	42.97	68.82	79.93	86.59	58.86	45.45	48.36	73.33	89.69	63.02
0.045 mm x 0	73.14	46.52	46.52	54.24	9.00	9.00	89.34	71.11	71.11	39.59	67.37	92.34	59.71	39.59	67.37	92.34	59.71
Cumulative	100.00	37.87		100.00	8.28		100.00	68.07		50.51	74.57	88.96	63.52				
Head Ash		37.00			8.45			67.27		51.46	74.78	88.25	63.03				

Test 211

	Feed			Concentrate			Tails			Individual by Size				Cum. From Bottom			
	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.
+.300 mm	1.14	7.54	33.67	1.25	4.54	6.83	0.72	13.53	68.30	66.63	68.79	59.88	28.67	56.34	79.13	88.57	67.71
.300 x .150	5.10	8.16	33.97	8.58	4.96	6.86	1.61	24.66	68.69	83.76	86.67	49.09	35.76	56.16	79.22	88.66	67.88
.150 x 0.045	21.87	10.62	35.37	34.45	6.08	7.04	8.86	37.45	69.42	85.53	89.87	51.04	40.91	54.58	78.51	89.14	67.65
0.045 mm x 0	71.89	42.90	42.90	55.72	7.63	7.63	88.81	72.61	72.61	45.72	73.96	91.87	65.83	45.72	73.96	91.87	65.83
Cumulative	100.00	33.67		100.00	6.83		100.00	68.30		56.34	79.13	88.57	67.70				

Test 212

	Feed			Concentrate			Tails			Individual by Size				Cum. From Bottom			
	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.
+.300 mm	1.27	9.01	34.40	1.86	4.79	7.08	0.84	18.69	72.03	69.64	72.87	62.98	35.85	57.94	82.07	88.07	70.14
.300 x .150	5.89	11.39	34.72	10.63	5.88	7.12	1.77	38.59	72.48	83.15	88.33	57.07	45.40	57.77	82.20	88.15	70.34
.150 x 0.045	20.79	11.92	36.20	39.36	6.56	7.28	7.58	45.78	73.09	86.33	91.59	52.49	44.07	56.05	81.46	88.74	70.20
0.045 mm x 0	72.05	43.21	43.21	48.15	7.86	7.86	89.81	75.40	75.40	47.66	77.33	91.33	68.66	47.66	77.33	91.33	68.66
Cumulative	100.00	34.40		100.00	7.08		100.00	72.03		57.94	82.07	88.08	70.14				

Test 213

	Feed			Concentrate			Tails			Individual by Size				Cum. From Bottom			
	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.
+.300 mm	1.47	9.36	35.33	1.78	5.34	7.85	0.77	19.15	73.65	70.89	74.03	59.56	33.59	58.23	82.98	87.06	70.04
.300 x .150	5.75	11.43	35.72	9.63	6.05	7.89	1.63	42.71	74.07	85.32	90.51	54.84	45.34	57.95	83.04	87.19	70.23
.150 x 0.045	22.42	13.77	37.23	32.49	7.12	8.10	7.13	49.05	74.60	84.14	90.63	56.49	47.12	56.20	82.27	87.78	70.05
0.045 mm x 0	70.36	44.70	44.70	56.10	8.66	8.66	90.47	76.61	76.61	46.96	77.57	90.90	68.47	46.96	77.57	90.90	68.47
Cumulative	100.00	35.33		100.00	7.85		100.00	73.65		58.24	82.98	87.06	70.04				

Test 214

	Feed			Concentrate			Tails			Individual by Size				Cum. From Bottom			
	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.
+.300 mm	1.20	9.67	34.65	1.57	5.41	7.34	1.10	17.06	68.27	63.43	66.43	64.51	30.94	55.18	78.24	88.31	66.55
.300 x .150	5.91	12.08	34.95	10.27	5.24	7.37	2.45	31.62	68.84	74.07	79.83	67.87	47.70	55.13	78.51	88.37	66.88
.150 x 0.045	21.21	13.65	36.41	33.14	6.43	7.62	9.37	40.00	69.79	78.49	85.06	63.03	48.08	53.70	78.00	88.76	66.76
0.045 mm x 0	71.68	43.14	43.14	55.02	8.34	8.34	87.08	72.99	72.99	46.17	74.43	91.07	65.50	46.17	74.43	91.07	65.50
Cumulative	100.00	34.65		100.00	7.34		100.00	68.27		55.18	78.24	88.31	66.55				

Test 215

	Feed			Concentrate			Tails			Individual by Size				Cum. From Bottom			
	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.
+.300 mm	0.98	8.16	35.50	1.30	4.85	8.14	1.42	18.13	63.88	75.08	77.78	55.38	33.16	50.92	72.51	88.32	60.83
.300 x .150	4.94	9.57	35.77	8.35	4.97	8.19	3.24	25.07	64.54	77.11	81.04	59.95	40.99	51.05	72.97	88.32	61.29
.150 x 0.045	20.14	12.01	37.14	30.65	6.33	8.48	10.43	32.47	65.88	78.27	83.32	58.75	42.07	50.06	72.89	88.56	61.46
0.045 mm x 0	73.94	43.99	43.99	59.70	9.59	9.59	84.91	69.98	69.98	43.04	69.47	90.62	60.09	43.04	69.47	90.62	60.09
Cumulative	100.00	35.50		100.00	8.14		100.00	63.88		50.91	72.51	88.33	60.84				

Test Series 301-302
Test Results

Test #	Ash %			% Solids				% Wt. Yield	Comb. Rec.%	Ash Rej.%	Comb. Sep Eff. %
	Feed	Conc.	Tails	Feed	Conc.	Add Push Water	Tails				
301	38.88	8.25	55.52	5.97	10.76	N	3.23	35.20	52.84	92.53	45.37
302	40.23	12.47	59.17	7.91	9.55	N	2.50	40.56	59.39	87.43	46.82

Test Series 301-302
Operating Parameters

Test #	Feed Rate (gpm)	Feed Rate (kg/min)	Conc Rate (kg/min)	Conc Rate (Tph/m2)	Wash Water	Air Rate lpm	Air Rate (cms)	Air Fraction %	Frother (ml/min)	Collect (Diesel) (ml/min)	Coll. (g/T)	Feed Sump (gpm)	Froth Depth (inch)
301	100	23.3	8.2	1.08	14.4	544	2.0	22	10.5	0	0	110	20
302	100	30.8	12.5	1.64	14.4	544	2.0	18	7	25	579	110	26

Results by Size

Test 301

	Feed			Concentrate			Tails			Individual by Size				Cum. From Bottom			
	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.
+.300 mm	3.51	16.47	38.88	2.44	3.26	8.25	4.12	17.35	55.52	6.25	7.23	98.76	6.00	35.19	52.83	92.53	45.36
.300 x .150	10.42	16.80	39.70	14.48	4.64	8.38	9.35	22.75	57.16	32.85	37.66	90.93	28.58	35.79	54.38	92.45	46.83
.150 x 0.045	21.76	23.50	42.47	30.84	7.36	9.03	19.75	29.55	60.88	27.26	33.02	91.46	24.48	35.50	56.13	92.46	48.59
0.045 mm x 0	64.31	48.89	48.89	52.24	10.01	10.01	66.78	70.14	70.14	35.34	62.22	92.76	54.99	35.34	62.22	92.76	54.99
Cumulative	100.00	38.88		100.00	8.25		100.00	55.52		35.20	52.84	92.53	45.37				

Test 302

	Feed			Concentrate			Tails			Individual by Size				Cum. From Bottom			
	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.
+.300 mm	3.57	16.20	39.21	1.24	4.83	10.54	4.82	18.44	58.01	16.46	18.69	95.09	13.78	39.61	58.29	89.35	47.64
.300 x .150	11.11	19.11	40.06	8.91	4.85	10.61	12.15	26.31	60.02	33.55	39.47	91.49	30.95	40.39	60.23	89.30	49.54
.150 x 0.045	22	24.49	42.79	28.40	7.12	11.18	20.02	42.35	64.95	50.70	62.36	85.26	47.62	41.21	63.98	89.23	53.21
0.045 mm x 0	63.32	49.15	49.15	61.45	13.06	13.06	63.01	72.13	72.13	38.90	66.51	89.66	56.18	38.90	66.51	89.66	56.18
Cumulative	100.00	39.21		100.00	10.54		100.00	58.01		39.60	58.28	89.35	47.64				

First Parametric Series (401-416)
Test Results

	Total Ash %			% Solids						
Test #	Feed	Conc.	Tails	Feed	Conc.	Tails	% Wt. Yield	Comb. Rec.%	Ash Rej.%	Comb. Sep Eff. %
401	34.36	12.06	52.65	7.96	13.40	5.60	45.06	60.37	84.18	44.55
402	36.07	9.47	58.54	7.62	15.77	4.88	45.79	64.84	87.98	52.82
403	40.67	9.03	60.75	11.22	11.95	6.62	38.82	59.53	91.38	50.91
404	36.76	8.51	48.11	13.73	12.63	10.00	28.66	41.47	93.36	34.83
405	38.50	10.73	72.58	7.11	16.79	5.02	55.10	79.98	84.64	64.62
406	39.85	10.64	58.34	17.23	17.51	7.73	38.76	57.59	89.65	47.24
407	41.40	11.67	68.07	11.93	21.53	6.93	47.29	71.28	86.67	57.95
408	41.00	8.86	51.37	14.33	13.00	8.70	24.39	37.68	94.73	32.41
409	37.92	8.84	50.42	9.97	12.95	8.52	30.06	44.14	92.99	37.14
410	40.21	9.48	55.41	10.07	15.68	6.46	33.09	50.10	92.20	42.30
411	41.91	8.80	54.59	12.98	12.13	8.40	27.69	43.48	94.19	37.66
412	39.65	9.71	55.22	10.69	13.87	9.04	34.21	51.19	91.62	42.81
413	39.99	13.91	50.96	14.52	9.80	10.88	29.61	42.48	89.70	32.18
414	44.65	12.64	71.30	9.09	16.42	5.62	45.43	71.71	87.14	58.84
415	46.02	11.70	66.51	10.38	18.60	5.00	37.38	61.15	90.50	51.65
416	43.62	12.57	69.22	7.44	17.73	5.85	45.19	70.08	86.98	57.05

First Parametric Series (401-416)
Operating Parameters

Test #	Feed Rate (gpm)	Feed Rate (kg/min)	Conc Rate (kg/min)	Conc Rate (Tph/m2)	Conc Rate -.5 mm	Wash Water lpm	Air Rate lpm	Air Rate (cms)	Air Fraction %	Frother (ml/min)	Collect (Diesel) (ml/min)	Coll. (g/T)	Coll. -.5 mm (g/T)	Feed Sump (gpm)	Froth Depth (inch)
401	100	31.0	14.0	1.84		48	544	2.0	17	10.5	30	690	920	110	24
402	100	29.7	13.6	1.79		68	465	1.7	15	11.6	30	721	961	110	28
403	100	43.7	17.0	2.23	2.00	68	425	1.6	17	11.6	30	518	696	104	28
404	80	42.8	12.3	1.61	1.55	55	425	1.6	20	11.6	30	423	552	104	24
405	80	22.2	12.2	1.61	1.32	55	425	1.6	13	11.6	48	1307	1805	104	28
406	60	40.3	15.6	2.05	1.78	55	425	1.6	13	24	48	539	753	104	30
407	60	27.9	13.2	1.73	1.32	55	425	1.6	17	10	30	487	707	104	28
408	80	44.7	10.9	1.43	1.41	68	425	1.6	18	10	15	201	254	105	26
409	100	38.8	11.7	1.54	1.49	68	425	1.6	?	11	18	346	445	105	22
410	100	39.2	13.0	1.71	1.60	68	425	1.6	?	11	30	571	745	105	24
411	90	45.5	12.6	1.66	1.61	68	425	1.6	?	11	30	405	530	115	26
412	103	42.9	14.7	1.93	1.74	82	425	1.6	?	10	40	731	1007	103	Turb
413	80	45.2	13.4	1.76	1.63	68	408	1.5	?	10	30	520	758		32
414	80	28.3	12.9	1.69	1.35	67	408	1.5	?	8	30	831	1249		32
415	80	32.3	12.1	1.59	1.29	65	425	1.6	?	8	45	1091	1521		32
416	95	27.5	12.4	1.64	1.38	64	425	1.6	?	8	45	1282	2020		32

Results by Size

Test 401

	Feed			Concentrate			Tails			Individual by Size				Cum. From Bottom			
	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.
			34.36			12.06			52.65					45.06	60.36	84.18	44.54
			34.36			12.06			52.65					45.06	60.36	84.18	44.54
+0.300 mm	35.44	29.30	34.36	17.32	6.28	12.06	51.57	36.23	52.65	23.14	30.67	95.04	25.71	45.06	60.36	84.18	44.54
.300 x.150	13.42	32.22	37.14	21.45	11.32	13.27	11.65	57.93	70.13	55.16	72.17	80.62	52.79	58.03	80.05	79.26	59.31
.150 x .045	16.26	25.68	38.43	28.16	12.22	13.96	10.17	65.90	73.99	74.93	88.50	64.35	52.84	59.24	82.78	78.48	61.26
.045 mm x 0	34.88	44.37	44.37	33.07	15.44	15.44	26.61	77.08	77.08	53.07	80.66	81.53	62.20	53.07	80.66	81.53	62.20
Cumulative	100.00	34.36		100.00	12.06		100.00	52.65		45.06	60.37	84.18	44.55				

Test 402

	Feed			Concentrate			Tails			Individual by Size				Cum. From Bottom			
	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.
			36.07			9.47			58.54					45.79	64.85	87.98	52.83
			36.07			9.47			58.54					45.79	64.85	87.98	52.83
+0.300 mm	38.02	30.09	36.07	25.97	6.13	9.47	47.36	41.92	58.54	33.05	44.38	93.27	37.65	45.79	64.85	87.98	52.83
.300 x.150	14.54	37.40	39.74	22.19	10.23	10.64	12.16	66.43	73.49	51.65	74.07	85.87	59.95	53.71	79.64	85.62	65.26
.150 x .045	16.38	32.45	40.45	25.99	11.19	10.81	10.20	70.43	75.61	64.11	84.29	77.89	62.18	54.26	81.27	85.50	66.77
.045 mm x 0	31.06	44.67	44.67	25.85	10.43	10.43	30.28	77.36	77.36	48.84	79.07	88.60	67.66	48.84	79.07	88.60	67.66
Cumulative	100.00	36.07		100.00	9.47		100.00	58.54		45.79	64.84	87.98	52.82				

Test 403

	Feed			Concentrate			Tails			Individual by Size				Cum. From Bottom			
	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.
+1mm	3.25	36.23	40.67	0.13	6.07	9.03	4.96	42.18	60.75	16.48	24.27	97.24	21.51	38.82	59.53	91.38	50.91
1 x .5 mm	22.42	32.59	40.82	10.10	4.07	9.04	25.43	38.78	61.72	17.83	25.38	97.77	23.15	39.67	60.98	91.22	52.20
.5 x .250 mm	13.94	40.68	43.30	16.19	6.80	9.59	12.08	54.43	70.10	28.87	45.36	95.17	40.53	44.29	70.62	90.19	60.81
.250 x .150	10.57	43.59	43.91	14.67	9.17	10.21	7.39	64.82	73.39	38.15	61.43	91.97	53.40	46.66	74.70	89.15	63.85
.150 x .045	16.76	34.30	43.98	27.63	9.91	10.47	10.90	64.12	74.65	55.01	75.43	84.11	59.54	47.80	76.38	88.62	65.01
.045 mm x 0	33.06	48.88	48.88	31.28	10.96	10.96	39.24	77.58	77.58	43.08	75.04	90.34	65.38	43.08	75.04	90.34	65.38
Cumulative	100.00	40.67		100.00	9.03		100.00	60.75		38.82	59.53	91.38	50.91				

Test 404

	Feed			Concentrate			Tails			Individual by Size				Cum. From Bottom			
	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.
+1mm	3.05	29.85	36.76	0.07	18.74	8.51	5.52	31.37	48.11	12.03	13.94	92.44	6.39	28.67	41.47	93.36	34.83
1 x .5 mm	20.26	27.18	36.98	4.09	3.23	8.51	28.21	28.87	49.09	6.59	8.76	99.22	7.98	29.85	43.33	93.13	36.46
.5 x .250 mm	13.28	33.34	39.57	12.79	4.75	8.73	13.64	37.41	57.70	12.46	17.81	98.22	16.03	37.03	55.92	91.83	47.75
.250 x .150	10.18	35.95	40.87	13.62	7.47	9.34	7.58	42.65	62.95	19.04	27.51	96.04	23.56	41.20	63.16	90.58	53.74
.150 x .045	16.61	29.44	41.81	28.49	8.75	9.71	11.06	49.56	66.37	49.30	63.76	85.35	49.10	43.35	67.26	89.93	57.19
.045 mm x 0	36.62	47.42	47.42	40.94	10.38	10.38	33.99	71.84	71.84	39.73	67.72	91.30	59.03	39.73	67.72	91.30	59.03
Cumulative	100.00	36.76		100.00	8.51		100.00	48.11		28.66	41.47	93.36	34.83				

Test 405

	Feed			Concentrate			Tails			Individual by Size				Cum. From Bottom			
	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.
+1mm	4.72	34.06	38.50	0.50	2.89	10.73	5.67	49.35	72.58	32.91	48.47	97.21	45.67	55.10	79.98	84.64	64.62
1 x .5 mm	22.86	31.44	38.72	17.51	5.28	10.77	26.36	55.63	73.97	48.04	66.38	91.93	58.31	55.78	81.22	84.48	65.70
.5 x .250 mm	14.53	37.88	41.02	17.58	9.98	11.95	12.68	75.49	81.09	57.41	83.20	84.87	68.07	57.95	86.52	83.12	69.64
.250 x .150	10.68	39.97	41.81	14.58	14.11	12.48	8.06	83.94	82.37	62.97	90.09	77.77	67.86	58.04	87.29	82.67	69.96
.150 x .045	16.86	36.40	42.23	23.25	13.08	12.01	11.00	82.32	82.11	66.32	90.64	76.17	66.81	56.89	86.65	83.82	70.47
.045 mm x 0	30.37	45.46	45.46	26.58	11.07	11.07	36.23	82.04	82.04	51.54	84.04	87.45	71.49	51.54	84.04	87.45	71.49
Cumulative	100.02	38.50		100.00	10.73		100.00	72.58		55.10	79.98	84.64	64.62				

Test 406

	Feed			Concentrate			Tails			Individual by Size				Cum. From Bottom			
	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.
+1mm	3.66	37.28	39.85	0.30	3.71	10.84	6.88	38.74	58.34	4.17	6.40	99.59	5.98	38.92	57.69	89.41	47.10
1 x .5 mm	24.68	34.27	39.95	13.08	4.67	10.87	32.96	38.13	59.79	11.54	16.73	98.43	15.16	40.55	60.19	88.97	49.16
.5 x .250 mm	15.20	34.90	41.91	17.43	8.30	11.80	13.75	54.66	71.66	42.62	60.04	89.86	49.90	49.70	75.46	86.01	61.46
.250 x .150	10.20	37.13	43.80	13.22	11.91	12.68	8.00	69.12	76.69	55.92	78.35	82.06	60.41	51.39	79.84	85.12	64.96
.150 x .045	15.53	36.89	45.27	22.25	12.13	12.87	9.67	73.15	78.27	59.42	82.74	80.46	63.20	50.46	80.33	85.66	65.99
.045 mm x 0	30.73	49.50	49.50	33.72	13.35	13.35	28.74	79.99	79.99	45.75	78.51	87.66	66.17	45.75	78.51	87.66	66.17
Cumulative	100.00	39.85		100.00	10.84		100.00	58.34		38.93	57.70	89.41	47.11				

Test 407

	Feed			Concentrate			Tails			Individual by Size				Cum. From Bottom			
	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.
+1mm	6.87	40.56	41.40	0.76	3.01	11.67	9.02	50.59	68.07	21.08	34.40	98.44	32.83	47.28	71.27	86.68	57.95
1 x .5 mm	24.27	35.88	41.46	22.98	6.09	11.73	27.78	54.61	69.80	38.60	56.54	93.45	49.99	48.80	73.59	86.19	59.78
.5 x .250 mm	14.92	38.86	43.43	16.60	11.24	13.44	13.58	64.57	76.48	48.21	69.99	86.06	56.04	52.42	80.22	83.78	64.00
.250 x .150	9.98	40.16	44.70	13.93	15.99	14.05	7.24	71.13	79.74	56.17	78.85	77.64	56.49	53.34	82.91	83.24	66.14
.150 x .045	14.50	37.23	45.73	20.73	15.81	13.45	8.92	75.89	81.21	64.35	86.31	72.67	58.98	52.37	83.51	84.59	68.10
.045 mm x 0	29.46	49.91	49.91	25.00	11.50	11.50	33.46	82.63	82.63	46.00	81.27	89.40	70.68	46.00	81.27	89.40	70.68
Cumulative	100.00	41.40		100.00	11.67		100.00	68.07		47.29	71.28	86.67	57.95				

Test 408

	Feed			Concentrate			Tails			Individual by Size				Cum. From Bottom			
	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.
+1mm	3.04	36.06	41.00	0.01	24.31	8.86	4.61	37.56	51.37	11.32	13.40	92.37	5.77	24.40	37.69	94.72	32.42
1 x .5 mm	17.78	35.09	41.15	1.92	11.38	8.86	26.64	38.08	52.04	11.20	15.29	96.37	11.66	25.21	39.05	94.57	33.62
.5 x .250 mm	16.58	37.85	42.52	14.08	5.18	8.81	15.21	39.61	57.45	5.11	7.80	99.30	7.10	30.70	48.70	93.64	42.34
.250 x .150	9.43	35.75	43.75	14.73	7.44	9.42	7.73	46.24	62.52	27.04	38.95	94.37	33.32	35.34	56.91	92.39	49.30
.150 x .045	15.39	31.28	45.17	29.34	8.79	9.84	11.12	45.65	65.26	38.99	51.74	89.04	40.79	36.25	59.61	92.10	51.71
.045 mm x 0	37.78	50.83	50.83	39.92	10.62	10.62	34.69	71.55	71.55	34.01	61.82	92.90	54.71	34.01	61.82	92.90	54.71
Cumulative	100.00	41.00		100.00	8.86		100.00	51.37		24.39	37.68	94.73	32.41				

Test 409

	Feed			Concentrate			Tails			Individual by Size				Cum. From Bottom			
	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.
+1mm	3.41	32.30	37.92	0.39	22.82	8.84	7.86	29.23	50.42	0.00	0.00	0.00	-100.00	30.06	44.14	92.99	37.13
1 x .5 mm	18.81	29.67	38.12	2.90	4.96	8.79	29.64	31.46	52.23	6.75	9.13	98.87	8.00	32.48	47.87	92.51	40.38
.5 x .250 mm	16.17	31.18	40.16	16.65	5.52	8.90	16.66	43.01	62.07	31.56	43.32	94.41	37.73	41.21	62.74	90.87	53.60
.250 x .150	9.25	26.66	42.52	14.73	7.71	9.61	6.34	54.33	69.00	59.35	74.69	82.84	57.52	44.59	70.12	89.93	60.05
.150 x .045	14.79	26.24	45.32	27.96	9.18	10.03	8.78	57.35	71.36	64.58	79.52	77.41	56.93	42.46	69.86	90.60	60.46
.045 mm x 0	37.57	52.83	52.83	37.37	10.67	10.67	30.72	75.36	75.36	34.83	65.96	92.97	58.92	34.83	65.96	92.97	58.92
Cumulative	100.00	37.92		100.00	8.84		100.00	50.42		30.06	44.14	92.99	37.14				

Test 410

	Feed			Concentrate			Tails			Individual by Size				Cum. From Bottom			
	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.
+1mm	4.15	33.49	40.21	0.08	6.26	9.48	8.15	36.5	55.41	9.95	14.03	98.14	12.17	33.09	50.10	92.20	42.30
1 x .5 mm	19.20	33.80	40.50	6.04	3.83	9.48	30.47	36.53	57.09	8.35	12.13	99.05	11.18	34.84	53.01	91.85	44.85
.5 x .250 mm	18.86	37.21	42.18	20.95	6.71	9.84	14.84	51.40	67.30	31.75	47.18	94.27	41.45	43.71	68.16	89.80	57.96
.250 x .150	9.28	36.13	43.81	14.76	10.34	10.74	6.42	62.98	72.37	51.01	71.60	85.40	57.01	46.35	73.62	88.64	62.25
.150 x .045	14.53	29.67	45.28	27.17	10.69	10.84	7.94	59.98	73.87	61.49	78.09	77.84	55.93	45.37	73.92	89.13	63.05
.045 mm x 0	33.98	51.95	51.95	31.00	10.98	10.98	32.18	77.30	77.30	38.22	70.82	91.92	62.74	38.22	70.82	91.92	62.74
Cumulative	100.00	40.21		100.00	9.48		100.00	55.41		33.09	50.10	92.20	42.30				

Test 411

	Feed			Concentrate			Tails			Individual by Size				Cum. From Bottom			
	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.
+1mm	4.12	33.61	41.91	0.03	14.71	8.80	5.57	35.93	54.59	10.93	14.05	95.21	9.26	27.68	43.46	94.19	37.65
1 x .5 mm	19.59	35.50	42.27	2.62	3.74	8.79	24.4	39.02	55.69	9.98	14.89	98.95	13.84	28.62	45.21	94.05	39.26
.5 x .250 mm	16.86	42.34	44.01	14.72	5.35	8.93	16.26	45.92	61.49	8.82	14.49	98.88	13.37	33.27	54.11	93.25	47.36
.250 x .150	8.93	35.48	44.48	14.63	8.08	9.57	7.55	51.46	66.20	36.84	52.48	91.61	44.09	38.36	62.48	91.75	54.23
.150 x .045	14.30	28.80	46.07	31.47	9.06	9.89	10.06	49.97	68.61	51.75	66.09	83.72	49.82	38.39	64.14	91.76	55.91
.045 mm x 0	36.20	52.89	52.89	36.52	10.60	10.60	36.16	73.80	73.80	33.09	62.79	93.37	56.15	33.09	62.79	93.37	56.15
Cumulative	100.00	41.91		99.99	8.80		100.00	54.59		27.69	43.48	94.19	37.66				

Test 412

	Feed			Concentrate			Tails			Individual by Size				Cum. From Bottom			
	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.
+1mm	5.92	37.41	39.65	0.23	8.51	9.71	9.47	42.34	55.22	14.57	21.30	96.68	17.99	34.21	51.18	91.62	42.80
1 x .5 mm	21.41	29.09	39.79	9.57	5.81	9.71	25.26	41.42	56.56	34.63	45.99	93.08	39.08	35.80	53.68	91.26	44.94
.5 x .250 mm	10.78	31.82	42.94	13.03	5.62	10.13	10.45	46.89	62.42	36.52	50.55	93.55	44.10	37.25	58.67	91.21	49.89
.250 x .150	8.67	30.88	44.88	10.97	5.81	10.89	6.56	47.34	65.38	39.63	54.01	92.54	46.55	37.62	60.83	90.87	51.70
.150 x .045	14.09	27.90	47.16	24.66	6.58	11.73	9.69	47.09	67.84	47.37	61.38	88.83	50.21	36.85	61.56	90.83	52.39
.045 mm x 0	39.13	54.10	54.10	41.54	14.79	14.79	38.57	73.05	73.05	32.53	60.38	91.11	51.49	32.53	60.38	91.11	51.49
Cumulative	100.00	39.65		100.00	9.71		100.00	55.22		34.21	51.19	91.62	42.81				

Test 413

	Feed			Concentrate			Tails			Individual by Size				Cum. From Bottom			
	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.
+1mm	5.01	38.69	39.99	0.16	7.00	13.91	6.75	36.88	50.96	na	na	na	na	29.61	42.48	89.70	32.18
1 x .5 mm	26.36	33.71	40.06	7.58	5.18	13.93	26.77	35.82	51.98	6.89	9.85	98.94	8.79	31.33	44.99	89.11	34.10
.5 x .250 mm	10.65	35.77	42.50	12.84	5.20	14.64	11.51	40.81	58.49	14.15	20.89	97.94	18.83	36.47	54.14	87.43	41.57
.250 x .150	8.61	31.78	43.74	11.54	5.44	16.17	7.75	43.74	62.19	31.23	43.28	94.65	37.94	40.10	59.75	85.17	44.92
.150 x .045	13.97	27.55	45.82	23.00	6.53	17.99	10.94	43.93	65.22	43.80	56.50	89.62	46.12	41.08	62.18	83.87	46.04
.045 mm x 0	35.40	53.03	53.03	44.88	23.87	23.87	36.28	71.64	71.64	38.96	63.14	82.46	45.61	38.96	63.14	82.46	45.61
Cumulative	100.00	39.99		100.00	13.91		100.00	50.96		29.61	42.48	89.70	32.18				

Test 414

	Feed			Concentrate			Tails			Individual by Size				Cum. From Bottom			
	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.
+1mm	7.53	44.54	44.65	0.40	3.04	12.84	8.9	50.93	71.30	13.34	23.33	99.09	22.42	45.58	71.78	86.90	58.68
1 x .5 mm	25.98	42.45	44.66	19.91	5.62	12.88	27.7	56.93	73.29	28.22	46.28	96.26	42.54	47.39	74.61	86.34	60.95
.5 x .250 mm	11.63	39.41	45.53	16.46	11.13	14.69	9.99	71.72	80.44	53.33	78.22	84.94	63.16	53.10	83.16	82.87	66.03
.250 x .150	8.35	36.80	46.83	12.74	15.01	15.61	5.15	76.82	82.07	64.75	87.07	73.59	60.66	53.04	84.17	82.31	66.48
.150 x .045	12.38	29.23	48.63	22.33	15.41	15.77	6.87	73.32	82.63	76.14	91.00	59.86	50.86	50.86	83.39	83.51	66.90
.045 mm x 0	34.13	55.66	55.66	28.16	16.05	16.05	41.39	84.18	84.18	41.86	79.26	87.93	67.19	41.86	79.26	87.93	67.19
Cumulative	100.00	44.65		100.00	12.84		100.00	71.30		45.59	71.79	86.89	58.68				

Test 415

	Feed			Concentrate			Tails			Individual by Size				Cum. From Bottom			
	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.
+1mm	5.66	38.84	46.02	0.42	3.66	11.70	10.66	46.49	66.51	17.86	28.14	98.32	26.45	37.39	61.16	90.50	51.66
1 x .5 mm	22.60	43.04	46.45	18.36	5.95	11.73	33.24	52.96	68.90	21.10	34.84	97.08	31.93	39.27	64.73	90.08	54.82
.5 x .250 mm	13.20	45.32	47.52	16.30	11.21	13.04	10.12	68.47	78.35	40.43	65.65	90.00	55.65	47.20	78.21	87.05	65.27
.250 x .150	9.17	39.95	48.02	12.65	11.95	13.49	5.05	74.73	80.52	55.40	81.23	83.43	64.66	48.49	80.70	86.37	67.07
.150 x .045	12.90	30.97	49.52	22.51	13.31	13.87	6.77	72.95	81.24	70.39	88.40	69.75	58.15	47.08	80.33	86.81	67.15
.045 mm x 0	36.47	56.08	56.08	29.76	14.29	14.29	34.16	82.88	82.88	39.07	76.25	90.04	66.29	39.07	76.25	90.04	66.29
Cumulative	100.00	46.02		100.00	11.70		100.00	66.51		37.38	61.15	90.50	51.65				

Test 416

	Feed			Concentrate			Tails			Individual by Size				Cum. From Bottom			
	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.
+1mm	11.71	42.25	43.26	0.32	4.17	12.57	10.71	46	69.22	8.96	14.88	99.12	13.99	45.84	70.62	86.68	57.30
1 x .5 mm	24.83	42.44	43.39	15.40	5.85	12.60	27.37	54.35	72.01	24.56	40.17	96.62	36.78	48.17	74.37	86.01	60.38
.5 x .250 mm	11.76	39.94	43.76	15.66	11.77	13.83	9.72	71.46	79.81	52.81	77.57	84.44	62.01	54.64	83.72	82.73	66.44
.250 x .150	8.27	34.18	44.63	13.36	12.30	14.30	4.64	78.18	81.37	66.79	88.99	75.97	64.96	54.78	84.78	82.44	67.23
.150 x .045	12.64	29.21	46.62	23.89	14.25	14.79	5.80	73.10	81.68	74.58	90.34	63.62	53.96	52.41	83.67	83.37	67.04
.045 mm x 0	30.79	53.77	53.77	31.37	15.20	15.20	41.76	82.87	82.87	43.00	78.88	87.84	66.72	43.00	78.88	87.84	66.72
Cumulative	100.00	43.26		100.00	12.57		100.00	69.22		45.83	70.61	86.68	57.30				

Second Parametric Series (451-465)
Test Results

Test #	**Ash %**			**% Solids**			**% Wt. Yield**	**Comb. Rec.%**	**Ash Rej.%**	**Comb. Sep Eff. %**
	Feed	**Conc.**	**Tails**	**Feed**	**Conc.**	**Tails**				
451	36.95	11.18	67.83	7.32	12.35	3.11	54.50	76.79	83.52	60.30
452	38.62	11.24	69.13	7.06	15.40	5.23	52.71	76.22	84.66	60.88
453	36.29	11.23	67.32	8.27	12.73	3.40	55.31	77.08	82.89	59.97
454	34.26	11.43	66.05	6.76	12.31	2.33	58.20	78.42	80.59	59.00
455	33.87	10.08	66.29	7.31	15.90	2.76	57.68	78.43	82.84	61.26
456	35.63	11.12	51.39	7.35	14.55	11.38	39.14	54.04	87.78	41.82
457	42.61	10.60	66.45	14.18	12.32	6.28	42.69	66.50	89.38	55.88
458	34.70	9.33	56.41	13.16	15.57	7.64	46.10	64.02	87.61	51.63
459	38.86	10.66	68.06	13.43	12.01	7.44	50.88	74.34	86.05	60.39
460	40.24	11.52	68.43	10.10	15.51	5.75	49.54	73.35	85.81	59.16
461	42.80	11.78	72.56	9.19	14.17	3.86	48.97	75.52	86.52	62.03
462	36.31	12.63	69.73	12.24	10.47	4.81	58.53	80.29	79.64	59.93
463	45.02	12.08	75.53	13.24	17.25	8.66	48.10	76.90	87.09	63.99
464	42.44	12.60	74.06	10.08	11.50	4.90	51.44	78.11	84.74	62.85
465	42.30	12.68	75.10	13.90	13.77	6.32	52.55	79.52	84.25	63.77

Second Parametric Series (451-465)
Operating Parameters

Test #	Feed Rate (gpm)	Feed Rate (kg/min)	Conc Rate (kg/min)	Conc Rate Tph/m2	Conc Rate -.5 mm	Wash Water lpm	Air Rate lpm	Air Rate cms	Air Fraction %	Frother (ml/min)	Collect (Diesel) (ml/min)	Coll. (g/T)	Coll. -.5 mm (g/T)	Froth Depth (inch)
451	41	11.7	6.4	0.84	0.73	54.6	323	1.2	5.0	4.5	16.5	1107	1358	24
452	60	16.5	8.7	1.14	1.03	47.8	350	1.3	3.5	7.3	24	1141	1383	20
453	50	16.1	8.9	1.17	1.02	40.9	306	1.1	3.5	9.3	20	974	1171	26
454	50	13.2	7.7	1.01	0.87	43.7	350	1.3	2.0	9.3	20	1191	1445	24
455	50	14.2	8.2	1.08	0.98	43.7	350	1.3	3.3	5.5	20	1102	1348	28
456	60	17.2	6.7	0.88	0.83	43.7	329	1.2	3.3	11.0	24	1096	1385	25
457	50	27.6	11.8	1.55	1.38	51.8	329	1.2	11.0	8.0	20	568	699	29
458	50	25.6	11.8	1.55	1.37	51.8	329	1.2	11.9	8.0	20	612	765	30
459	50	26.2	13.3	1.75	1.54	47.8	306	1.1	9.3	6.0	20	600	749	27
460	60	23.6	11.7	1.54	1.37	47.8	306	1.1	10.0	9.0	23	764	1041	28
461	40	14.3	7.0	0.92	0.76	47.8	306	1.1	11.0	6.0	16	876	1128	25
462	60	28.6	16.7	2.20	1.90	54.6	329	1.2	9.5	6.3	23	631	745	27
463	40	20.6	9.9	1.31	1.04	50.5	350	1.3	12.9	6.0	16	608	834	25
464	40	15.7	8.1	1.06	0.86	50.5	329	1.2	13.6	7.5	16	799	1136	25
465	50	27.1	14.2	1.87	1.66	50.5	329	1.2	11.3	7.5	19	550	724	28

Results by Size

Test 451

	Feed			Concentrate			Tails			Individual by Size				Cum. From Bottom			
	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.
+1mm	1.98	20.70	36.95	0.75	4.33	11.18	1.9	30.05	67.83	36.35	43.86	92.40	36.25	54.50	76.79	83.52	60.30
1 x .5 mm	16.48	23.66	37.28	11.63	5.32	11.23	13.21	35.35	68.56	38.93	48.28	91.25	39.53	54.56	77.22	83.57	60.79
.5 x .250 mm	19.14	34.31	40.04	24.84	10.02	12.01	12.96	54.39	73.73	45.26	61.99	86.78	48.77	54.60	80.11	83.62	63.73
.250 x .150	9.69	34.51	41.79	13.64	13.59	12.80	8.28	74.13	77.22	65.44	86.35	74.23	60.58	54.99	82.38	83.16	65.54
.150 x .045	16.74	32.43	43.13	23.48	13.50	12.58	13.05	73.04	77.62	68.21	87.31	71.61	58.92	53.03	81.51	84.53	66.05
.045 mm x 0	35.97	48.11	48.11	25.66	11.74	11.74	50.60	78.80	78.80	45.76	77.84	88.83	66.67	45.76	77.84	88.83	66.67
Cumulative	100.00	36.96		100.00	11.18		100.00	67.83		54.49	76.78	83.52	60.29				

Test 452

	Feed			Concentrate			Tails			Individual by Size				Cum. From Bottom			
	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.
+1mm	1.4	24.21	38.62	0.21	3.24	11.24	2.00	40.59	69.13	43.86	55.99	94.13	50.12	52.71	76.22	84.66	60.88
1 x .5 mm	16.11	26.61	38.82	9.99	4.96	11.26	22.06	52.73	69.72	54.68	70.81	89.81	60.62	52.85	76.66	84.68	61.34
.5 x .250 mm	20.97	37.98	41.21	23.04	9.06	11.96	17.92	67.12	74.65	50.19	73.59	88.03	61.62	53.34	79.88	84.52	64.41
.250 x .150	11.22	39.35	42.30	12.35	13.13	12.95	9.73	74.80	76.97	57.48	82.33	80.82	63.15	54.16	81.70	83.42	65.12
.150 x .045	16.84	34.07	42.96	24.82	13.10	12.92	10.57	71.00	77.41	63.78	84.07	75.48	59.54	53.41	81.55	83.94	65.49
.045 mm x 0	33.46	47.44	47.44	29.59	12.76	12.76	37.72	79.21	79.21	47.81	79.36	87.14	66.50	47.81	79.36	87.14	66.50
Cumulative	100.00	38.62		100.00	11.24		100.00	69.13		52.70	76.21	84.66	60.87				

Test 453

	Feed			Concentrate			Tails			Individual by Size				Cum. From Bottom			
	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.
+1mm	1.38	20.21	36.29	0.58	3.54	11.23	1.69	24.30	67.32	19.70	23.82	96.55	20.37	55.31	77.08	82.89	59.97
1 x .5 mm	15.44	24.09	36.52	12.64	5.69	11.27	13.06	36.31	68.06	39.91	49.58	90.57	40.16	55.54	77.63	82.86	60.49
.5 x .250 mm	19.51	38.33	38.82	19.05	9.55	12.08	14.47	58.52	72.92	41.23	60.47	89.73	50.20	56.05	80.55	82.55	63.10
.250 x .150	11.78	38.07	38.98	13.75	13.34	12.80	9.39	73.97	75.87	59.21	82.86	79.25	62.11	58.49	83.58	80.79	64.38
.150 x .045	16.14	29.37	39.18	22.79	12.85	12.66	12.70	71.04	76.16	71.61	88.36	68.67	57.03	58.23	83.62	81.19	64.81
.045 mm x 0	35.75	43.61	43.61	31.19	12.52	12.52	48.69	77.49	77.49	52.15	80.90	85.03	65.93	52.15	80.90	85.03	65.93
Cumulative	100.00	36.29		100.00	11.23		100.00	67.32		55.32	77.08	82.88	59.96				

Test 454

	Feed			Concentrate			Tails			Individual by Size				Cum. From Bottom			
	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.
+1mm	1.67	22.88	34.26	0.75	4.35	11.43	1.95	24.82	66.05	9.48	11.75	98.20	9.95	58.20	78.42	80.59	59.00
1 x .5 mm	15.9	23.81	34.45	12.67	5.70	11.48	13.36	29.31	66.87	23.30	28.83	94.42	23.26	58.53	79.04	80.50	59.54
.5 x .250 mm	17.05	30.19	36.51	20.05	9.18	12.33	12.62	54.77	72.80	53.92	70.14	83.61	53.75	60.02	82.87	79.73	62.61
.250 x .150	11.36	32.11	38.15	14.65	13.62	13.27	6.46	67.78	75.96	65.86	83.80	72.06	55.86	60.31	84.57	79.02	63.59
.150 x .045	17.33	30.37	39.42	24.80	13.95	13.18	10.89	71.25	76.76	71.34	88.17	67.23	55.40	58.72	84.17	80.37	64.54
.045 mm x 0	36.69	43.70	43.70	27.08	12.47	12.47	54.72	77.86	77.86	52.24	81.22	85.09	66.31	52.24	81.22	85.09	66.31
Cumulative	100.00	34.25		100.00	11.43		100.00	66.05		58.22	78.43	80.57	59.00				

Test 455

	Feed			Concentrate			Tails			Individual by Size				Cum. From Bottom			
	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.
+1mm	1.61	20.51	33.87	0.50	3.47	10.08	2.04	22.25	66.29	9.27	11.25	98.43	9.68	57.68	78.43	82.84	61.26
1 x .5 mm	16.66	23.50	34.09	8.49	5.25	10.11	14.67	32.25	67.21	32.41	40.14	92.76	32.90	58.01	79.11	82.79	61.90
.5 x .250 mm	18.25	28.48	36.24	14.03	8.68	10.56	14.61	57.48	73.36	59.43	75.88	81.89	57.77	59.11	82.91	82.77	65.69
.250 x .150	9.56	29.50	38.48	8.03	11.90	10.91	6.38	66.45	76.74	67.74	84.65	72.68	57.32	58.12	84.17	83.52	67.69
.150 x .045	16.04	26.20	40.07	12.93	11.67	10.79	10.53	71.28	77.79	75.62	90.51	66.32	56.83	56.31	83.81	84.83	68.64
.045 mm x 0	37.88	45.94	45.94	56.02	10.59	10.59	51.77	79.12	79.12	48.42	80.08	88.84	68.92	48.42	80.08	88.84	68.92
Cumulative	100.00	33.87		100.00	10.08		100.00	66.29		57.68	78.43	82.83	61.26				

Test 456

	Feed			Concentrate			Tails			Individual by Size				Cum. From Bottom			
	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.
+1mm	2.05	25.47	35.63	0.14	6.60	11.12	2.05	26.72	51.39	6.21	7.79	98.39	6.18	39.14	54.04	87.78	41.82
1 x .5 mm	18.84	26.60	35.84	5.95	3.85	11.13	21.50	29.06	51.90	9.76	12.78	98.59	11.37	39.39	54.57	87.77	42.33
.5 x .250 mm	19.98	33.25	38.04	16.62	5.99	11.59	17.56	41.03	58.33	22.20	31.27	96.00	27.27	43.40	61.93	86.78	48.71
.250 x .150	9.97	30.93	39.66	13.07	7.46	12.79	9.94	50.95	63.48	46.03	61.68	88.90	50.57	47.00	67.93	84.84	52.76
.150 x .045	16.41	29.22	41.43	26.13	8.40	13.88	13.77	50.98	66.03	51.10	66.14	85.31	51.45	47.17	69.36	84.20	53.55
.045 mm x 0	32.75	47.55	47.55	38.09	17.64	17.64	35.18	71.92	71.92	44.90	70.50	83.34	53.84	44.90	70.50	83.34	53.84
Cumulative	100.00	35.63		100.00	11.12		100.00	51.39		39.14	54.04	87.79	41.82				

Test 457

	Feed			Concentrate			Tails			Individual by Size				Cum. From Bottom			
	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.
+1mm	1.51	33.92	42.61	0.35	3.43	10.60	2.03	35.09	66.45	3.70	5.40	99.63	5.03	42.69	66.50	89.38	55.88
1 x .5 mm	17.25	35.16	42.74	10.54	4.66	10.62	15.58	35.89	67.10	2.34	3.44	99.69	3.13	43.13	67.32	89.28	56.60
.5 x .250 mm	19.66	43.18	44.35	21.47	8.76	11.33	12.73	57.15	73.00	28.87	46.36	94.14	40.50	46.46	74.02	88.13	62.16
.250 x .150	11.12	41.43	44.72	12.93	11.14	12.14	7.09	67.85	75.90	46.59	70.68	87.47	58.15	48.90	77.72	86.72	64.44
.150 x .045	17.34	35.06	45.45	22.98	10.31	12.38	11.25	62.63	76.81	52.69	72.78	84.50	57.28	48.67	78.18	86.74	64.92
.045 mm x 0	33.12	50.89	50.89	31.73	13.88	13.88	51.32	79.92	79.92	43.96	77.09	88.01	65.10	43.96	77.09	88.01	65.10
Cumulative	100.00	42.61		100.00	10.60		100.00	66.45		42.69	66.49	89.38	55.88				

Test 458

	Feed			Concentrate			Tails			Individual by Size				Cum. From Bottom			
	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.
+1mm	2.19	26.65	34.70	0.40	4.98	9.33	2.77	29.66	56.41	12.20	15.80	97.72	13.52	46.10	64.02	87.61	51.63
1 x .5 mm	17.77	25.53	34.88	11.48	4.87	9.34	23.88	29.06	57.17	14.59	18.64	97.22	15.86	46.60	64.88	87.52	52.39
.5 x .250 mm	18.07	28.93	36.96	19.78	7.48	9.92	12.69	41.70	66.32	37.32	48.58	90.35	38.93	52.06	74.39	86.02	60.41
.250 x .150	8.68	29.27	39.30	11.45	9.49	10.63	7.01	57.20	71.47	58.54	74.91	81.02	55.93	52.88	77.85	85.69	63.55
.150 x .045	15.17	26.54	40.93	23.07	9.75	10.86	11.12	58.47	73.33	65.54	80.52	75.92	56.44	51.87	78.27	86.24	64.51
.045 mm x 0	38.12	46.66	46.66	33.85	11.62	11.62	42.53	77.22	77.22	46.59	77.19	88.40	65.59	46.59	77.19	88.40	65.59
Cumulative	100.00	34.70		100.03	9.33		100.00	56.41		46.11	64.03	87.60	51.63				

Test 459

	Feed			Concentrate			Tails			Individual by Size				Cum. From Bottom			
	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.
+1mm	2.09	28.19	38.86	0.50	6.31	10.66	2.89	36.85	68.06	28.36	37.00	93.65	30.65	50.88	74.34	86.05	60.39
1 x .5 mm	17.8	30.23	39.08	11.43	5.17	10.68	20.05	47.54	68.99	40.85	55.53	93.01	48.54	51.29	75.20	85.99	61.19
.5 x .250 mm	17.42	33.40	41.05	24.84	9.06	11.39	19.06	68.05	74.57	58.74	80.21	84.07	64.27	53.06	79.75	85.27	65.03
.250 x .150	8.87	34.37	43.18	12.17	10.28	12.31	8.32	73.34	76.72	61.80	84.48	81.52	66.00	52.08	80.36	85.15	65.51
.150 x .045	15.52	30.36	44.63	22.83	11.39	12.79	10.56	68.47	77.28	66.77	84.95	74.95	59.90	50.64	79.75	85.48	65.23
.045 mm x 0	38.3	50.41	50.41	28.23	13.93	13.93	39.12	79.66	79.66	44.50	77.24	87.70	64.94	44.50	77.24	87.70	64.94
Cumulative	100.00	38.86		100.00	10.66		100.00	68.06		50.87	74.33	86.05	60.38				

Test 460

	Feed			Concentrate			Tails			Individual by Size				Cum. From Bottom			
	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.
+1mm	3.74	35.81	40.24	0.55	4.44	11.52	1.66	38.05	68.43	6.66	9.92	99.17	9.10	49.54	73.35	85.81	59.16
1 x .5 mm	22.84	40.13	40.41	10.15	5.45	11.56	13.97	44.79	68.94	11.85	18.71	98.39	17.10	49.73	73.80	85.77	59.57
.5 x .250 mm	17.48	35.43	40.49	20.49	9.64	12.25	16.97	67.04	72.94	55.07	77.07	85.02	62.08	53.47	78.84	83.82	62.66
.250 x .150	12.48	34.85	42.08	14.27	12.17	13.03	9.11	71.24	74.43	61.60	83.05	78.49	61.54	52.70	79.12	83.68	62.79
.150 x .045	14.47	31.93	44.15	25.22	12.49	13.26	11.79	66.31	74.93	63.88	82.12	75.01	57.14	49.91	77.51	85.01	62.52
.045 mm x 0	28.99	50.25	50.25	29.32	13.92	13.92	46.50	77.11	77.11	42.51	73.55	88.23	61.77	42.51	73.55	88.23	61.77
Cumulative	100.00	40.24		100.00	11.52		100.00	68.43		49.53	73.34	85.82	59.16				

Test 461

	Feed			Concentrate			Tails			Individual by Size				Cum. From Bottom			
	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.
+1mm	2.89	27.90	42.80	1.29	4.86	11.78	2.55	34.72	72.56	22.84	30.14	96.02	26.16	48.97	75.52	86.52	62.03
1 x .5 mm	19.39	34.34	43.24	16.09	6.00	11.87	13.74	48.93	73.55	33.99	48.65	94.06	42.72	49.14	76.30	86.51	62.80
.5 x .250 mm	22.07	46.52	45.46	23.13	10.39	13.02	10.62	69.64	77.59	39.02	65.38	91.28	56.67	49.76	79.35	85.75	65.10
.250 x .150	10.58	48.46	45.04	13.01	14.75	14.04	40.02	77.41	78.74	46.20	76.42	85.94	62.36	52.09	81.47	83.76	65.23
.150 x .045	14.97	34.59	44.23	21.94	14.41	13.84	5.52	74.66	80.35	66.51	87.02	72.29	59.32	54.30	83.90	83.01	66.91
.045 mm x 0	30.10	49.03	49.03	24.54	13.33	13.33	27.55	81.49	81.49	47.62	80.98	87.05	68.03	47.62	80.98	87.05	68.03
Cumulative	100.00	42.80		100.00	11.78		100.00	72.56		48.96	75.52	86.52	62.04				

Test 462

	Feed			Concentrate			Tails			Individual by Size				Cum. From Bottom			
	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.
+1mm	2.35	29.50	36.31	0.79	4.62	12.63	2.44	31.30	69.73	6.75	9.13	98.94	8.07	58.53	80.29	79.64	59.93
1 x .5 mm	13.03	21.22	36.47	12.97	6.40	12.69	10.95	33.79	70.69	45.89	54.53	86.16	40.68	59.00	81.09	79.47	60.55
.5 x .250 mm	15.38	20.32	38.82	23.06	10.98	13.64	7.54	53.30	75.36	77.93	87.06	57.89	44.96	59.20	83.57	79.20	62.77
.250 x .150	8.98	24.66	42.93	12.92	12.63	14.61	5.67	67.70	77.46	78.16	90.63	59.97	50.61	54.94	82.20	81.30	63.51
.150 x .045	18.67	29.58	45.65	23.09	12.85	15.12	10.81	68.00	78.22	69.66	86.22	69.74	55.95	51.61	80.60	82.91	63.51
.045 mm x 0	41.59	52.87	52.87	27.17	17.05	17.05	62.59	79.98	79.98	43.08	75.82	86.11	61.93	43.08	75.82	86.11	61.93
Cumulative	100.00	36.31		100.00	12.63		100.00	69.73		58.53	80.29	79.64	59.93				

Test 463

	Feed			Concentrate			Tails			Individual by Size				Cum. From Bottom			
	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.
+1mm	3.28	40.07	45.02	1.19	4.72	12.08	8.37	66.47	75.53	42.75	67.97	94.96	62.94	48.10	76.90	87.09	63.99
1 x .5 mm	23.78	43.45	45.19	19.41	7.03	12.17	41.06	70.57	76.36	42.68	70.17	93.09	63.26	48.57	77.82	86.91	64.74
.5 x .250 mm	17.17	45.50	45.75	19.08	10.50	13.43	16.98	79.09	81.07	48.97	80.42	88.70	69.12	52.21	83.32	84.67	67.99
.250 x .150	10.35	41.99	45.83	11.40	13.31	14.36	5.79	82.36	82.06	58.46	87.37	81.47	68.84	53.52	84.61	83.23	67.84
.150 x .045	14.54	35.07	46.70	21.42	13.14	14.60	6.05	78.74	82.00	66.57	89.05	75.06	64.11	52.37	83.92	83.63	67.54
.045 mm x 0	30.88	52.18	52.18	27.50	15.74	15.74	21.75	82.91	82.91	45.75	80.61	86.20	66.81	45.75	80.61	86.20	66.81
Cumulative	100.00	45.02		100.00	12.08		100.00	75.53		48.09	76.89	87.10	63.99				

Test 464

	Feed			Concentrate			Tails			Individual by Size				Cum. From Bottom			
	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.
+1mm	4.43	45.61	42.44	0.96	3.96	12.60	4.50	52.03	74.06	13.36	23.58	98.84	22.42	51.44	78.11	84.74	62.85
1 x .5 mm	25.23	43.77	42.30	18.13	6.15	12.68	23.23	60.68	75.10	31.01	51.76	95.64	47.40	52.55	79.52	84.25	63.77
.5 x .250 mm	16.95	35.62	41.77	21.11	11.33	14.14	12.86	70.25	79.73	58.77	80.95	81.30	62.25	57.88	85.34	80.40	65.74
.250 x .150	9.2	39.06	43.72	13.21	15.15	15.14	4.85	77.46	81.78	61.63	85.81	76.10	61.90	57.11	86.12	80.23	66.35
.150 x .045	14.53	32.92	44.69	20.80	15.16	15.13	8.72	77.74	82.17	71.62	90.58	67.02	57.60	55.90	85.78	81.07	66.85
.045 mm x 0	29.66	50.46	50.46	25.79	15.11	15.11	45.84	83.01	83.01	47.94	82.15	85.65	67.79	47.94	82.15	85.65	67.79
Cumulative	100.00	42.44		100.00	12.60		100.00	74.06		51.45	78.12	84.73	62.85				

Test 465

	Feed			Concentrate			Tails			Individual by Size				Cum. From Bottom			
	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. %	Ash %	Cum F.Ash	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.	Wt. Yield	C. Rec.	Ash Rej.	Sep Eff.
+1mm	3.39	38.27	40.99	0.54	4.48	11.36	5.82	57.12	70.10	35.81	55.41	95.81	51.22	49.55	74.43	86.27	60.70
1 x .5 mm	20.62	36.40	41.09	10.62	5.88	11.40	23.05	54.38	70.90	37.07	54.86	94.01	48.87	50.10	75.35	86.10	61.45
.5 x .250 mm	18.48	38.32	42.36	16.93	9.80	12.06	11.09	63.09	76.25	46.48	67.97	88.11	56.09	52.80	80.55	84.97	65.52
.250 x .150	9.98	38.06	43.66	12.37	10.90	12.59	5.85	71.79	78.68	55.39	79.69	84.14	63.82	52.99	82.21	84.72	66.93
.150 x .045	14.63	30.44	44.83	26.49	11.00	12.94	9.70	71.97	79.42	68.12	87.15	75.39	62.54	52.03	82.11	84.98	67.09
.045 mm x 0	32.9	51.23	51.23	33.05	14.49	14.49	44.49	81.05	81.05	44.80	78.55	87.33	65.88	44.80	78.55	87.33	65.88
Cumulative	100.00	40.99		100.00	11.36		100.00	70.10		49.56	74.44	86.27	60.71				

APPENDIX B STATISTICAL ANALYSIS OF PLANT TEST DATA

Design - Expert Analysis **First Series (401-416)**

Response: **.25x0 Rec.**

Factor	**Name**	**Units**	**Type**	**-1 Level**	**+1 Level**
A	Feed Rate	kg/min	Numeric	25.00	45.00
B	Frother	ml/min	Numeric	8.00	12.00
C	Diesel	g/ton -.5mm	Numeric	200.00	2000.00

***** WARNING: The Cubic Model is Aliased! *****

Sequential Model Sum of Squares

Source	**Sum of Squares**	**DF**	**Mean Square**	**F Value**	**Prob > F**
Mean	87165.18	1	87165.18		
Linear	1308.93	3	436.31	21.11	< 0.0001
Quadratic	157.68	6	26.28	1.75	0.2578
Cubic	90.36	6	15.06		
Residual	0.000	0			
Total	88722.14	16	5545.13		

"Sequential Model Sum of Squares": Select the highest order polynomial where the additional terms are significant.

Model Summary Statistics

Source	**Root MSE**	**R-Squared**	**Adjusted R-Squared**	**Predicted R-Squared**	**PRESS**
Linear	4.55	0.8407	0.8009	0.6635	523.93
Quadratic	3.88	0.9420	0.8549	0.0804	1431.81
Cubic					

Case(s) with leverage of 1.0000: PRESS statistic not defined

"Model Summary Statistics": Focus on the model minimizing the "PRESS", or equivalently maximizing the "PRED R-SQR".

Response: .25x0 Rec.

Factor	**Name**	**Units**	**Type**	**-1 Level**	**+1 Level**
A	Feed Rate	kg/min	Numeric	25.00	45.00
B	Frother	ml/min	Numeric	8.00	12.00
C	Diesel	g/ton -.5mm	Numeric	200.00	2000.00

ANOVA for Response Surface Quadratic Model

Source	**Sum of Squares**	**DF**	**Mean Square**	**F Value**	**Prob > F**
Model	1466.60	9	162.96	10.82	0.0045
Residual	90.36	6	15.06		
Cor Total	1556.96	15			

Root MSE	3.88	R-Squared	0.9420	
Dep Mean	73.81	Adj R-Squared	0.8549	
C.V.	5.26	Pred R-Squared	0.0804	
PRESS	1431.81	Adeq Precision	9.914	Desire > 4

Factor	**Coefficient Estimate**	**DF**	**Standard Error**	**t for H0 Coeff=0**	**Prob > |t|**	**VIF**
Intercept	77.95	1	3.66			
A-Feed Rate	-12.42	1	3.56	-3.49	0.0130	7.60
B-Frother	4.27	1	2.80	1.52	0.1787	7.22
C-Diesel	5.32	1	5.97	0.89	0.4067	10.53
A^2	-4.38	1	5.59	-0.78	0.4637	5.73
B^2	-1.09	1	1.24	-0.88	0.4140	7.22
C^2	-0.50	1	11.14	-0.045	0.9658	12.02
AB	10.70	1	6.78	1.58	0.1655	15.53
AC	-0.42	1	9.90	-0.042	0.9677	13.43
BC	7.05	1	11.69	0.60	0.5684	23.90

Final Equation in Terms of Coded Factors:

.25x0 Rec. =
+77.95
-12.42 * A
+4.27 * B
+5.32 * C
-4.38 * A^2

-1.09 * B^2
-0.50 * C^2
+10.70 * A * B
-0.42 * A * C
+7.05 * B * C

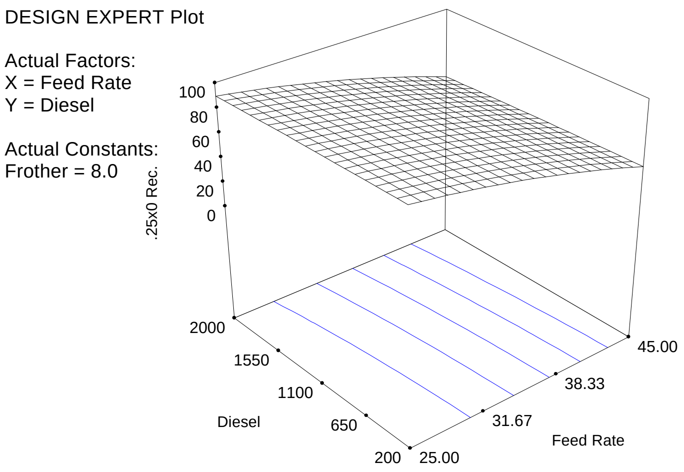
Final Equation in Terms of Actual Factors:

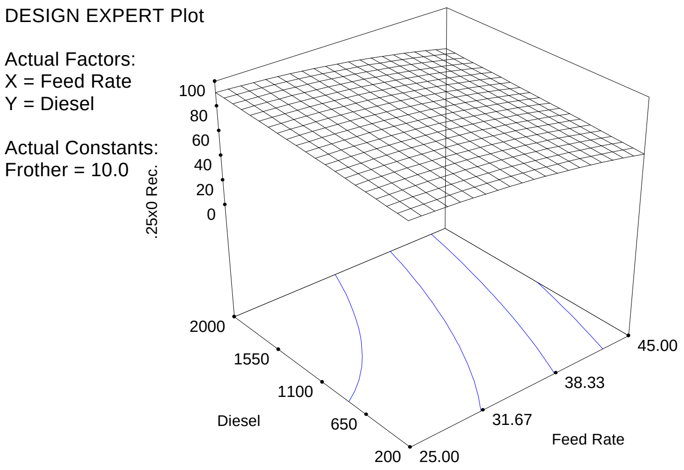
.25x0 Rec. =
+240.61
-3.48 * Feed Rate
-15.47 * Frother
-0.030 * Diesel
-0.044 * Feed Rate^2
-0.27 * Frother^2
-6.154E-07 * Diesel^2
+0.53 * Feed Rate * Frother
-4.646E-05 * Feed Rate * Diesel
+3.917E-03 * Frother * Diesel

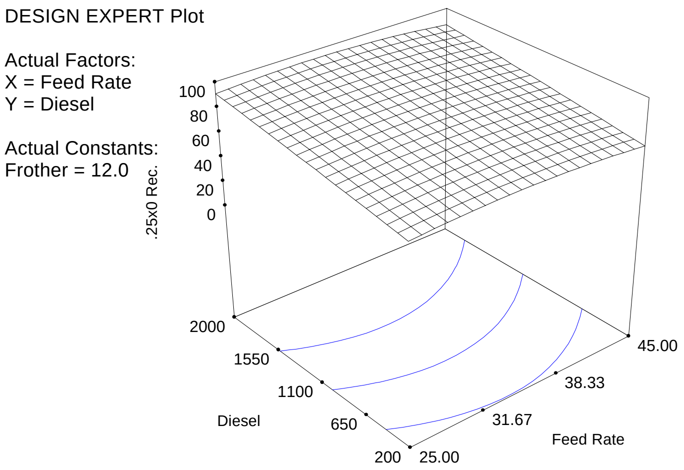
Diagnostics Case Statistics

Standard Order	Actual Value	Predicted Value	Residual	Leverage	Student Residual	Cook's Distance	Outlier t	Run Order
1	74.70	69.14	5.56	0.411	1.867	0.243	2.634	3
2	70.12	69.90	0.22	0.388	0.072	0.000	0.065	9
3	84.17	85.87	-1.70	0.853	-1.139	0.750	-1.174	14
4	87.29	87.14	0.15	0.960	0.199	0.096	0.182	5
5	80.05	80.67	-0.62	0.258	-0.186	0.001	-0.171	1
6	82.91	82.01	0.90	0.904	0.748	0.524	0.717	7
7	79.64	79.74	-0.10	0.718	-0.050	0.001	-0.045	2
8	73.62	72.57	1.05	0.393	0.347	0.008	0.320	10
9	56.91	56.72	0.19	0.798	0.108	0.005	0.099	8
10	80.70	77.65	3.05	0.521	1.134	0.140	1.168	15
11	84.78	85.51	-0.73	0.906	-0.610	0.357	-0.575	16
12	60.83	64.88	-4.05	0.640	-1.742	0.540	-2.262	12
13	63.16	68.33	-5.17	0.301	-1.594	0.109	-1.916	4
14	79.84	79.98	-0.14	0.995	-0.522	5.399	-0.488	6
15	59.75	58.79	0.96	0.501	0.349	0.012	0.321	13
16	62.48	62.04	0.44	0.454	0.155	0.002	0.142	11

3D Response Plots First Series (401-416) 0.25 mm x 0







Design - Expert Analysis **First Series (401-416)**

Response: **.5x.25 Rec.**

Factor	Name	Units	Type	-1 Level	+1 Level
A	Feed Rate	kg/min	Numeric	25.00	45.00
B	Frother	ml/min	Numeric	8.00	12.00
C	Diesel	g/ton -.5mm	Numeric	200.00	2000.00

***** WARNING: The Cubic Model is Aliased! *****

Sequential Model Sum of Squares

Source	Sum of Squares	DF	Mean Square	F Value	Prob > F
Mean	40307.59	1	40307.59		
Linear	7184.16	3	2394.72	19.95	< 0.0001
Quadratic	1171.56	6	195.26	4.36	0.0480
Cubic	268.52	6	44.75		
Residual	0.000	0			
Total	48931.83	16	3058.24		

"Sequential Model Sum of Squares": Select the highest order polynomial where the additional terms are significant.

Model Summary Statistics

Source	Root MSE	R-Squared	Adjusted R-Squared	Predicted R-Squared	PRESS
Linear	10.95	0.8330	0.7913	0.6434	3075.81
Quadratic	6.69	0.9689	0.9222	0.0011	8614.65
Cubic					

Case(s) with leverage of 1.0000: PRESS statistic not defined

"Model Summary Statistics": Focus on the model minimizing the "PRESS", or equivalently maximizing the "PRED R-SQR".

Response: .5x.25 Rec.

Factor	**Name**	**Units**	**Type**	**-1 Level**	**+1 Level**
A	Feed Rate	kg/min	Numeric	25.00	45.00
B	Frother	ml/min	Numeric	8.00	12.00
C	Diesel	g/ton -.5mm	Numeric	200.00	2000.00

ANOVA for Response Surface Quadratic Model

Source	**Sum of Squares**	**DF**	**Mean Square**	**F Value**	**Prob > F**
Model	8355.72	9	928.41	20.75	0.0007
Residual	268.52	6	44.75		
Cor Total	8624.24	15			

Root MSE	6.69	R-Squared	0.9689	
Dep Mean	50.19	Adj R-Squared	0.9222	
C.V.	13.33	Pred R-Squared	0.0011	
PRESS	8614.65	Adeq Precision	14.232	Desire > 4

Factor	**Coefficient Estimate**	**DF**	**Standard Error**	**t for H0 Coeff=0**	**Prob > |t|**	**VIF**
Intercept	72.02	1	6.31			
A-Feed Rate	-7.26	1	6.13	-1.18	0.2814	7.60
B-Frother	11.01	1	4.83	2.28	0.0628	7.22
C-Diesel	42.91	1	10.29	4.17	0.0059	10.53
A^2	-14.87	1	9.64	-1.54	0.1738	5.73
B^2	-0.42	1	2.13	-0.20	0.8491	7.22
C^2	41.28	1	19.21	2.15	0.0752	12.02
AB	34.05	1	11.68	2.91	0.0268	15.53
AC	42.71	1	17.07	2.50	0.0464	13.43
BC	58.59	1	20.15	2.91	0.0271	23.90

Final Equation in Terms of Coded Factors:

.5x.25 Rec. =
+72.02
-7.26 * A
+11.01 * B
+42.91 * C
-14.87 * A^2

-0.42 * B^2
+41.28 * C^2
+34.05 * A * B
+42.71 * A * C
+58.59 * B * C

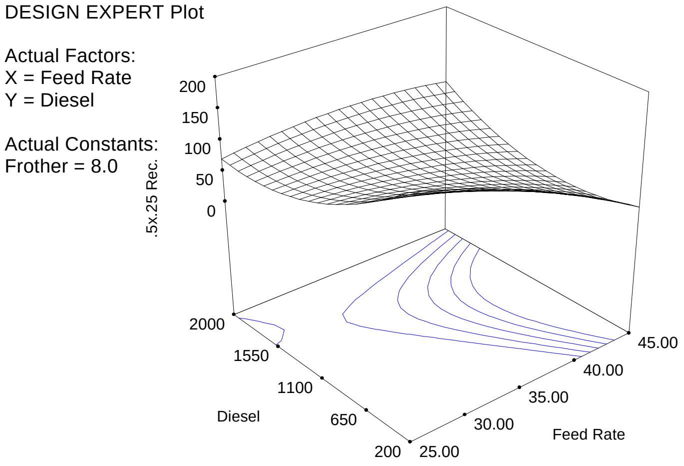
Final Equation in Terms of Actual Factors:

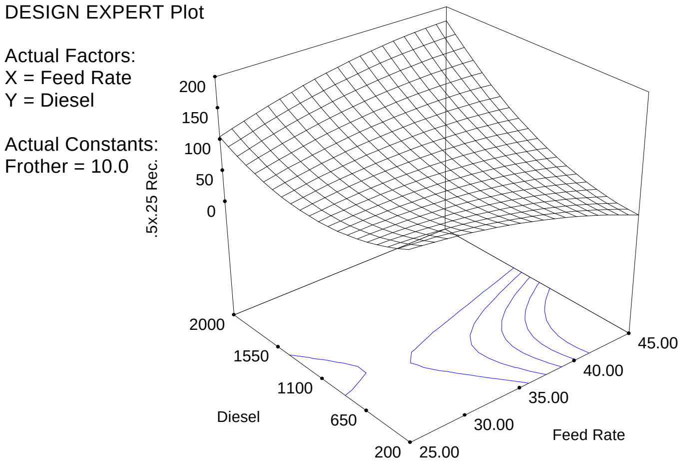
.5x.25 Rec. =
+995.35
-12.56 * Feed Rate
-87.76 * Frother
-0.56 * Diesel
-0.15 * Feed Rate^2
-0.11 * Frother^2
+5.096E-05 * Diesel^2
+1.70 * Feed Rate * Frother
+4.745E-03 * Feed Rate * Diesel
+0.033 * Frother * Diesel

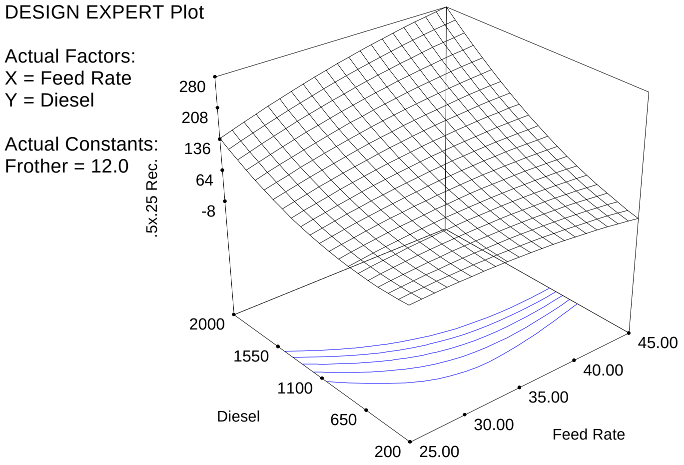
Diagnostics Case Statistics

Standard Order	Actual Value	Predicted Value	Residual	Leverage	Student Residual	Cook's Distance	Outlier t	Run Order
1	45.36	38.02	7.34	0.411	1.429	0.142	1.606	3
2	43.32	36.49	6.83	0.388	1.306	0.108	1.409	9
3	78.22	75.38	2.84	0.853	1.105	0.706	1.130	14
4	83.20	83.45	-0.25	0.960	-0.188	0.085	-0.172	5
5	60.00	65.42	-5.42	0.258	-0.941	0.031	-0.931	1
6	69.99	72.05	-2.06	0.904	-0.992	0.922	-0.990	7
7	61.00	56.41	4.59	0.718	1.292	0.424	1.388	2
8	47.18	49.77	-2.59	0.393	-0.496	0.016	-0.463	10
9	7.80	8.18	-0.38	0.798	-0.127	0.006	-0.116	8
10	65.65	66.96	-1.31	0.521	-0.283	0.009	-0.260	15
11	77.57	77.56	8.544E-03	0.906	0.004	0.000	0.004	16
12	50.55	49.53	1.02	0.640	0.255	0.012	0.234	12
13	17.81	27.45	-9.64	0.301	-1.723	0.128	-2.214	4
14	60.04	60.46	-0.42	0.995	-0.882	15.370	-0.862	6
15	20.89	22.24	-1.35	0.501	-0.287	0.008	-0.263	13
16	14.49	13.70	0.79	0.454	0.159	0.002	0.145	11

3D Response Plots First Series (401-416) 0.5 x 0.25 mm







Design - Expert Analysis **First Series (401-416)**

Response: **.5mmx0 Rec.**

Factor	**Name**	**Units**	**Type**	**-1 Level**	**+1 Level**
A	Feed Rate	kg/min	Numeric	25.00	45.00
B	Frother	ml/min	Numeric	8.00	12.00
C	Diesel	g/ton -.5mm	Numeric	200.00	2000.00

***** WARNING: The Cubic Model is Aliased! *****

Sequential Model Sum of Squares

Source	**Sum of Squares**	**DF**	**Mean Square**	**F Value**	**Prob > F**
Mean	75948.47	1	75948.47		
Linear	1788.61	3	596.20	19.19	< 0.0001
Quadratic	247.57	6	41.26	1.98	0.2135
Cubic	125.16	6	20.86		
Residual	0.000	0			
Total	78109.81	16	4881.86		

"Sequential Model Sum of Squares": Select the highest order polynomial where the additional terms are significant.

Model Summary Statistics

Source	**Root MSE**	**R-Squared**	**Adjusted R-Squared**	**Predicted R-Squared**	**PRESS**
Linear	5.57	0.8275	0.7844	0.6304	798.83
Quadratic	4.57	0.9421	0.8552	-2.9196	8471.57
Cubic					

Case(s) with leverage of 1.0000: PRESS statistic not defined

"Model Summary Statistics": Focus on the model minimizing the "PRESS", or equivalently maximizing the "PRED R-SQR".

Response: .5mmx0 Rec.

Factor	**Name**	**Units**	**Type**	**-1 Level**	**+1 Level**
A	Feed Rate	kg/min	Numeric	25.00	45.00
B	Frother	ml/min	Numeric	8.00	12.00
C	Diesel	g/ton -.5mm	Numeric	200.00	2000.00

ANOVA for Response Surface Quadratic Model

Source	**Sum of Squares**	**DF**	**Mean Square**	**F Value**	**Prob > F**
Model	2036.18	9	226.24	10.85	0.0045
Residual	125.16	6	20.86		
Cor Total	2161.34	15			

Root MSE	4.57	R-Squared	0.9421	
Dep Mean	68.90	Adj R-Squared	0.8552	
C.V.	6.63	Pred R-Squared	-2.9196	
PRESS	8471.57	Adeq Precision	10.276	Desire > 4

Factor	**Coefficient Estimate**	**DF**	**Standard Error**	**t for H0 Coeff=0**	**Prob > |t|**	**VIF**
Intercept	74.59	1	4.31			
A-Feed Rate	-9.65	1	4.19	-2.30	0.0607	7.60
B-Frother	4.87	1	3.30	1.48	0.1901	7.22
C-Diesel	14.04	1	7.02	2.00	0.0926	10.53
A^2	-3.39	1	6.58	-0.52	0.6250	5.73
B^2	-0.78	1	1.46	-0.53	0.6128	7.22
C^2	10.66	1	13.12	0.81	0.4475	12.02
AB	18.11	1	7.98	2.27	0.0637	15.53
AC	10.28	1	11.65	0.88	0.4114	13.43
BC	19.98	1	13.76	1.45	0.1965	23.90

Final Equation in Terms of Coded Factors:

.5mmx0 Rec. =
+74.59
-9.65 * A
+4.87 * B
+14.04 * C
-3.39 * A^2

-0.78 * B^2
+10.66 * C^2
+18.11 * A * B
+10.28 * A * C
+19.98 * B * C

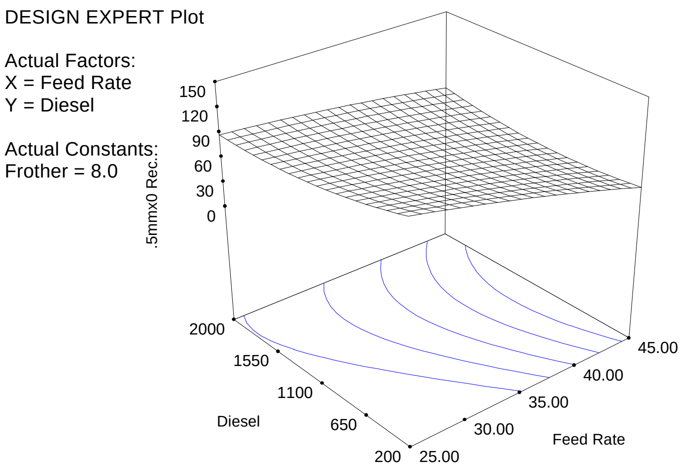
Final Equation in Terms of Actual Factors:

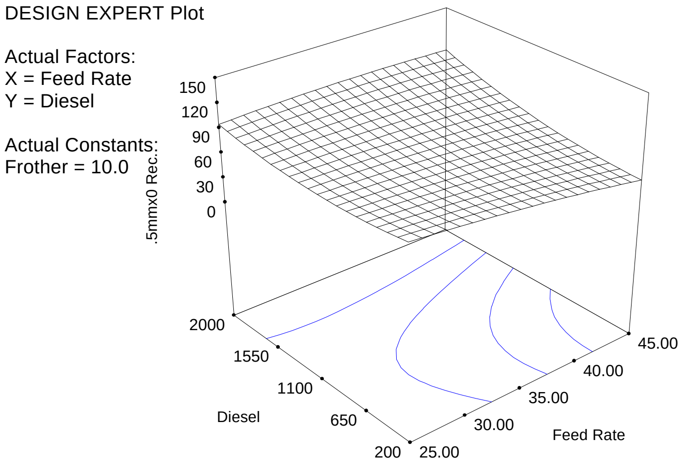
.5mmx0 Rec. =
+504.84
-8.90 * Feed Rate
-37.58 * Frother
-0.16 * Diesel
-0.034 * Feed Rate^2
-0.19 * Frother^2
+1.316E-05 * Diesel^2
+0.91 * Feed Rate * Frother
+1.143E-03 * Feed Rate * Diesel
+0.011 * Frother * Diesel

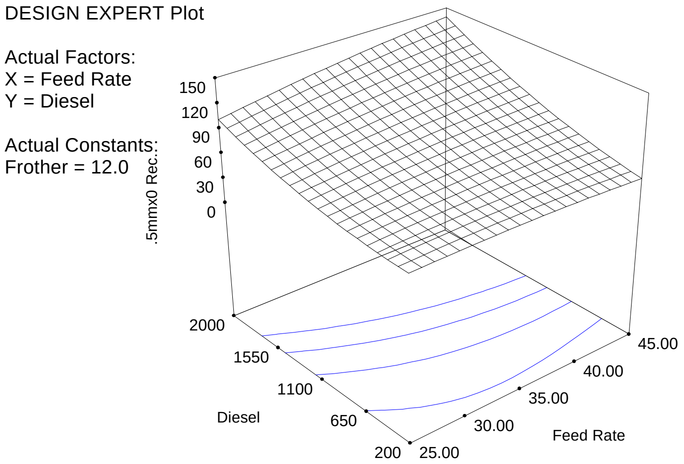
Diagnostics Case Statistics

Standard Order	Actual Value	Predicted Value	Residual	Leverage	Student Residual	Cook's Distance	Outlier t	Run Order
1	70.62	64.29	6.33	0.411	1.807	0.228	2.443	3
2	62.74	61.43	1.31	0.388	0.367	0.009	0.339	9
3	83.16	84.19	-1.03	0.853	-0.585	0.198	-0.550	14
4	86.52	85.99	0.53	0.960	0.582	0.824	0.547	5
5	70.00	74.71	-4.71	0.258	-1.197	0.050	-1.252	1
6	80.22	78.82	1.40	0.904	0.985	0.909	0.982	7
7	72.00	70.93	1.07	0.718	0.440	0.049	0.408	2
8	68.16	66.46	1.70	0.393	0.478	0.015	0.445	10
9	48.70	48.88	-0.18	0.798	-0.090	0.003	-0.082	8
10	78.21	74.44	3.77	0.521	1.192	0.155	1.246	15
11	83.72	85.03	-1.31	0.906	-0.933	0.837	-0.921	16
12	58.67	62.68	-4.01	0.640	-1.462	0.380	-1.663	12
13	55.92	60.49	-4.57	0.301	-1.196	0.062	-1.250	4
14	75.46	75.89	-0.43	0.995	-1.338	35.422	-1.459	6
15	54.14	53.44	0.70	0.501	0.217	0.005	0.199	13
16	54.11	54.69	-0.58	0.454	-0.171	0.002	-0.156	11

3D Response Plots First Series (401-416) 0.5 mm x 0







Design - Expert Analysis **Second Series (451-465)**

Response: **.25mmx0 Rec**

Factor	**Name**	**Units**	**Type**	**-1 Level**	**+1 Level**
A	Feed GPM	gpm	Numeric	40.00	60.00
B	Air Vg	cm/sec	Numeric	1.10	1.30
C	Frother	ml/min	Numeric	5.60	9.30

***** WARNING: The Cubic Model is Aliased! *****

Sequential Model Sum of Squares

Source	**Sum of Squares**	**DF**	**Mean Square**	**F Value**	**Prob > F**
Mean	98575.45	1	98575.45		
Blocks	1.19	1	1.19		
Linear	105.67	3	35.22	2.16	0.1555
Quadratic	148.41	6	24.74	6.90	0.0413
Cubic	14.33	3	4.78	565.13	0.0309
Residual	8.450E-03	1	8.450E-03		
Total	98845.05	15	6589.67		

"Sequential Model Sum of Squares": Select the highest order polynomial where the additional terms are significant.

Lack of Fit Tests

Source	**Sum of Squares**	**DF**	**Mean Square**	**F Value**	**Prob > F**
Linear	162.74	9	18.08	2139.86	0.0168
Quadratic	14.33	3	4.78	565.13	0.0309
Cubic	0.000	0			
Pure Error	8.450E-03	1	8.450E-03		

"Lack of Fit Tests": Want the selected model to have insignificant lack-of-fit.

Model Summary Statistics

Source	**Root MSE**	**R-Squared**	**Adjusted R-Squared**	**Predicted R-Squared**	**PRESS**
Linear	4.03	0.3937	0.2118	-0.7181	461.17
Quadratic	1.89	0.9466	0.8264	-0.0892	292.36
Cubic	0.092	1.0000	0.9996		

Case(s) with leverage of 1.0000: PRESS statistic not defined

"Model Summary Statistics": Focus on the model minimizing the "PRESS", or equivalently maximizing the "PRED R-SQR".

Response: .25mmx0 Rec

Factor	**Name**	**Units**	**Type**	**-1 Level**	**+1 Level**
A	Feed GPM	gpm	Numeric	40.00	60.00
B	Air Vg	cm/sec	Numeric	1.10	1.30
C	Frother	ml/min	Numeric	5.60	9.30

ANOVA for Response Surface Quadratic Model

Source	**Sum of Squares**	**DF**	**Mean Square**	**F Value**	**Prob > F**
Block	1.19	1	1.19		
Model	254.08	9	28.23	7.88	0.0311
Residual	14.33	4	3.58		
Lack of Fit	*14.33*	*3*	*4.78*	*565.13*	*0.0309*
Pure Error	*8.450E-03*	*1*	*8.450E-03*		
Cor Total	269.60	14			

Root MSE	1.89	R-Squared	0.9466
Dep Mean	81.07	Adj R-Squared	0.8264
C.V.	2.34	Pred R-Squared	-0.0892
PRESS	292.36	Adeq Precision	11.201 Desire > 4

Factor	**Coefficient Estimate**	**DF**	**Standard Error**	**t for H0 Coeff=0**	**Prob > |t|**	**VIF**
Intercept	83.26	1	1.75			
Block 1	3.64	1				
Block 2	-3.64					
A-Feed GPM	-3.25	1	0.93	-3.48	0.0254	1.95
B-Air Vg	-1.09	1	0.91	-1.20	0.2963	1.86
C-Frother	-1.09	1	0.73	-1.49	0.2097	1.86
A^2	3.29	1	1.31	2.51	0.0660	1.79
B^2	0.54	1	1.24	0.43	0.6867	1.59
C^2	-2.49	1	1.60	-1.55	0.1957	10.54
AB	-4.22	1	1.61	-2.62	0.0590	2.90
AC	-3.97	1	1.87	-2.12	0.1009	6.58
BC	1.14	1	1.15	0.99	0.3801	2.07

Final Equation in Terms of Coded Factors:

.25mmx0 Rec =
+83.26
-3.25 * A

-1.09 * B
-1.09 * C
+3.29 * A^2
+0.54 * B^2
-2.49 * C^2
-4.22 * A * B
-3.97 * A * C
+1.14 * B * C

Final Equation in Terms of Actual Factors:

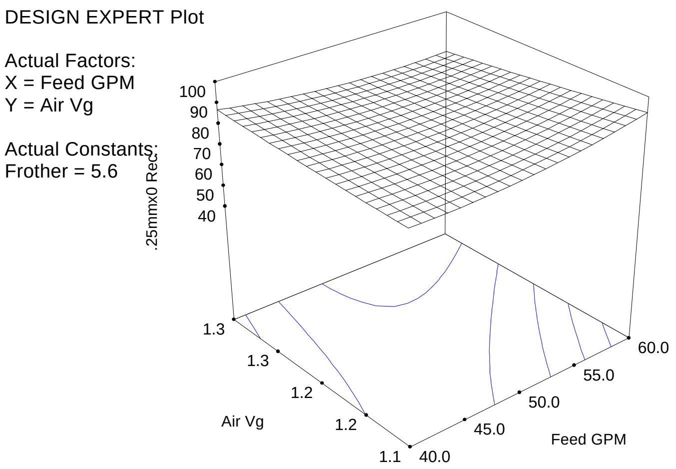
.25mmx0 Rec =
-41.84
+3.05 * Feed GPM
+25.36 * Air Vg
+13.58 * Frother
+0.033 * Feed GPM^2
+53.66 * Air Vg^2
-0.73 * $Frother^2$
-4.22 * Feed GPM * Air Vg
-0.21 * Feed GPM * Frother
+6.15 * Air Vg * Frother

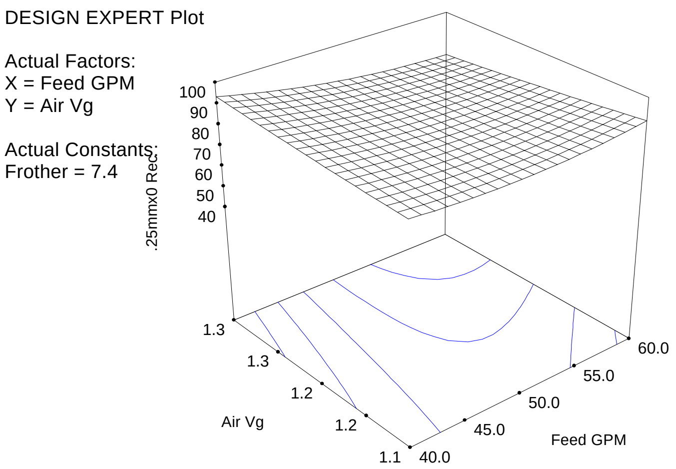
Diagnostics Case Statistics

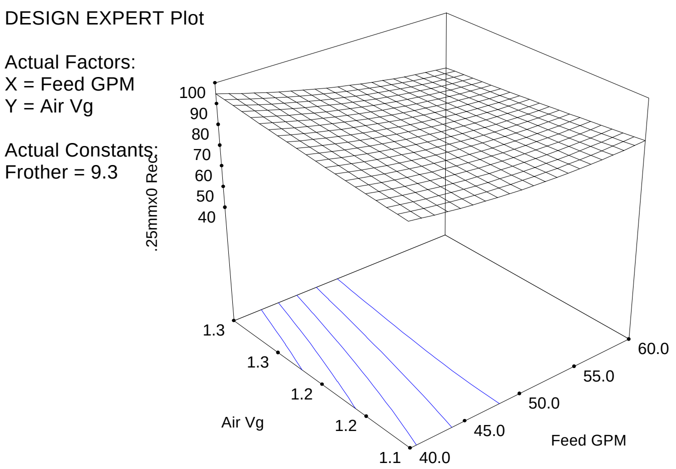
Standard Order	**Actual Value**	**Predicted Value**	**Residual**	**Leverage**	**Student Residual**	**Cook's Distance**	**Outlier t**	**Run Order**
1	81.47	80.68	0.79	0.848	1.074	0.585	1.103	11
2	79.12	78.57	0.55	0.847	0.748	0.281	0.699	10
3	84.61	85.14	-0.53	0.798	-0.625	0.140	-0.569	13
4	81.70	82.47	-0.77	0.890	-1.223	1.097	-1.339	2
5	82.38	82.53	-0.15	0.984	-0.631	2.231	-0.576	1
6	82.20	81.84	0.36	0.872	0.527	0.172	0.473	12
7	86.12	86.23	-0.11	0.655	-0.100	0.002	-0.086	14
8	82.21	79.59	2.62	0.350	1.718	0.144	2.909	15
9	67.93	68.07	-0.14	0.962	-0.384	0.337	-0.339	6
10	80.36	81.47	-1.11	0.659	-1.004	0.177	-1.005	9
11	84.17	83.54	0.63	0.735	0.650	0.106	0.595	5
12	83.58	83.82	-0.24	0.862	-0.337	0.064	-0.296	3
13	84.57	83.90	0.67	0.898	1.102	0.974	1.143	4
14	77.85	79.07	-1.22	0.321	-0.784	0.026	-0.738	8
15	77.72	79.07	-1.35	0.321	-0.867	0.032	-0.833	7

Note: Predicted values include block corrections.

3D Response Plots Second Series (451-465) 0.25 mm x 0







Design - Expert Analysis **Second Series (451-465)**

Response: **.5mmx.25Rec**

Factor	Name	Units	Type	-1 Level	+1 Level
A	Feed GPM	gpm	Numeric	40.00	60.00
B	Air Vg	cm/sec	Numeric	1.10	1.30
C	Frother	ml/min	Numeric	5.60	9.30

***** WARNING: The Cubic Model is Aliased! *****

Sequential Model Sum of Squares

Source	Sum of Squares	DF	Mean Square	F Value	Prob > F
Mean	67648.93	1	67648.93		
Blocks	243.31	1	243.31		
Linear	977.83	3	325.94	1.57	0.2575
Quadratic	1763.09	6	293.85	3.75	0.1109
Cubic	311.34	3	103.78	42.11	0.1127
Residual	2.46	1	2.46		
Total	70946.96	15	4729.80		

"Sequential Model Sum of Squares": Select the highest order polynomial where the additional terms are significant.

Lack of Fit Tests

Source	Sum of Squares	DF	Mean Square	F Value	Prob > F
Linear	2074.42	9	230.49	93.54	0.0801
Quadratic	311.34	3	103.78	42.11	0.1127
Cubic	0.000	0			
Pure Error	2.46	1	2.46		

"Lack of Fit Tests": Want the selected model to have insignificant lack-of-fit.

Model Summary Statistics

Source	Root MSE	R-Squared	Adjusted R-Squared	Predicted R-Squared	PRESS
Linear	14.41	0.3201	0.1161	-0.5218	4648.67
Quadratic	8.86	0.8973	0.6661	-4.8765	17950.97
Cubic	1.57	0.9992	0.9895		

Case(s) with leverage of 1.0000: PRESS statistic not defined

Response: **.5mmx.25Rec**

Factor	**Name**	**Units**	**Type**	**-1 Level**	**+1 Level**
A	Feed GPM	gpm	Numeric	40.00	60.00
B	Air Vg	cm/sec	Numeric	1.10	1.30
C	Frother	ml/min	Numeric	5.60	9.30

ANOVA for Response Surface Quadratic Model

Source	**Sum of Squares**	**DF**	**Mean Square**	**F Value**	**Prob > F**
Block	243.31	1	243.31		
Model	2740.92	9	304.55	3.88	0.1022
Residual	313.80	4	78.45		
Lack of Fit	*311.34*	*3*	*103.78*	*42.11*	*0.1127*
Pure Error	*2.46*	*1*	*2.46*		
Cor Total	3298.03	14			

Root MSE	8.86	R-Squared	0.8973	
Dep Mean	67.16	Adj R-Squared	0.6661	
C.V.	13.19	Pred R-Squared	-4.8765	
PRESS	17950.97	Adeq Precision	7.886	Desire > 4

Factor	**Coefficient Estimate**	**DF**	**Standard Error**	**t for H0 Coeff=0**	**Prob > |t|**	**VIF**
Intercept	61.55	1	8.18			
Block 1	4.78	1				
Block 2	-4.78					
A-Feed GPM	-0.47	1	4.37	-0.11	0.9187	1.95
B-Air Vg	-2.98	1	4.27	-0.70	0.5239	1.86
C-Frother	-7.33	1	3.42	-2.14	0.0990	1.86
A^2	17.86	1	6.13	2.91	0.0435	1.79
B^2	9.02	1	5.79	1.56	0.1940	1.59
C^2	-1.28	1	7.50	-0.17	0.8725	10.54
AB	-16.92	1	7.54	-2.24	0.0884	2.90
AC	-18.31	1	8.74	-2.09	0.1043	6.58
BC	6.51	1	5.40	1.20	0.2947	2.07

Final Equation in Terms of Coded Factors:

.5mmx.25Rec =
+61.55
-0.47 * A
-2.98 * B

-7.33 * C
+17.86 * A^2
+9.02 * B^2
-1.28 * C^2
-16.92 * A * B
-18.31 * A * C
+6.51 * B * C

Final Equation in Terms of Actual Factors:

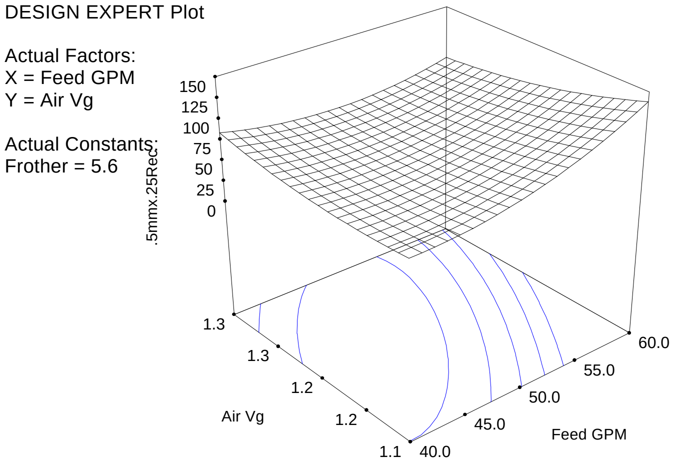
.5mmx.25Rec =
+784.34
+9.77 * Feed GPM
-1610.50 * Air Vg
+8.91 * Frother
+0.18 * Feed GPM^2
+901.88 * Air Vg^2
-0.37 * $Frother^2$
-16.92 * Feed GPM * Air Vg
-0.99 * Feed GPM * Frother
+35.18 * Air Vg * Frother

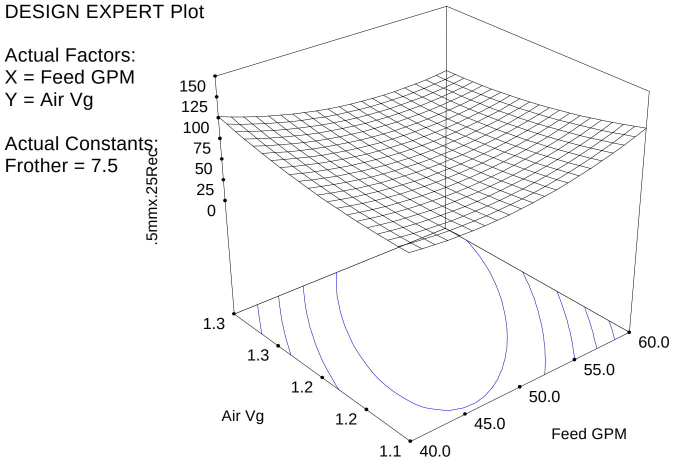
Diagnostics Case Statistics

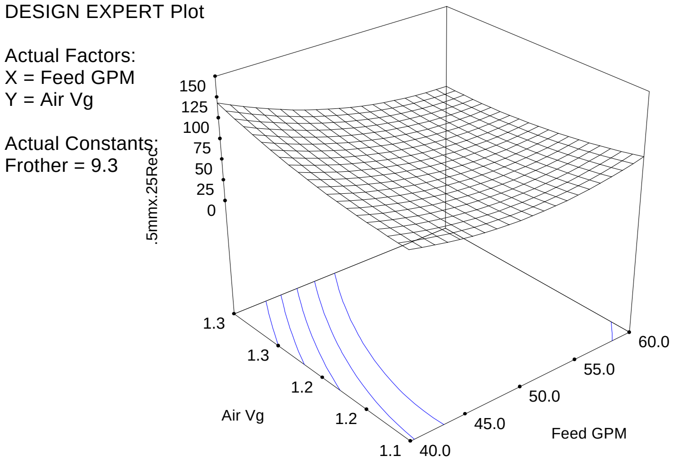
Standard Order	Actual Value	Predicted Value	Residual	Leverage	Student Residual	Cook's Distance	Outlier t	Run Order
1	65.38	65.88	-0.50	0.848	-0.146	0.011	-0.127	11
2	77.07	75.22	1.85	0.847	0.532	0.142	0.478	10
3	80.42	83.57	-3.15	0.798	-0.791	0.225	-0.746	13
4	73.59	74.39	-0.80	0.890	-0.270	0.054	-0.236	2
5	61.99	63.89	-1.90	0.984	-1.699	16.155	-2.787	1
6	87.06	89.60	-2.54	0.872	-0.802	0.398	-0.758	12
7	80.95	75.40	5.55	0.655	1.067	0.196	1.092	14
8	67.97	56.57	11.40	0.350	1.596	0.125	2.295	15
9	31.27	29.78	1.49	0.962	0.859	1.685	0.823	6
10	80.21	78.82	1.39	0.659	0.269	0.013	0.235	9
11	75.88	71.81	4.07	0.735	0.891	0.200	0.862	5
12	60.47	63.20	-2.73	0.862	-0.829	0.389	-0.789	3
13	70.14	70.26	-0.12	0.898	-0.044	0.002	-0.038	4
14	48.58	54.47	-5.89	0.321	-0.807	0.028	-0.764	8
15	46.36	54.47	-8.11	0.321	-1.111	0.053	-1.157	7

Note: Predicted values include block corrections.

3D Response Plots Second Series (451-465) 0.5 x .25 mm







Design - Expert Analysis Second Series (451-465)

Response: .5mmx0 Rec

Factor	Name	Units	Type	-1 Level	+1 Level
A	Feed GPM	gpm	Numeric	40.00	60.00
B	Air Vg	cm/sec	Numeric	1.10	1.30
C	Frother	ml/min	Numeric	5.60	9.30

***** WARNING: The Cubic Model is Aliased! *****

Sequential Model Sum of Squares

Source	Sum of Squares	DF	Mean Square	F Value	Prob > F
Mean	93991.42	1	93991.42		
Blocks	12.48	1	12.48		
Linear	153.26	3	51.09	1.77	0.2165
Quadratic	258.47	6	43.08	5.67	0.0574
Cubic	30.31	3	10.10	147.62	0.0604
Residual	0.068	1	0.068		
Total	94446.01	15	6296.40		

"Sequential Model Sum of Squares": Select the highest order polynomial where the additional terms are significant.

Lack of Fit Tests

Source	Sum of Squares	DF	Mean Square	F Value	Prob > F
Linear	288.79	9	32.09	468.78	0.0358
Quadratic	30.31	3	10.10	147.62	0.0604
Cubic	0.000	0			
Pure Error	0.068	1	0.068		

Model Summary Statistics

Source	Root MSE	R-Squared	Adjusted R-Squared	Predicted R-Squared	PRESS
Linear	5.37	0.3466	0.1506	-0.7578	777.15
Quadratic	2.76	0.9313	0.7767	-0.6719	739.16
Cubic	0.26	0.9998	0.9980		

Case(s) with leverage of 1.0000: PRESS statistic not defined

"Model Summary Statistics": Focus on the model minimizing the "PRESS", or equivalently maximizing the "PRED R-SQR".

Response: .5mmx0 Rec

Factor	**Name**	**Units**	**Type**	**-1 Level**	**+1 Level**
A	Feed GPM	gpm	Numeric	40.00	60.00
B	Air Vg	cm/sec	Numeric	1.10	1.30
C	Frother	ml/min	Numeric	5.60	9.30

ANOVA for Response Surface Quadratic Model

Source	**Sum of Squares**	**DF**	**Mean Square**	**F Value**	**Prob > F**
Block	12.48	1	12.48		
Model	411.73	9	45.75	6.02	0.0497
Residual	30.38	4	7.60		
Lack of Fit	*30.31*	*3*	*10.10*	*147.62*	*0.0604*
Pure Error	*0.068*	*1*	*0.068*		
Cor Total	454.59	14			

Root MSE	2.76	R-Squared	0.9313	
Dep Mean	79.16	Adj R-Squared	0.7767	
C.V.	3.48	Pred R-Squared	-0.6719	
PRESS	739.16	Adeq Precision	9.759	Desire > 4

Factor	**Coefficient Estimate**	**DF**	**Standard Error**	**t for H0 Coeff=0**	**Prob > |t|**	**VIF**
Intercept	81.16	1	2.55			
Block 1	4.22	1				
Block 2	-4.22					
A-Feed GPM	-2.90	1	1.36	-2.13	0.1001	1.95
B-Air Vg	-1.70	1	1.33	-1.28	0.2697	1.86
C-Frother	-2.12	1	1.07	-1.99	0.1177	1.86
A^2	5.19	1	1.91	2.72	0.0529	1.79
B^2	0.98	1	1.80	0.54	0.6163	1.59
C^2	-2.71	1	2.33	-1.16	0.3101	10.54
AB	-6.39	1	2.35	-2.72	0.0528	2.90
AC	-6.00	1	2.72	-2.21	0.0919	6.58
BC	2.09	1	1.68	1.24	0.2818	2.07

Final Equation in Terms of Coded Factors:

.5mmx0 Rec =
+81.16
-2.90 * A
-1.70 * B
-2.12 * C
+5.19 * A^2

+0.98 * B^2
-2.71 * C^2
-6.39 * A * B
-6.00 * A * C
+2.09 * B * C

Final Equation in Terms of Actual Factors:

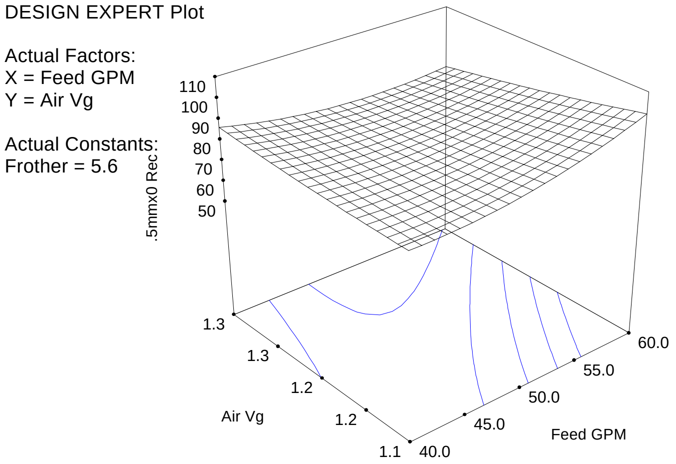
.5mmx0 Rec =
-52.50
+4.61 * Feed GPM
-15.90 * Air Vg
+13.33 * Frother
+0.052 * Feed GPM^2
+97.66 * Air Vg^2
-0.79 * $Frother^2$
-6.39 * Feed GPM * Air Vg
-0.32 * Feed GPM * Frother
+11.29 * Air Vg * Frother

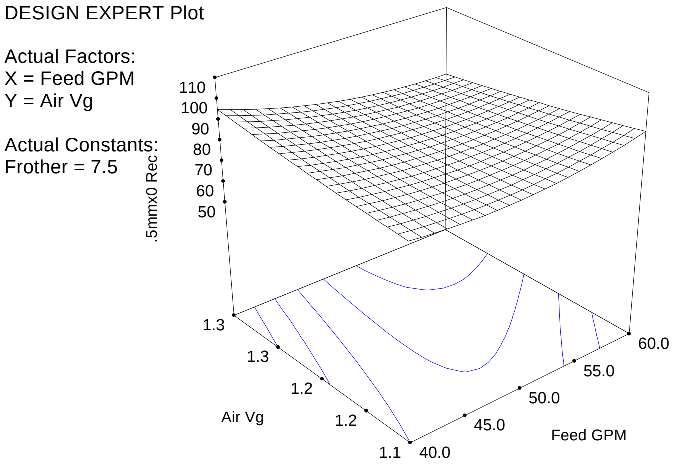
Diagnostics Case Statistics

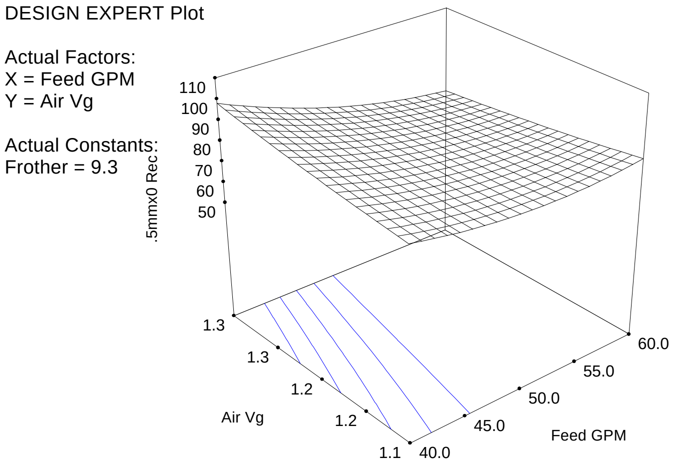
Standard Order	**Actual Value**	**Predicted Value**	**Residual**	**Leverage**	**Student Residual**	**Cook's Distance**	**Outlier t**	**Run Order**
1	79.35	78.25	1.10	0.848	1.028	0.536	1.038	11
2	78.84	77.85	0.99	0.847	0.920	0.425	0.897	10
3	83.32	84.36	-1.04	0.798	-0.837	0.252	-0.798	13
4	79.88	81.03	-1.15	0.890	-1.253	1.153	-1.393	2
5	80.11	80.38	-0.27	0.984	-0.779	3.395	-0.732	1
6	83.57	83.24	0.33	0.872	0.334	0.069	0.294	12
7	85.34	85.14	0.20	0.655	0.125	0.003	0.109	14
8	80.55	76.89	3.66	0.350	1.647	0.133	2.515	15
9	61.93	62.11	-0.18	0.962	-0.325	0.241	-0.285	6
10	79.75	81.26	-1.51	0.659	-0.937	0.154	-0.918	9
11	82.91	81.68	1.23	0.735	0.867	0.189	0.834	5
12	80.55	81.14	-0.59	0.862	-0.575	0.187	-0.520	3
13	82.87	81.92	0.95	0.898	1.082	0.939	1.114	4
14	74.39	76.08	-1.69	0.321	-0.743	0.024	-0.693	8
15	74.02	76.08	-2.06	0.321	-0.906	0.035	-0.880	7

Note: Predicted values include block corrections.

3D Response Plots Second Series (451-465) 0.5 mm x 0







Design - Expert Analysis Second Series - Revisited (451-465)

Response: .25mmx0 Rec

Factor	Name	Units	Type	-1 Level	+1 Level
A	Feed GPM	gpm	Numeric	40.00	60.00
B	Diesel	g/T (-.5mm)	Numeric	700.00	1400.00
C	Frother	ml/min	Numeric	5.60	9.30

***** WARNING: The Cubic Model is Aliased! *****

Sequential Model Sum of Squares

Source	Sum of Squares	DF	Mean Square	F Value	Prob > F
Mean	98575.45	1	98575.45		
Blocks	1.19	1	1.19		
Linear	98.39	3	32.80	1.93	0.1889
Quadratic	134.23	6	22.37	2.50	0.1971
Cubic	35.79	4	8.95		
Residual	0.000	0			
Total	98845.05	15	6589.67		

"Sequential Model Sum of Squares": Select the highest order polynomial where the additional terms are significant.

Model Summary Statistics

Source	Root MSE	R-Squared	Adjusted R-Squared	Predicted R-Squared	PRESS
Linear	4.12	0.3666	0.1765	-0.7193	461.48
Quadratic	2.99	0.8667	0.5667	-5.7482	1811.32
Cubic					

Case(s) with leverage of 1.0000: PRESS statistic not defined

"Model Summary Statistics": Focus on the model minimizing the "PRESS", or equivalently maximizing the "PRED R-SQR".

Response: .25mmx0 Rec **Second Series - Revisited (451-465)**

Factor	Name	Units	Type	-1 Level	+1 Level
A	Feed GPM	gpm	Numeric	40.00	60.00
B	Diesel	g/T (-.5mm)	Numeric	700.00	1400.00
C	Frother	ml/min	Numeric	5.60	9.30

ANOVA for Response Surface Quadratic Model

Source	Sum of Squares	DF	Mean Square	F Value	Prob > F
Block	1.19	1	1.19		
Model	232.63	9	25.85	2.89	0.1597
Residual	35.79	4	8.95		
Cor Total	269.60	14			

Root MSE	2.99	R-Squared	0.8667	
Dep Mean	81.07	Adj R-Squared	0.5667	
C.V.	3.69	Pred R-Squared	-5.7482	
PRESS	1811.32	Adeq Precision	6.463	Desire > 4

Factor	Coefficient Estimate	DF	Standard Error	t for H0 Coeff=0	Prob > |t|	VIF
Intercept	86.29	1	3.38			
Block 1	4.28	1				
Block 2	-4.28					
A-Feed GPM	-2.19	1	1.68	-1.30	0.2643	2.54
B-Diesel	-0.63	1	2.67	-0.24	0.8237	7.69
C-Frother	-1.90	1	1.75	-1.09	0.3379	4.25
A^2	0.66	1	2.70	0.25	0.8180	3.04
B^2	-1.84	1	2.78	-0.66	0.5442	1.88
C^2	-5.29	1	3.67	-1.44	0.2236	22.20
AB	-3.31	1	2.85	-1.16	0.3094	3.86
AC	1.59	1	4.94	0.32	0.7639	18.42
BC	2.87	1	2.50	1.15	0.3150	6.05

Final Equation in Terms of Coded Factors:

.25mmx0 Rec =
+86.29
-2.19 * A
-0.63 * B
-1.90 * C
+0.66 * A^2

-1.84 * B^2
-5.29 * C^2
-3.31 * A * B
+1.59 * A * C
+2.87 * B * C

Final Equation in Terms of Actual Factors:

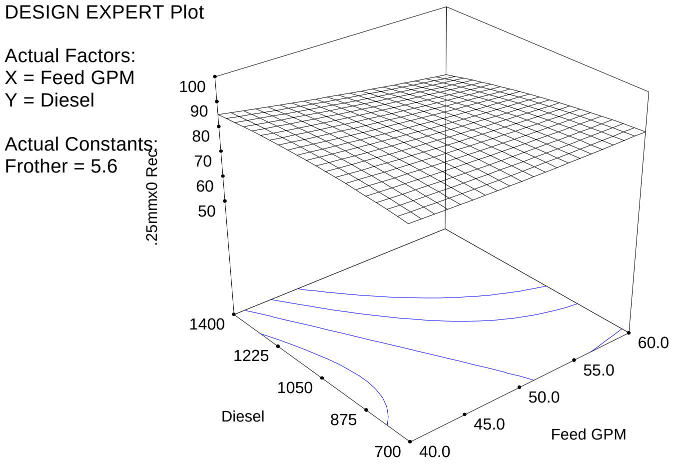
.25mmx0 Rec =
+37.98
-0.53 * Feed GPM
+0.044 * Diesel
+13.05 * Frother
+6.636E-03 * Feed GPM^2
-1.501E-05 * Diesel^2
-1.54 * Frother^2
-9.466E-04 * Feed GPM * Diesel
+0.086 * Feed GPM * Frother
+4.426E-03 * Diesel * Frother

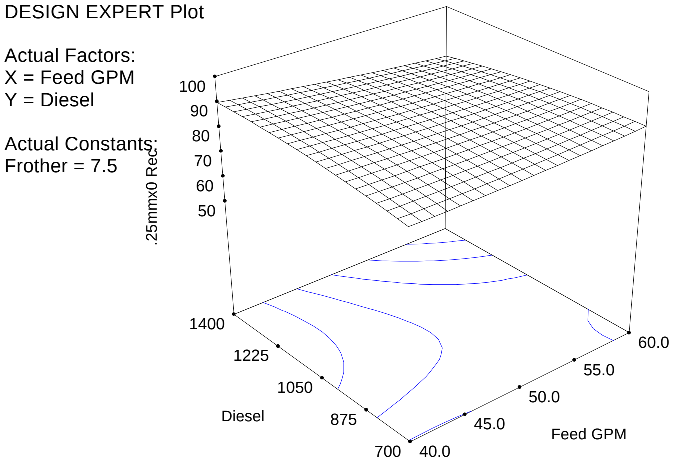
Diagnostics Case Statistics

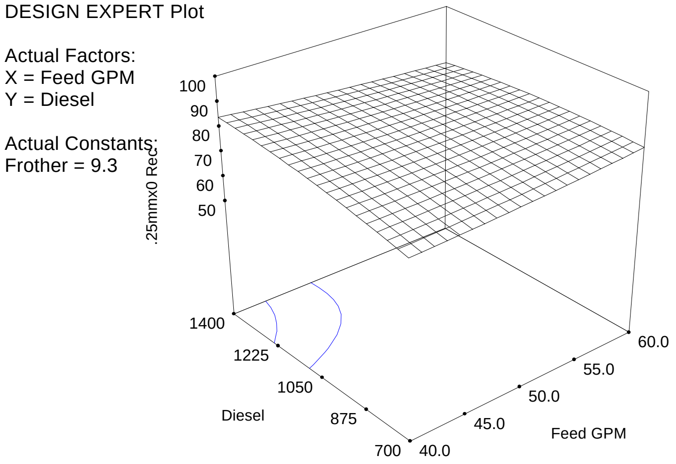
Standard Order	Actual Value	Predicted Value	Residual	Leverage	Student Residual	Cook's Distance	Outlier t	Run Order
1	81.47	84.35	-2.88	0.500	-1.362	0.168	-1.611	11
2	79.12	76.55	2.57	0.752	1.725	0.822	2.951	10
3	84.61	82.46	2.15	0.787	1.558	0.813	2.152	13
4	81.70	83.39	-1.69	0.877	-1.612	1.685	-2.358	2
5	82.38	82.45	-0.065	0.936	-0.086	0.010	-0.074	1
6	82.20	82.24	-0.036	0.948	-0.053	0.005	-0.046	12
7	86.12	85.33	0.79	0.792	0.581	0.117	0.526	14
8	82.21	80.88	1.33	0.283	0.526	0.010	0.472	15
9	67.93	68.77	-0.84	0.953	-1.299	3.118	-1.480	6
10	80.36	81.37	-1.01	0.649	-0.570	0.055	-0.515	9
11	84.17	82.25	1.92	0.809	1.471	0.835	1.881	5
12	83.58	83.93	-0.35	0.987	-1.030	7.595	-1.041	3
13	84.57	83.55	1.02	0.919	1.200	1.492	1.299	4
14	77.85	79.58	-1.73	0.301	-0.692	0.019	-0.638	8
15	77.72	78.91	-1.19	0.506	-0.566	0.030	-0.511	7

Note: Predicted values include block corrections.

3D Response Plots Second Series - Revisited (451-465) 0.25 mm x 0







Response: **.5mmx.25Rec - Second Series - Revisited (451-465)**

Factor	Name	Units	Type	-1 Level	+1 Level
A	Feed GPM	gpm	Numeric	40.00	60.00
B	Diesel	g/T (-.5mm)	Numeric	700.00	1400.00
C	Frother	ml/min	Numeric	5.60	9.30

***** WARNING: The Cubic Model is Aliased! *****

Sequential Model Sum of Squares

Source	Sum of Squares	DF	Mean Square	F Value	Prob > F
Mean	67648.93	1	67648.93		
Blocks	243.31	1	243.31		
Linear	1123.12	3	374.37	1.94	0.1875
Quadratic	1205.86	6	200.98	1.11	0.4825
Cubic	725.74	4	181.44		
Residual	0.000	0			
Total	70946.96	15	4729.80		

"Sequential Model Sum of Squares": Select the highest order polynomial where the additional terms are significant.

Model Summary Statistics

Source	Root MSE	R-Squared	Adjusted R-Squared	Predicted R-Squared	PRESS
Linear	13.90	0.3677	0.1780	-0.5432	4714.02
Quadratic	13.47	0.7624	0.2279	-11.3740	37799.02
Cubic					

Case(s) with leverage of 1.0000: PRESS statistic not defined

"Model Summary Statistics": Focus on the model minimizing the "PRESS", or equivalently maximizing the "PRED R-SQR".

Response: .5mmx.25Rec **Second Series - Revisited (451-465)**

Factor	**Name**	**Units**	**Type**	**-1 Level**	**+1 Level**
A	Feed GPM	gpm	Numeric	40.00	60.00
B	Diesel	g/T (-.5mm)	Numeric	700.00	1400.00
C	Frother	ml/min	Numeric	5.60	9.30

ANOVA for Response Surface Quadratic Model

Source	**Sum of Squares**	**DF**	**Mean Square**	**F Value**	**Prob > F**
Block	243.31	1	243.31		
Model	2328.98	9	258.78	1.43	0.3895
Residual	725.74	4	181.44		
Cor Total	3298.03	14			

Root MSE	13.47	R-Squared	0.7624	
Dep Mean	67.16	Adj R-Squared	0.2279	
C.V.	20.06	Pred R-Squared	-11.3740	
PRESS	37799.02	Adeq Precision	4.644	Desire > 4

Factor	**Coefficient Estimate**	**DF**	**Standard Error**	**t for H_0 Coeff=0**	**Prob > |t|**	**VIF**
Intercept	83.31	1	15.24			
Block 1	4.98	1				
Block 2	-4.98					
A-Feed GPM	5.94	1	7.59	0.78	0.4772	2.54
B-Diesel	4.77	1	12.01	0.40	0.7112	7.69
C-Frother	-16.63	1	7.88	-2.11	0.1023	4.25
A^2	0.24	1	12.16	0.020	0.9850	3.04
B^2	-11.09	1	12.51	-0.89	0.4255	1.88
C^2	-15.67	1	16.55	-0.95	0.3973	22.20
AB	-9.03	1	12.83	-0.70	0.5204	3.86
AC	7.14	1	22.24	0.32	0.7642	18.42
BC	16.78	1	11.24	1.49	0.2099	6.05

Final Equation in Terms of Coded Factors:

.5mmx.25Rec =
+83.31
+5.94 * A
+4.77 * B
-16.63 * C
+0.24 * A^2

-11.09 * B^2
-15.67 * C^2
-9.03 * A * B
+7.14 * A * C
+16.78 * B * C

Final Equation in Terms of Actual Factors:

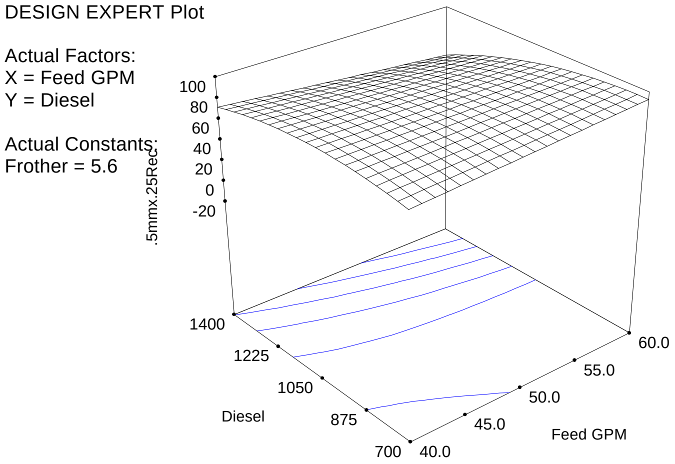
.5mmx.25Rec =
-30.42
+0.18 * Feed GPM
+0.14 * Diesel
+12.71 * Frother
+2.429E-03 * Feed GPM^2
-9.049E-05 * $Diesel^2$
-4.58 * $Frother^2$
-2.579E-03 * Feed GPM * Diesel
+0.39 * Feed GPM * Frother
+0.026 * Diesel * Frother

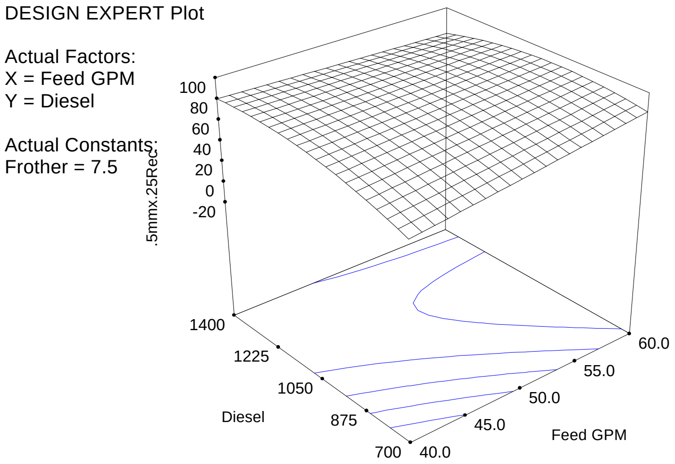
Diagnostics Case Statistics

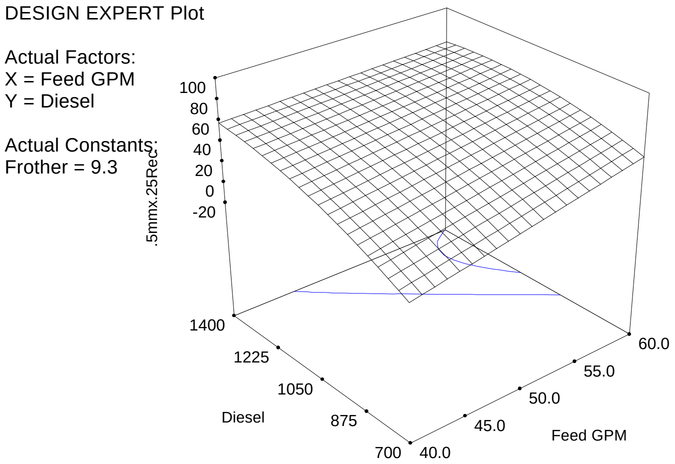
Standard Order	Actual Value	Predicted Value	Residual	Leverage	Student Residual	Cook's Distance	Outlier t	Run Order
1	65.38	81.23	-15.85	0.500	-1.664	0.252	-2.598	11
2	77.07	65.31	11.76	0.752	1.755	0.850	3.167	10
3	80.42	72.02	8.40	0.787	1.350	0.611	1.585	13
4	73.59	79.77	-6.18	0.877	-1.309	1.112	-1.500	2
5	61.99	60.68	1.31	0.936	0.384	0.195	0.339	1
6	87.06	88.74	-1.68	0.948	-0.548	0.502	-0.494	12
7	80.95	74.81	6.14	0.792	0.998	0.344	0.998	14
8	67.97	63.38	4.59	0.283	0.402	0.006	0.356	15
9	31.27	35.17	-3.90	0.953	-1.337	3.305	-1.558	6
10	80.21	80.75	-0.54	0.649	-0.067	0.001	-0.058	9
11	75.88	69.39	6.49	0.809	1.104	0.470	1.146	5
12	60.47	62.12	-1.65	0.987	-1.096	8.605	-1.135	3
13	70.14	66.20	3.94	0.919	1.028	1.095	1.039	4
14	48.58	56.70	-8.12	0.301	-0.721	0.020	-0.669	8
15	46.36	51.06	-4.70	0.506	-0.497	0.023	-0.444	7

Note: Predicted values include block corrections.

3D Response Plots Second Series - Revisited (451-465) 0.5 x 0.25 mm







Response: .5mmx0 Rec Second Series - Revisited (451-465)

Factor	Name	Units	Type	-1 Level	+1 Level
A	Feed GPM	gpm	Numeric	40.00	60.00
B	Diesel	g/T (-.5mm)	Numeric	700.00	1400.00
C	Frother	ml/min	Numeric	5.60	9.30

***** WARNING: The Cubic Model is Aliased! *****

Sequential Model Sum of Squares

Source	Sum of Squares	DF	Mean Square	F Value	Prob > F
Mean	93991.42	1	93991.42		
Blocks	12.48	1	12.48		
Linear	148.71	3	49.57	1.69	0.2318
Quadratic	218.50	6	36.42	1.94	0.2707
Cubic	74.90	4	18.72		
Residual	0.000	0			
Total	94446.01	15	6296.40		

"Sequential Model Sum of Squares": Select the highest order polynomial where the additional terms are significant.

Model Summary Statistics

Source	Root MSE	R-Squared	Adjusted R-Squared	Predicted R-Squared	PRESS
Linear	5.42	0.3364	0.1373	-0.7723	783.57
Quadratic	4.33	0.8306	0.4494	-7.8506	3912.96
Cubic					

Case(s) with leverage of 1.0000: PRESS statistic not defined

"Model Summary Statistics": Focus on the model minimizing the "PRESS", or equivalently maximizing the "PRED R-SQR".

Response: .5mmx0 Rec **Second Series - Revisited (451-465)**

Factor	Name	Units	Type	-1 Level	+1 Level
A	Feed GPM	gpm	Numeric	40.00	60.00
B	Diesel	g/T (-.5mm)	Numeric	700.00	1400.00
C	Frother	ml/min	Numeric	5.60	9.30

ANOVA for Response Surface Quadratic Model

Source	Sum of Squares	DF	Mean Square	F Value	Prob > F
Block	12.48	1	12.48		
Model	367.21	9	40.80	2.18	0.2356
Residual	74.90	4	18.72		
Cor Total	454.59	14			

Root MSE	4.33	R-Squared	0.8306	
Dep Mean	79.16	Adj R-Squared	0.4494	
C.V.	5.47	Pred R-Squared	-7.8506	
PRESS	3912.96	Adeq Precision	5.627	Desire > 4

Factor	Coefficient Estimate	DF	Standard Error	t for H0 Coeff=0	Prob > |t|	VIF
Intercept	85.79	1	4.90			
Block 1	4.57	1				
Block 2	-4.57					
A-Feed GPM	-1.18	1	2.44	-0.48	0.6532	2.54
B-Diesel	-0.17	1	3.86	-0.044	0.9673	7.69
C-Frother	-3.54	1	2.53	-1.40	0.2345	4.25
A^2	0.75	1	3.91	0.19	0.8581	3.04
B^2	-2.66	1	4.02	-0.66	0.5436	1.88
C^2	-6.94	1	5.32	-1.31	0.2617	22.20
AB	-5.03	1	4.12	-1.22	0.2895	3.86
AC	2.47	1	7.15	0.35	0.7472	18.42
BC	4.51	1	3.61	1.25	0.2797	6.05

Final Equation in Terms of Coded Factors:

.5mmx0 Rec =
+85.79
-1.18 * A
-0.17 * B
-3.54 * C
+0.75 * A^2

-2.66 * B^2
-6.94 * C^2
-5.03 * A * B
+2.47 * A * C
+4.51 * B * C

Final Equation in Terms of Actual Factors:

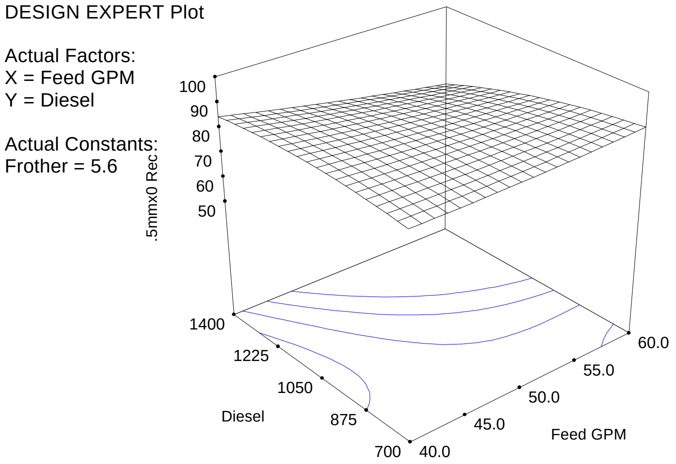
.5mmx0 Rec =
+17.38
-0.35 * Feed GPM
+0.065 * Diesel
+14.31 * Frother
+7.450E-03 * Feed GPM^2
-2.175E-05 * $Diesel^2$
-2.03 * $Frother^2$
-1.436E-03 * Feed GPM * Diesel
+0.13 * Feed GPM * Frother
+6.970E-03 * Diesel * Frother

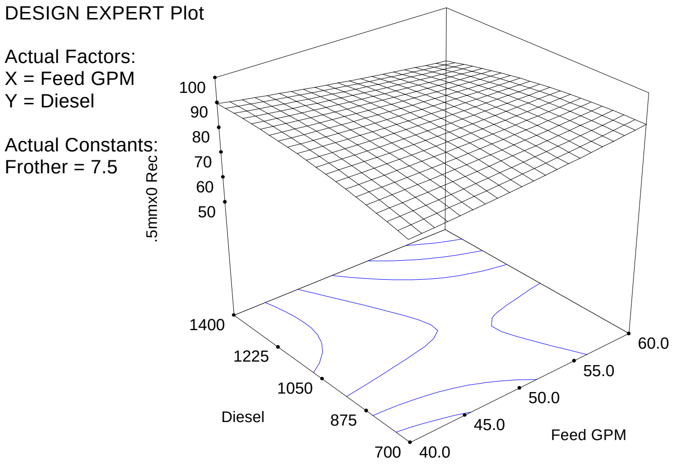
Diagnostics Case Statistics

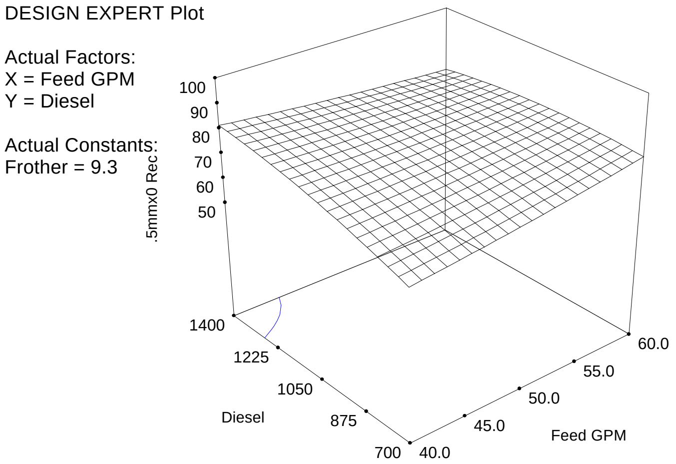
Standard Order	**Actual Value**	**Predicted Value**	**Residual**	**Leverage**	**Student Residual**	**Cook's Distance**	**Outlier t**	**Run Order**
1	79.35	83.75	-4.40	0.500	-1.439	0.188	-1.795	11
2	78.84	75.05	3.79	0.752	1.759	0.854	3.201	10
3	83.32	80.23	3.09	0.787	1.546	0.801	2.112	13
4	79.88	82.26	-2.38	0.877	-1.569	1.596	-2.190	2
5	80.11	80.09	0.016	0.936	0.014	0.000	0.012	1
6	83.57	83.72	-0.15	0.948	-0.149	0.037	-0.130	12
7	85.34	84.04	1.30	0.792	0.656	0.149	0.602	14
8	80.55	78.85	1.70	0.283	0.463	0.008	0.412	15
9	61.93	63.19	-1.26	0.953	-1.346	3.345	-1.575	6
10	79.75	80.95	-1.20	0.649	-0.467	0.037	-0.416	9
11	82.91	80.25	2.66	0.809	1.407	0.763	1.714	5
12	80.55	81.06	-0.51	0.987	-1.059	8.032	-1.081	3
13	82.87	81.39	1.48	0.919	1.204	1.501	1.306	4
14	74.39	76.83	-2.44	0.301	-0.675	0.018	-0.621	8
15	74.02	75.70	-1.68	0.506	-0.553	0.029	-0.498	7

Note: Predicted values include block corrections.

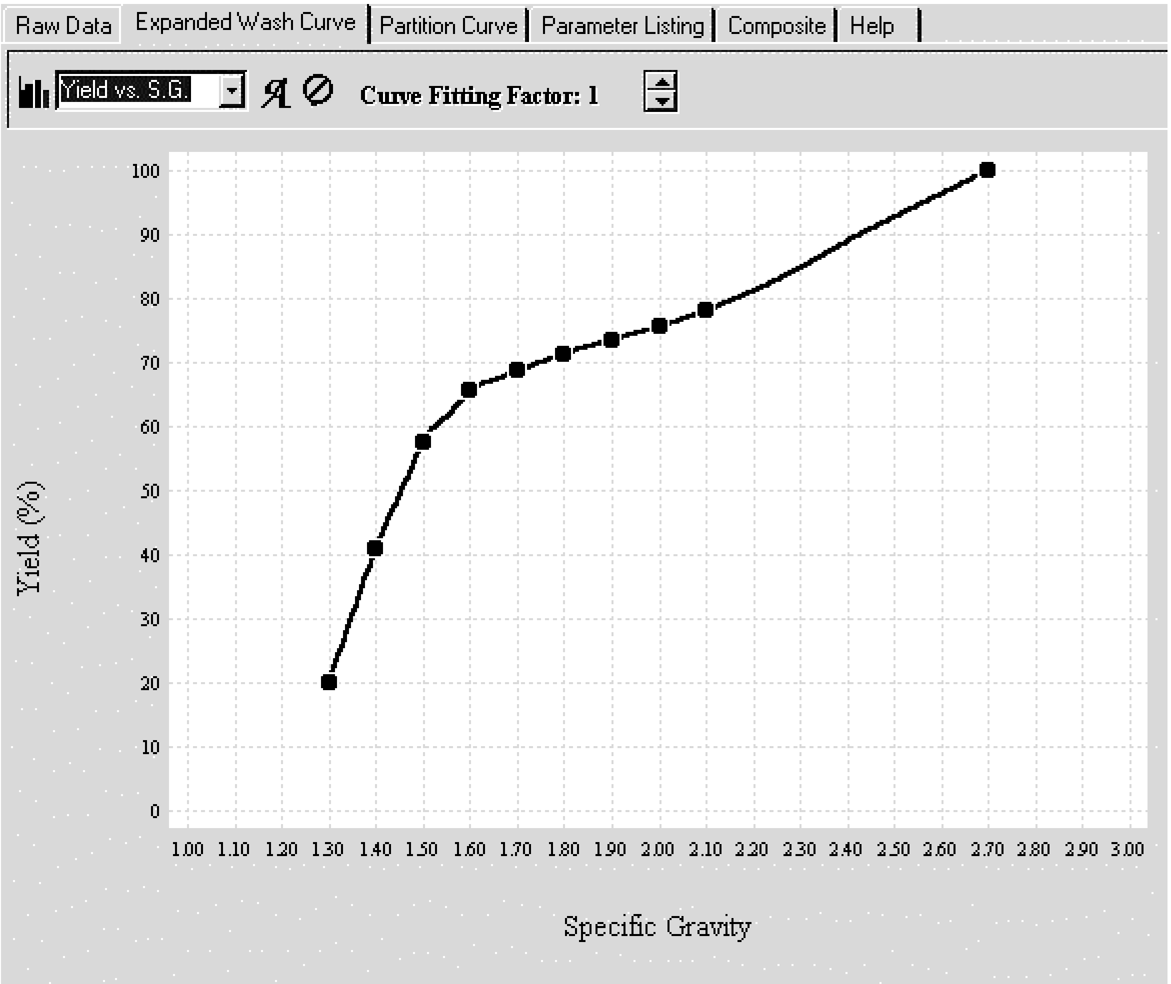
3D Response Plots Second Series - Revisited (451-465) 0.5 mm x 0







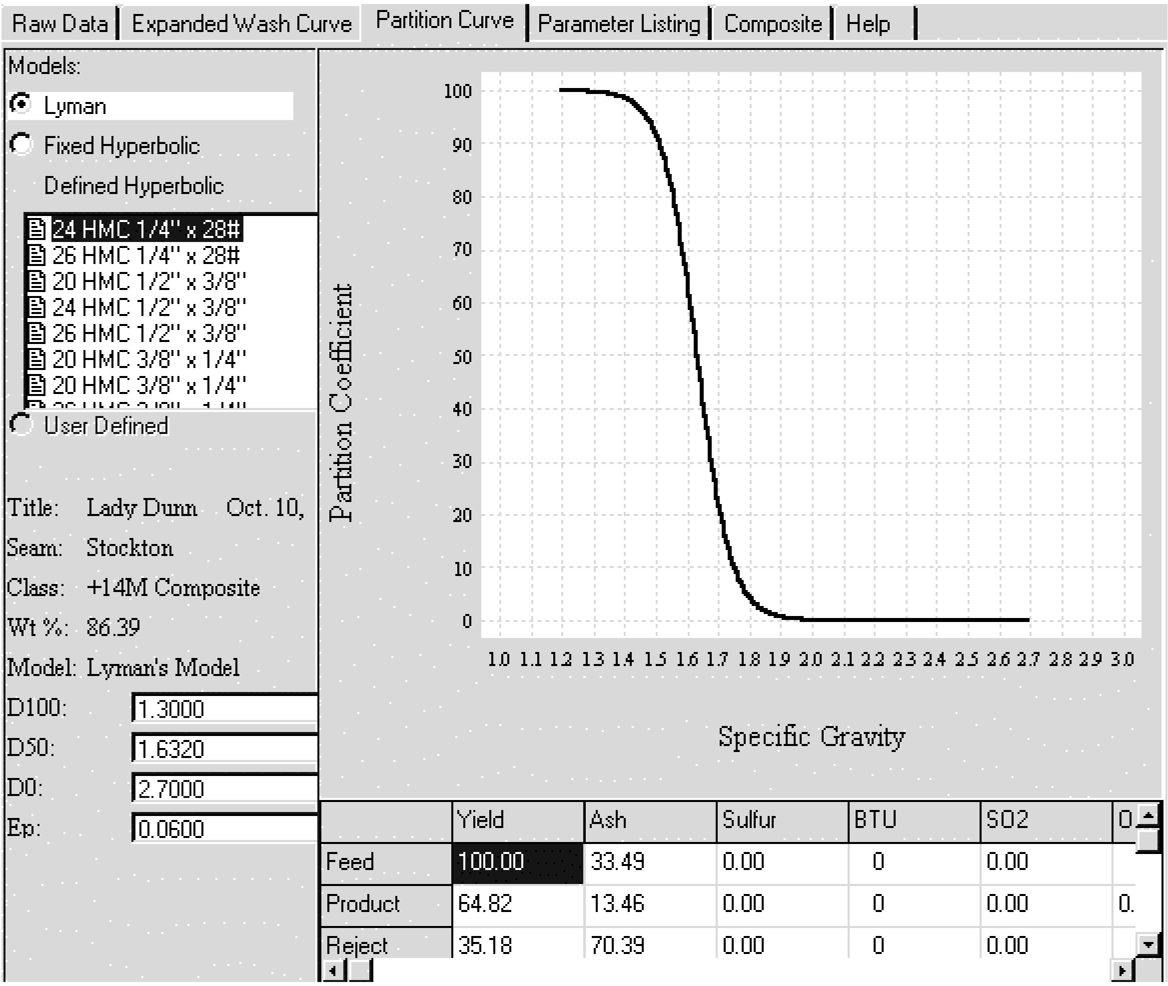
APPENDIX C QUANTITATIVE EVALUATION DATA



Yield vs. S.G.

Curve Fitting Factor: 1

Figure C-1. Heavy Media Cyclone feed washability, +1 mm, Stockton seam, taken from Coalpro computer simulation program.



Models:

● Lyman

○ Fixed Hyperbolic

Defined Hyperbolic

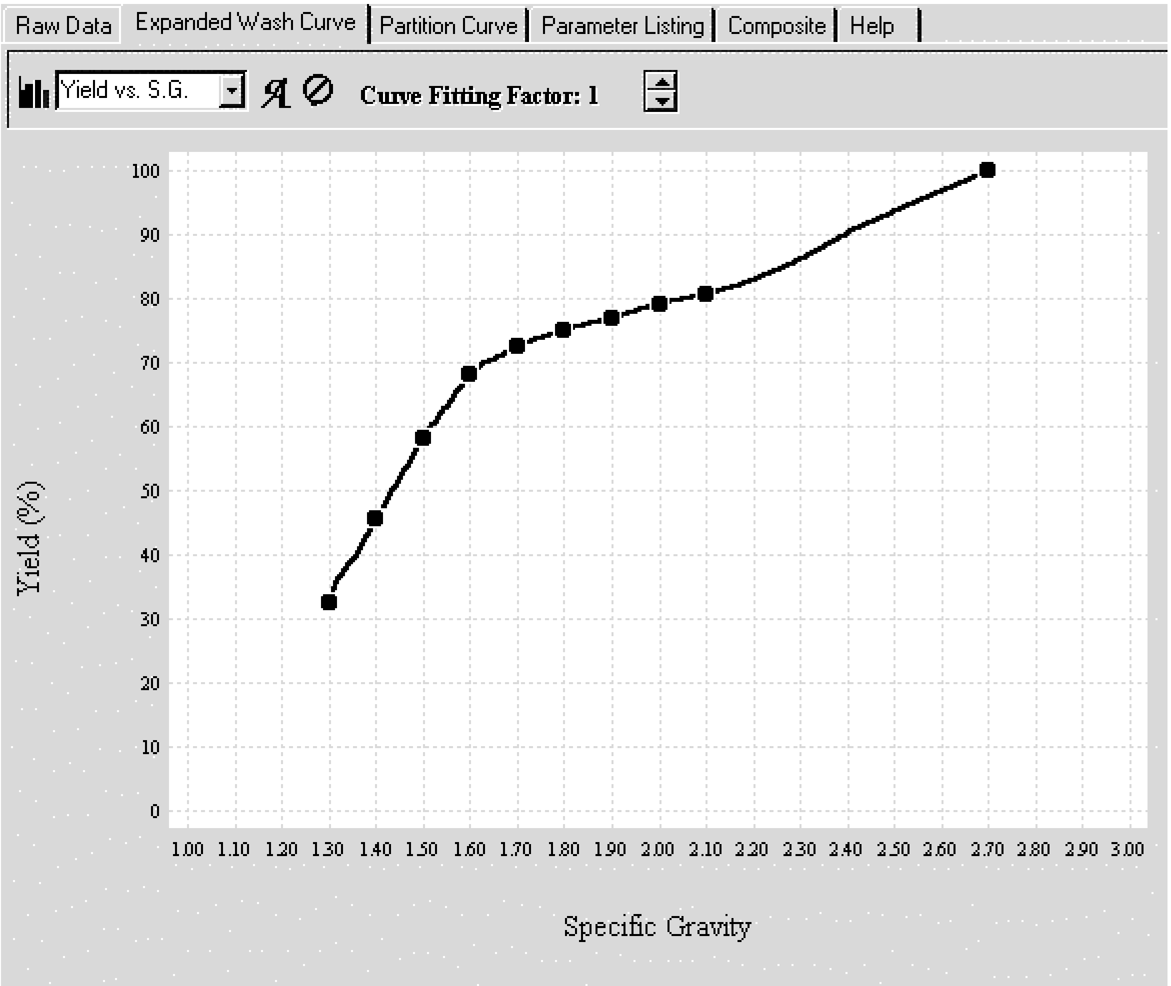
- 24 HMC 1/4" x 28#
- 26 HMC 1/4" x 28#
- 20 HMC 1/2" x 3/8"
- 24 HMC 1/2" x 3/8"
- 26 HMC 1/2" x 3/8"
- 20 HMC 3/8" x 1/4"
- 20 HMC 3/8" x 1/4"

○ User Defined

Title: Lady Dunn Oct. 10,
Seam: Stockton
Class: +14M Composite
Wt %: 86.39
Model: Lyman's Model
D100: 1.3000
D50: 1.6320
D0: 2.7000
Ep: 0.0600

	Yield	Ash	Sulfur	BTU	SO2	O
Feed	100.00	33.49	0.00	0	0.00	
Product	64.82	13.46	0.00	0	0.00	0.
Reject	35.18	70.39	0.00	0	0.00	

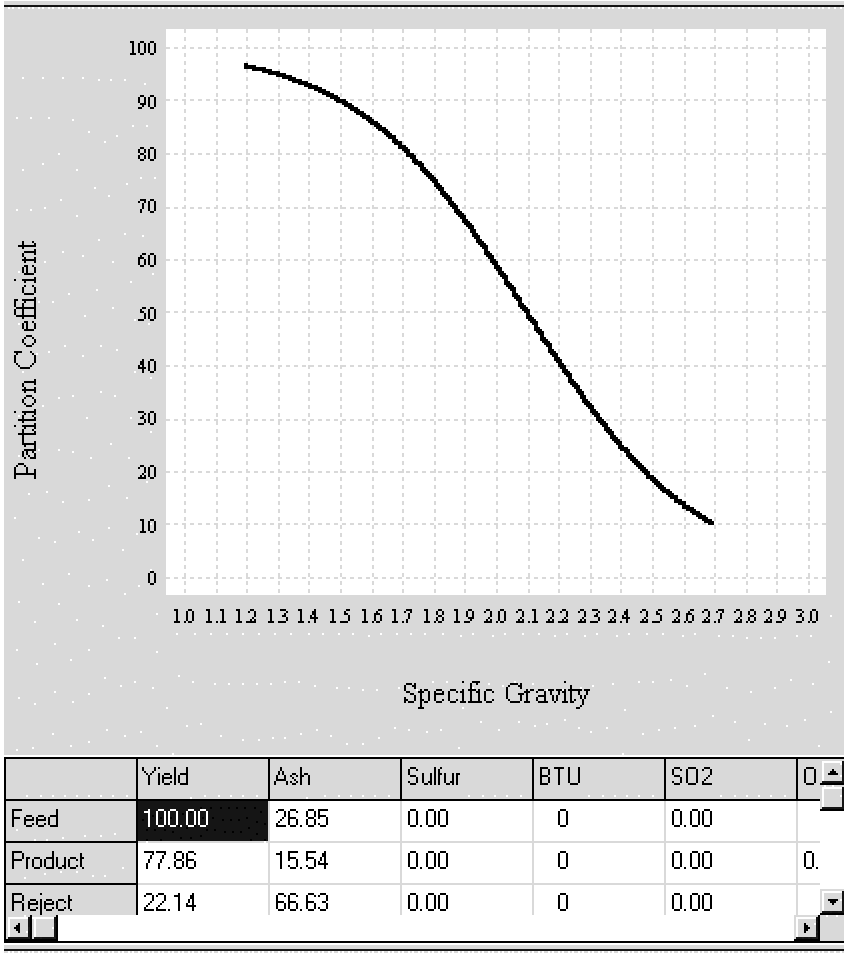
Figure C-2. Typical simulation results showing Heavy Media Cyclone partition curve, partition parameters, and product qualities, for Stockton seam. Taken from Coalpro simulation.



Yield vs. S.G.

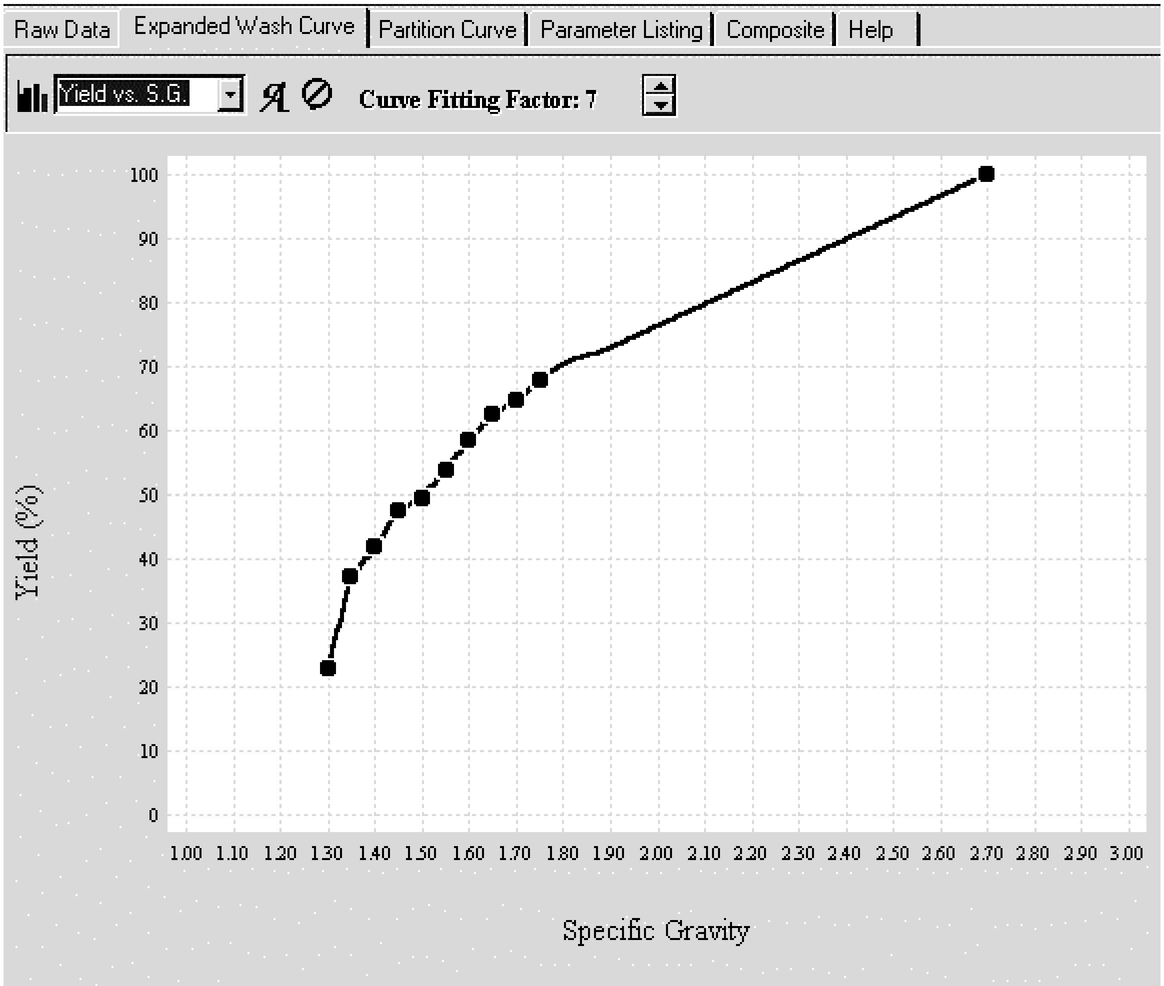
Curve Fitting Factor: 1

Figure C-3. Spiral feed washability, 0.150 x 0.25 mm, Stockton Seam.



	Yield	Ash	Sulfur	BTU	SO2	O.
Feed	100.00	26.85	0.00	0	0.00	
Product	77.86	15.54	0.00	0	0.00	0.
Reject	22.14	66.63	0.00	0	0.00	

Figure C-4. Simulation results showing partition curve and product qualities for the Spirals, 0.150 x 0.25 mm size, Stockton seam.



Yield vs. S.G.

Curve Fitting Factor: 7

Figure C-5. Heavy Media Cyclone feed washability, +1 mm, 5-Block seam. Steep slope indicates a high amount of near-gravity material.

VITA

Dennis Ivan Phillips was born on May 17, 1954, in Mullens, West Virginia. He grew up in the community of Corinne and graduated in 1972 from Mullens High School, where he was elected Senior Class President. After spending his summers working in underground coal mines, he received a B.S. degree in Civil Engineering from West Virginia University in 1977. He later received an M.B.A. from University of West Virginia College of Graduate Studies in 1990.

Mr. Phillips spent 15 years in coal preparation plant engineering and operations management. He worked as a Mining Engineer for Gates Engineering; as a Preparation Engineer for U.S. Steel; as Preparation Superintendent for Knox Creek Carbon; and six years for A.T. Massey Coal Company as Sr. Preparation Engineer, and Assistant Preparation Director; and after part of the Massey Company was partitioned to Shell Mining Company, he worked as Senior Mining Engineer in their corporate Coal Business Development group for three years. He spent a few months providing consulting services in plant design, operation, and optimization before joining the Center for Coal and Minerals Processing at Virginia Tech in 1992 as a Sr. Research Associate while working part-time on his Ph.D. He is a Registered Professional Engineer and has several publications and presentations at national meetings to his credit.