Chapter 20

CONDITIONING METHODS AND DESIGN PHILOSOPHY OF VARIOUS COAL FLOTATION CIRCUITS

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ABSTRACT

There is considerable variation in the types of coal flotation circuits in use in various parts of the world. For example, Russian plants frequently use scavenger flotation, while this is almost unheard of in the United States. The numerous circuit designs are intended to address specific problems in coal flotation, or to take advantage of the peculiarities of certain coal deposits. As a result, it can be difficult to decide which circuit is best for a given application.

For this paper, coal flotation circuits were collected worldwide from many operating plants. These circuits include varying degrees of conditioning, and complexity ranging from simple rougher flotation to multi-stage split-stream processing. The design goals of each circuit are described, and the advantages and disadvantages of each are discussed.

Extensive sampling campaigns were carried out in two plants with the most common types of coal flotation circuit (simple rougher, and rougher with deslimed feed). The performance of each plant was evaluated, and the benefits of more sophisticated processing in each case are described.

INTRODUCTION

The role of flotation in coal processing is to beneficiate material which is too fine to be efficiently cleaned by gravity separation processes, e.g., jigs or heavy media cyclones. In general, this means that regardless of raw coal feed characteristics, the flotation circuit is expected to efficiently process a flowstream composed of particles of widely varying characteristics. Consequently, the circuits used, or proposed, for processing fine coal by flotation reflect a variety of design approaches and solutions to the problem of removing ash-forming and sulfur-bearing minerals from coal.

Based on an extensive survey of coal preparation plants around the world and a study of the flotation circuits used to process different coals it can be concluded that the designer, in each case, is faced with a problem which consists of three interrelated parameters:

1. Variations in the flotation characteristics of raw coals, e.g. the rank and type of coal, liberation characteristics, particle size distribution, and the equilibrium recovery and rate of recovery of different species (size/mineralogical classes) in the slurry.

2. Variations in the product quality specifications; which can range from simple to stringent for steam coals, to high quality metallurgical coals, to ultraclean coals.

3. Variations in the total costs for different circuit configurations; including capital and operating costs, the costs of raw coal feed, the losses of recoverable coal, and the disposal of tailings. These costs are primarily controlled by the process separation efficiency required to meet market specifications.

The selection of a conditioning method and a coal flotation circuit should be determined by all of the above mentioned parameters.

CHEMICAL REAGENTS AND CONDITIONING METHODS

It is not the intention of this chapter to exhaustively list all chemical reagent accomplishments related to research, development and use in coal flotation, rather, it is to stress that reagent practice must be considered in conjunction with raw coal characteristics, equipment characteristics and the flotation circuit. For example, higher molecular weight frothers are used to improve coarse particle recovery. They function by producing a more hydrated froth which acts to reduce losses of coarse particles resulting from drainage or froth breakdown (Moxon et al, 1988; Nicol and Bensley, 1988).

Conditioning

A conditioning period is essential to ensure collisions between drops of fuel oil and coal particles and wetting of the coal particles by the oil. With many coals, adequate conditioning can be achieved by adding the oil to the feed box of the bank or to the feed slurry line upstream from the circuit. Other coals are more refractory to flotation and require intensive conditioning. In each case an optimum condition exists. Performance suffers if the system is operated away from this point. This performance degradation is observed as low yields in the first cells and increased losses in the tailings. However, it may only be noticeable with hard to float coals. The most important aspect of conditioning is achieving the necessary hydrophobicity on coarse, intermediate, and fine particles.
simultaneously, especially for the former (Smitham and Firth, 1982; Firth et al. 1978).

What follows are many examples of existing conditioning schemes and why these particular schemes are used.

**Single Stage Conditioning**

The normal practice is to add both collector and frother to a single stage of conditioning. Since the frother acts as an emulsifier for the nonpolar oil collector, it speeds the spreading of the collector on the coal and reduces conditioning time (Moxon and Keast-Jones, 1986). However, this increases the frother consumption, which can offset the benefit of faster conditioning. In the author's experience, a better approach is to mechanically emulsify the collector and add it to the conditioner alone. Frother is then added to the pulp after it leaves the conditioner, which minimizes the amount of frother adsorbed by the coal and thus reduces frother consumption.

**Two Stage Conditioning (Fig. 1)**

Intensive conditioning of the feed prior to the flotation bank is a feature of this circuit. Part of the collector is added to a high energy, high percent solids conditioner and the remainder is added to a low energy conditioner, perhaps the feed box for the bank. The mechanism by which this technique works is probably through the agglomeration of fines in the first stage, which would permit distribution of the remaining reagent to the coarse particles. The second stage allows distribution of the remaining reagent to the coarse particles or agglomerate to the coarse particles.

Using this procedure, the fines are conditioned at a sufficiently reduced reagent concentration, allowing decreased frother usage. The flotation of coarse particles is improved because the fines are unable to strip the reagents. This is particularly important with coals containing a large fraction of clay. Frother is added either before or after recombination of the fractions depending on test results. Various strategies for implementing this technology have been reported. Very often different chemical reagents are better suited for the coarse and for the fine size fraction, in split conditioning these different reagents can be added separately.

**Conditioning by Stage Addition of Reagents**

A substantial increase in coal recovery can be achieved for the same, or even a decreased, reagent addition by simply stage adding the reagents, especially in the case of hard to float coals, because the reagent is added and used as it is needed (Brown, 1962; Firth et al., 1978; Smitham and Firth, 1982). Froth mobility problems due to over addition of collector should be alleviated by minimizing collector additions in the early stages of flotation.

**Circuit Configuration**

Improving the performance of flotation circuits involves two separate aspects: 1. stabilization to reduce the effects of process disturbances; and 2. optimization, to yield optimal grade-recovery performance. Neale and Flintoff (1989) observed that certain variables, in particular collector addition, appear to displace the grade-recovery curve (i.e., optimizing behavior), while other variables, such as aeration, frother addition, and pulp percent solids tend to move the operating point along a given curve (i.e., stabilization behavior). The particular variables in each group may vary from coal to coal. Ideally, optimizing variables should be maintained at levels known to give the best average performance and control variables should be used to mitigate the effects of disturbances. Considerable
circuit stabilization can be achieved simply by providing the best conditions for flotation as follows.

1. Devoid the circuit of particles which are either too coarse or too fine and therefore would be easier to beneficiate or discard by processes other than flotation.

2. Stabilize the feed percent solids (Neale and Flintoff, 1969). The percent solids should also reflect the size distribution, e.g., minus 30 mesh (600 μm) - 7 to 10% solids, 30 x 100 mesh (600 x 150 μm) - 12 to 15% solids, and minus 100 mesh (150 μm) - 3 to 6% solids. Coarse particle recovery drops if the percent solids is too low, while the entrainment of fines is directly proportional to percent solids.

3. Water pH and hardness should be such that coal recovery is not depressed.

4. One final and important consideration is that automatic pulp level control is essential to achieving any degree of circuit stability.

Complex coal flotation circuits are more commonly used in some countries than others. This is the result of political and resource availability considerations, e.g., the desire to maximize domestic production of raw materials in order to avoid dependence on outside countries, possibly coupled with a limited availability of hard currencies to pay for importing natural resources, and frequently, the limited availability of higher grade reserves resulting in a need to process lower grade domestic reserves.

Feed Desliming (Fig. 3)

With many coals it is difficult to achieve any degree of selectivity if ultrafine particles are present in the feed to flotation. One of the earliest and still one of the more common circuit modifications is to deslime the feed in order to remove the ultrafine, harder to beneficiate, particles. The removal of this fraction facilitates recovery of the remaining coal by eliminating interference between species and significantly reducing the volume of slurry to process, which for an existing circuit, increases retention time. For a new circuit, it can reduce up front capital expenditures. However, on the negative side, it may also involve the loss of a significant portion of the coal (e.g., Firth et al., 1979; Venter et al., 1984).

This circuit has been used in Utah Development Co. preparation plants in the Bowen Basin, Australia, where flotation of the raw coal slurry is difficult due to the presence of slimes (Bateman and Miller, 1976). The feed slurries are deslimed at 200 mesh (74μm) using cyclones. An acceptable concentrate can be recovered from the underflow. The desliming improves circuit performance by minimizing entrainment of ash into the concentrate and slime interference with the flotation of coarse and intermediate coal particles. However, the cyclone overflow contains a lot of easily floatable coal and desliming at 20 μm would be more appropriate.

Split Feed Flotation (Fig. 4)

This type of circuit has two possible origins: 1. treating the slimes from a desliming circuit, or 2. treating a poorly floating coarse fraction. Conditioning after classification and prior to flotation can be conducted as required to achieve suitable floatability of the desired particles. The optimum percent solids can also be used in each circuit, i.e., higher density to support coarse particle recovery and lower density to reduce fines entrainment, respectively.

One objective of desliming flotation feed in Utah Development Co., Australian preparation plants was to limit the volumetric flow to flotation circuits. However, recognizing the presence of recoverable coal in the slimes fractions, additional flotation capacity has been installed to treat the slimes separately from the coarse particles in a parallel circuit.

The use of this technique to improve the recovery of coarse particles and overall selectivity follows from the separate conditioning of coarse and fine fractions discussed above; with the additional step of floating each fraction under optimal separation conditions, e.g., pulp percent solids, reagent additions, and aeration rates. An extreme case of applying this philosophy was described by Bearce (1961). In plant tests at the Georgetown preparation plant of the Hanna Mining Co., Cadiz, OH, the feed was divided into a coarse, intermediate, and fine fraction. Each fraction was treated in a different type of cell. The best results were achieved when machine and reagent conditions, i.e., size, power input, retention time, reagent type and dosage, etc., were optimized for each fraction. In particular, cell turbulence requirements.
differed, with coarser particles requiring higher impeller speeds.

Rougher-Cleaner (Fig. 5) and Rougher-Scavenger (Fig. 6) Circuits

Cleaning improves selectivity by reducing the effects of entrainment and rejecting a portion of slowly floating species, such as carbonaceous shale, pyrite, or composite particles. Plant tests indicate better results can be obtained, for the same total cell volume, using a rougher-cleaner circuit (3 cells each) than with a straight rougher circuit (Crawford, 1936). Open circuit cleaners are used in coal flotation because most of the rejected particles are entrained gangue. However, in some cases it may be advantageous to use the cleaner tails to dilute the rougher circuit feed pulp. Scavenger circuits assist selectivity by permitting the flotation of predominantly intermediate and fine liberated particles in the rougher bank and then coarse and/or middlings particles in the scavenger. One or two stages of cleaning have been used in the production of coking and low sulfur steam coals. Scavengers have been used in the production of steam coals from the tailings of circuits producing coking coals.

Rougher-cleaner circuits are in common use for processing coals where the feed percent ash or sulfur is high relative to the desired product specification, as in the treatment of some coking coals and in the treatment of lower quality coals. For example, they are in common use in the following regions: the United States, for coking coals (e.g., Bethlehem Steel Corp.), and some steam coals (Rutherford, 1984); France (Beauxis and Viellet, 1954; Cochet et al., 1962); U.K. (Lewis, 1950/51); South Africa (Botha, 1980; Venier et al., 1984), and West Germany (Supp, 1985). Cleaner circuits are not necessary unless higher quality concentrates must be produced from harder to process feeds. Beauxis and Viellet (1954) concluded that partial or total cleaning was necessary in order to maintain recovery at the desired grade if the feed percent ash was greater than approximately twice as high as the desired concentrate percent ash.

Reflotation of Size Classified Rougher Tailings (Fig. 7)

A variation on circuits using stage addition of reagents or split flotation is the scavenging of size-classified rougher tailings. Experience in preparation plants has shown that even when high ash rougher tailings of about 70% are produced, they can contain 5-20% plus 70 mesh (212 μm) particles with an ash content of 15-35% (e.g., see Lyakov and Mosin, 1978).

In this approach, conditions in the rougher circuit are directed towards recovery of fine and intermediate liberated particles. The rougher tailings are classified at an appropriate size, with the classifier fines being the primary tailing. The coarse fraction from the classifier is fed to a scavenger circuit. This circuit is operated under conditions designed to maximize coarse particle recovery. This circuit is well known in mineral flotation, e.g., lead flotation at Mount Isa. Firth et al. (1979) used laboratory experiments to demonstrate the benefits of this circuit over other alternatives. The low sensitivity of this circuit to fluctuations in the feed percent solids and imperfect size classification was especially noteworthy.
operate at a high relative density, and at the same time to provide the reagent as required to recover the hard to float coal, and to maximize selectivity in the scavenger. The combination of multiple reagent addition points and cleaning of only the coal requiring such treatment permits the use of a much smaller cleaner circuit. The scavenger cleaner tailings are either recycled to the rougher or scavenger or discarded.

Compared to a rougher-cleaner circuit, where all of the rougher concentrate is cleaned, this circuit allows for better use of cleaning capacity by only treating lower quality concentrates that require such retreatment. Elyashevich et al. (1960) studied the behavior of hard to float high rank coal slurries from the Shcherbinovka preparation plant. They found, for a 6 cell bank, that rougher-scavenger-scavenger cleaner (1-2-3 or 1-3-2 in each, respectively) circuits yielded better performance than a straight rougher-cleaner circuit (3-3). The Durnacol preparation plant, South Africa, converted to this circuit because it was difficult to achieve acceptable grade-recovery performance with a rougher-cleaner circuit, due to the presence of a large fraction of composite vitrinitre/inertinite and oxidized composite particles (Botha, 1980).

Rougher-Scavenger-Scavenger Cleaner Circuits (Fig. 8)

The screen underflow is fed to water only cyclones set to process circuit. As usual, prepared by wet screening. The cyclone underflow contains most of the pyrite, most of the coarse coal, and some coarse coal; the overflow contains most of the coal, fine shale, and some residual fine pyrite. The cyclone overflow is screened, the oversize is a clean coal concentrate and the undersize is sent to flotation. This feed to flotation is depleted of much of its initial ash and pyrite content, it has a top size of 100-50 mesh (150-300 μm), and it is of low and almost constant percent solids. Consequently, the flotation operation runs smoothly and efficiently with minimal supervision and perhaps even instrumentation and control.

The cyclone underflow contains some residual coarse coal; this is recovered by screening at about 50 mesh (300 μm), with the screen oversize being sent to flotation, a spiral (Campbell and Norris, 1988) or to the intermediate size processing circuit of the preparation plant, which can handle a moderate recycle stream which is depleted in coal. These circuits have been used in a number of preparation plants; e.g., the Brockdale plant, Bethlehem Steel corp., PA; the Johnstown, PA plant, Cerro-Marmo Coal group; and the Elkview, BC plant, Kaiser Resources Ltd.

![Fig. 8. Rougher-Scavenger-Scavenger Cleaner Circuit. Generally used only in Europe and South Africa, this circuit allows for better use of cleaning capacity by only treating lower quality concentrates. Best results have been achieved with difficult to float, high rank coals and oxidized composite particles.](image)

Water Only Cyclone/Classification/Flotation Circuits (Fig. 9)

Combination circuits employing the use of water only cyclones, classification, and flotation are particularly useful for processing hard to float, oxidized and pyritic coals (Schlepp and Schmidt, 1988). Feed to the fines process circuit is, as usual, prepared by wet screening. The screen underflow is fed to water only cyclones set to operate at a high relative density, and at the same time to classify shale at about 100 mesh (150 μm). The cyclone underflow contains most of the pyrite, most of the coarse shale, and some coarse coal; the overflow contains most of the coal, fine shale, and some residual fine pyrite. The cyclone overflow is screened, the oversize is a clean coal concentrate and the undersize is sent to flotation. This feed to flotation is depleted of much of its initial ash and pyrite content, it has a top size of 100-50 mesh (150-300 μm), and it is of low and almost constant percent solids. Consequently, the flotation operation runs smoothly and efficiently with minimal supervision and perhaps even instrumentation and control.

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![Fig. 9. Water Only Cyclone/Classification/Flotation Circuits. This configuration is particularly useful for hard to float, oxidized and pyritic coals. The water only cyclone rejects most of the ultrafine, liberated pyrite.](image)

INDUSTRIAL EXAMPLES

As mentioned earlier, the grade-recovery performance of flotation circuits is controlled by the mass distribution of species by size and liberation characteristics. When attempts are made to improve the response of one species vs. another, through altering operating practices, circuit layout, reagent regime, or control practices, the behavior of all other species present must be considered. The goal is to operate on the optimal grade-recovery curve, at the point where a reasonable compromise between coal recovery and loss is made in order to achieve an acceptable grade. Attaining this objective in a rational, not chance, fashion requires first determining the primary recovery mechanism for each species of interest, i.e., bubble attachment or entrainment, because different actions are required to affect entrainment, composite particle flotation, or liberated coal flotation. Then, the appropriate remedial response may be taken, e.g., decrease entrainment of ash, increase or decrease composite particle recovery, or increase true flotation of coarse coal. The overriding theme of this approach is to identify the cause of problems prior to investigating or suggesting corrective actions. The application of these concepts is discussed below for three examples; two from preparation plants and one for a coal...
Panther Valley Mine

The first operation was the Panther Valley Mine, Tamaqua, PA, which processed an anthracite coal, primarily from the Mammoth seam (Seitz, 1992; Kawatra et al, 1984). The fines processing circuit followed the general flowsheet given in Fig. 3. The desliming cyclone was fed 90 TPH and the cyclone underflow, 40 TPH of 30 x 200 mesh (600 x 74 μm), was fed to flotation. The flotation circuit consisted of three banks of three 150 cubic foot Wemco cells. The ash content of selected size fractions of the feed to flotation was: 16 x 30 mesh (1180 x 600 μm) - 10.0% ash, 50 x 70 mesh (300 x 212 μm) - 18.4% ash, and minus 200 mesh (74 μm) - 44.9% ash. The reagent regime was a fuel oil as collector and a polypropylene glycol (molecular weight of 200) as frother.

This circuit was designed and installed because about 7% of the plant feed was 30 x 200 mesh (600 x 74 μm) and 8% was minus 200 mesh (74 μm). The heavy media circuits were unable to beneficiate this material and there was too much of it to discard. At the time the designers were unaware of any means to achieve acceptable flotation circuit performance without removing the minus 200 mesh fraction by desliming. Hence a circuit design as illustrated in Fig. 3, with the 30 x 200 mesh to flotation and the minus 200 mesh fraction to waste, was selected. Samples were collected from each of the three cells in the flotation bank, so that the grade/recovery performance at each point along the bank could be determined. Samples were collected at two collector levels (0.7 and 1.4 lb/ton) and three frother levels (0.2, 0.4, and 0.8 lb/ton).

The results plotted in Fig. 10 show that for both the 16 x 30 mesh and the 50 x 70 mesh size fractions, changing the reagent dosage simply moves the plant performance along a single grade-recovery curve. This is because these particles are large enough for entrainment to be negligible, and so their ash content is unaffected by the amount of water carried into the froth. However, Fig. 11 shows that the minus 200 mesh material is highly sensitive to frother dosage, shifting to higher ash at the same recovery as the frother dosage is increased. This is caused by the increase in the amount of water which is carried into the froth at high frother dosages. Since entrainment is very pronounced at this particle size, a great deal of clay is carried into the froth and raises the ash content to unacceptable levels.
Several alterations are recommended for improving the performance of fines processing in this plant. It is shown in Fig. 10 that the recovery of the 16 x 30 mesh size fraction is only 40%. This is because the particles are too large to be easily carried into the froth by air bubbles, they should be diverted to intermediate-size cleaning processes. The data also indicates that it is best to increase the collector with the frother dosage held at levels just sufficient to maintain a froth overflow (Fig. 11). A column flotation cell could process the 50 TPH of minus 200 mesh material currently discarded as slimes.

Kitt Mine

The second operation was the Kitt Mine, PA, which treated a bituminous coal from the Lower Kittanning seam (Seitz, 1992; Kawatra et al., 1984 Kawatra and Waters, 1982). The flotation circuit processed 150 TPH of minus 30 mesh (600 μm) material in two rougher banks with four 300 cubic foot Wemco cells each. The ash content of selected size fraction of the flotation feed was: 30 x 50 mesh (600 - 300 μm) - 10.8%, 50 x 100 mesh (300 - 150 μm) - 11.0% ash, and minus 100 mesh (150 μm) - 24.7% ash (the minus 325 mesh was 23.2% of the feed and 36.3% ash). The reagent regime included fuel oil as a collector, MIBC as the frother, and a cationic flocculant as an ash depressant.

As shown in Fig. 12 (frother dosage 0.06 lb/ton) and Fig. 13 (frother dosage 0.04 lb/ton), the ash reduction at high weight recoveries was very good for the 30 x 50 and 50 x 100 mesh fractions; about 4.5 vs. 10.8 and 5.0 vs. 11.0% ash, respectively, regardless of the reagent regime. However, the ash reduction for the minus 100 mesh fraction was not as good, about 10.0 vs. 24.7% ash, due to the presence of ultrafine clay. In these tests, the effect of the frother dosage on the behavior of the minus 100 mesh stream was difficult to discern as plotted, as random variations appear to be large. The results are better understood by plotting ash and water recovery as in Fig. 14. The recovery rate of minus 100 mesh ash in the plant was strongly dependent on the water recovery rate, while coarser ash was recovered at a rate more consistent with locked particles. The reason that increasing the MIBC dosage in the plant did not have a large effect on the ash content of the minus 100 mesh material was that the absolute level of frother was not yet high enough to greatly boost the water recovery at the higher dosage level.

A series of laboratory tests with the minus 100 mesh Kitt feed were run with MIBC dosages varying from 0.108 to 0.325 lb/ton, and fuel oil dosages varying from 0.084 to 0.672 lb/ton to further discern the effects of frother dosage. The results, shown in Fig. 15, indicate that at low frother dosages, increasing the collector dosage causes the recovery to increase with a very small corresponding increase in ash recovery. At higher frother dosages, increasing the collector dosage causes little increase in combustibles recovery, but, a fairly large increase in ash recovery. Therefore, the ratio of collector to frother is not as important as the relative levels of reagents required to optimize the separation process for a particular coal.
Fuel oil addition rate (lbs/ton)  
0.14  
0.21  
0.42  

-50  -50-100  -100  
C  C  C  
△△△△△△  
■■■■■■  

Cumulative % Weight Recovery  
Cumulative % Ash Recovery  

Fig. 13. Grade-Recovery Curves for the Kitt Plant Flotation Circuit at a Frother Dosage of 0.04 lbs/ton.

There are two straightforward possibilities for improving the performance of a plant such as this one: 1. deslime the feed (as shown in Fig. 3); or 2. size classify by true flotation. In the first case, pyrite depressants are apparently liberated pyrite and pyrite present in composite particles reports to the concentrate during flotation of high sulfur coals. A means for preventing or reducing this is necessary. Many investigators have advocated the use of pyrite depressants for the treatment of such coals. However, Kawatra and Eisele (1992) studied the behavior of pyrite using laboratory tests on a Pittsburgh seam bituminous coal. They concluded that the primary recovery mechanism for fine, liberated pyrite was entrainment, while the pyrite present in composite particles is recovered by true flotation. In the first case, pyrite depressants are unlikely to be effective; and in the second they are likely to have only a marginal influence. These conclusions are supported by the lack of success experienced with industrial applications of depressants. In fact, Kawatra and Eisele (1992) suggested that the occasional successes obtained with depressants were possibly due to altering froth properties and thereby pyrite recovery by entrainment.

Based on these results, several alternatives exist for improving pyrite rejection in coal flotation: if the pyrite is present in composite particles, lower collector dosage should be used to minimize their recovery. If the pyrite is present as fine, liberated particles, recovery should be driven using higher collector dosage to minimize entrainment. Use of column flotation, where entrainment of fines is reduced by froth washing processes, or cleaner circuits, where the froth is re-washed to reduce the effects of entrainment, should yield better rejection of fine, liberated pyrite. This is probably one reason why rougher-cleaner circuits (Fig. 5) are presently used to produce low pyrite products. The composite water only cyclone/flotation circuit shown in Fig. 9 is another means for improving the rejection of fine, liberated or composite pyrite particles. Finally, good desliming practice (Fig. 3) to reject fine, liberated pyrite can drastically improve overall fines processing performance.

CONCLUSIONS

1. The best design of a coal flotation circuit will vary, depending on the nature of the coal being processed by it. For example, if a coal contains a great deal of fine clay, a desliming stage prior to flotation will reduce ash entrainment and improve the product quality. However, using a desliming stage for a low clay coal will have minimal benefit and will only reduce coal recovery.

2. In most cases, the capacity of a coal flotation circuit is not controlled by the flotation cells themselves but by the froth dewatering capacity. Since clay in the froth makes it more difficult to dewater, better removal of this clay from the froth will increase the dewatering rate and improve the capacity of the whole circuit.

3. The effectiveness of a collector is improved if the coal is thoroughly conditioned with the collector before it enters the flotation cell. The frother should be added just prior to flotation as extended conditioning with frother will tend to adsorb frother into pores in the coal and increase frother consumption.

4. The relative levels of reagent required to optimize flotation circuits are more important than a fixed collector to frother ratio. The frother dosage should be kept at level sufficient to maintain an adequate froth and collector dosage increased to obtain optimum recovery. The amount of frother should be kept at as low a dosage as possible to reduce the entrainment of ash and pyrite.

5. In the United States, many researchers have claimed to have developed effective pyrite depressants, yet these depressants are not successful on an industrial scale. The work of Kawatra and Eisele (1992) has shown that most liberated pyrite reaches the froth by entrainment, not by hydrophobic bubble attachment. Therefore, depressants which work by making hydrophobic pyrite hydrophilic will not be effective. It is much more effective to work to reduce entrainment, which will reduce both pyrite and ash in the froth product.

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Normalized Recovery Rate of Water

Fig. 14. Relationship Between Ash and Water Recovery for Individual Size Fractions at MIBC dosages of A. 0.04 and B. 0.06 lb/ton in the Kitt Plant Flotation Circuit. All test data was normalized to a solids feed rate of 100. Fuel oil dosages of 0.14, 0.21 and 0.42 lb/ton corresponding to 100, 150, 300 ml/min., respectively.

Fig. 15. Grade-Recovery Response for the Minus 100 Mesh Fractions of the Kitt Plant Feed Laboratory Tests. Fuel oil dosage was increased from 0.084 to 0.672 lb/ton in equal increments for each MIBC dosage level.
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