SCALE-UP OF USING NOVEL DEWATERING AIDS

By

Mert Kerem Eraydin

Dissertation submitted to the Faculty of the
Virginia Polytechnic Institute and State University
In partial fulfillment of the requirement for the degree of

Doctor of Philosophy
In
Mining and Minerals Engineering

R.-H. Yoon, Chairman
G.H. Luttrell
G.T. Adel
I. Yildirim

May 4, 2009
Blacksburg, Virginia

Keywords: Dewatering, Moisture, Flotation, Dewatering Aid, Vacuum, Disc, Horizontal Belt Filter, Flocculation, Hydrophobicity, Coal, Clay, Copper

SCALE-UP OF USING NOVEL DEWATERING AIDS

By

Mert Kerem Eraydin

ABSTRACT

Coal preparation plants use large quantities of water for cleaning processes. Upon cleaning, the spent water must be removed such that the final product moisture level meets market constraints. However, removal of free water from the surface of fine particles is difficult and costly, and often the results are less than desirable. Fine particles inherently have very large surface areas, and hence retain large amounts of water. Increased amounts of fines also cause denser particle packing, which creates relatively small capillaries in filter cakes and, thus, cause slower dewatering kinetics. As a result, dewatering costs for fine particles are much higher than for dewatering coarse particles.

Considering the technical and economic issues associated with dewatering coal and mineral fines, an extensive matrix of laboratory- and pilot-scale dewatering tests have been conducted to evaluate the use of novel dewatering aids. The reagents are designed to lower the surface tension of water, increase the hydrophobicity of the particles to be dewatered, and increase the capillary radius by hydrophobic coagulation. All of these are designed to lower the moisture of the filter cakes produced in mechanical dewatering processes. Laboratory-scale dewatering tests confirmed that using the novel dewatering aids can lower the final cake moisture of coal by 20-50%, while increasing the dewatering kinetics. Several on-site, pilot-scale tests were conducted to demonstrate that the process of using the novel dewatering aids can be scaled.

Based on the laboratory- and pilot-scale tests conducted, a scale-up model for the process of using the novel dewatering aids has been developed. It can predict the final cake moistures as a function of vacuum pressure, filtration time and specific cake weight. The model can be useful for the scale-up of vacuum disc filters (VDF) and horizontal belt filters (HBF). Simulation results indicate that dewatering aids can be very effective, especially when used in conjunction with HBF due to its ability to control cake thickness and drying cycle time independently.

In light of the promising laboratory- and pilot-scale test results, an industrial demonstration of the novel dewatering aids has been conducted at the Smith Branch impoundment site, which contains 2.9 million tons of recoverable coal. When the reagent was used for dewatering flotation products using a VDF, the moisture content was reduced from 26 to 20% at 0.5 lb/ton of reagent addition and to 17.5% at 1 lb/ton. The use of the dewatering aid also improved the kinetics of dewatering, increased the throughput, and reduced the power consumption of vacuum pumps by 30%.

The novel dewatering aids were also tested successfully for dewatering of kaolin clays. In this case, the mineral was treated with a cationic surfactant before adding the dewatering aids. This two-step hydrophobization process was able to reduce the cake moisture and also increase the throughput.

ACKNOWLEDGMENTS

The author would like to express his sincere appreciation to his advisor, Dr. Roe-Hoan Yoon, for the guidance, inspiration, suggestions and criticisms. Special thanks are also given to his committee members, Dr. Luttrell, Dr. Adel, and Dr. Ismail for their continued interests, useful suggestions, constructive comments and discussion to understand the overall process.

The financial support from the National Energy Technology Laboratory and Department of Mining and Minerals Engineering is deeply appreciated.

I owe special gratitude to Mr. David Brightbill, Steve Blubaugh and Steven Abbatello for their helpful suggestions and continuous support. I am also grateful to Dr. Jinming Zhang, Mr. Serhat Keles, for their friendship and support.

I would like to express my most sincere appreciation to my parents Ziynet and Ali Eraydin for their inspiration, encouragements, moral support and continued suggestions. I would also like to thank my sister Feyza for her continued support.

Finally, I would like to express my thanks to Emily A. Sarver for all her encouragements, patience and dedication.

TABLE OF CONTENTS

LIST OF FIGURES

LIST OF TABLES

CHAPTER 1 INTRODUCTION

1.1. INTRODUCTION

Fossil fuels, especially coal, oil and gas, are of great importance since they can be burned to produce energy. The Energy Information Administration estimates that in 2005, fossil fuels made up approximately 86% of United States' energy production and 76% of world consumption. The remaining sources are non-fossil sources such as hydroelectric, nuclear and other (geothermal, solar, wind, wood and waste) at 6.3%, 6.0% and 0.9 % used for energy production, respectively.[1, 2]

Fossil fuels are the most important energy sources; however, they are non-renewable. According to Oil & Gas Journal estimates, years of production left in the ground for fossil reserves are 45 years for oil, 72 years for gas and 252 years for coal. Of these fossil fuels coal has the most widely distributed reserves and it is mined in over 100 countries and on all continents except Antarctica. The total recoverable world reserve for coal was estimated by International Energy Annual-2005 to be around 908 million tons. British Petroleum's statistical review of world energy data from 2007 shows that the United States has enormous coal resources and recoverable reserves. Coal is classified into six types or ranks (peat, lignite, sub-bituminous, bituminous, anthracite and graphite) which depend on the amount and the types of carbon it contains and on the amount of heat energy it can produce. In the United States, the most widely used coal types are lignite, sub-bituminous, bituminous and anthracite. Table 1.1 shows the proved recoverable coal reserves in millions of tons for the United States and the top 10 countries as of the end of 2006.[1-4]

Table 1.1 Proved recoverable coal reserves in million tons (end-2006)

Country	Bituminous &Anthracite	Sub Bituminous & Lignite	Total	Share (%)
USA	111,338	135,305	246,643	27.1
Russia	49,088	107,922	157,010	17.3
China	62,200	52,300	114,500	12.6
India	90,085	2,360	92,445	10.2
Australia	38,600	39,900	78,500	8.6
South Africa	48,750	0	48,750	5.4
Ukraine	16,274	17,879	34,153	3.8
Kazakhstan	28,151	3,128	31,279	3.4
Poland	14,000	0	14,000	1.5
Brazil	0	10,113	10,113	1.1

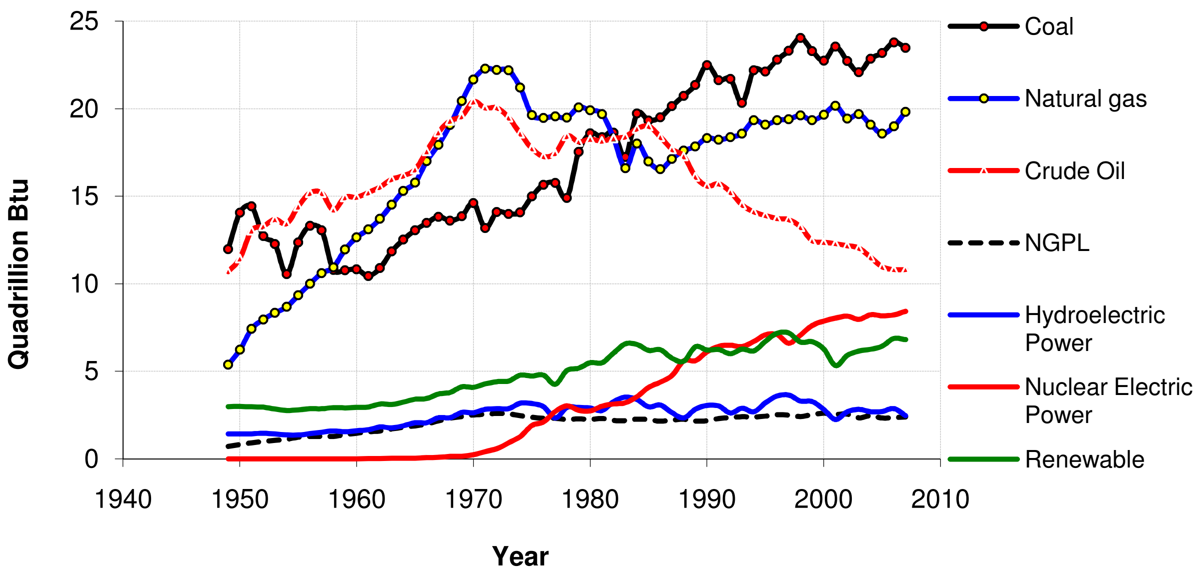


Figure 1.1 Primary energy production in the U.S by major source (Energy Information Administration)

In United States, coal represents approximately 95 percent of the nation's fossil energy reserves. Coal is mainly found in three large regions: the Appalachian Coal Region, the Interior Coal Region, and the Western Coal Region (includes the Powder River Basin). It is currently produced in 26 states, with most of it mined in Wyoming, followed by West Virginia, Kentucky, Pennsylvania, and Texas. The Energy Information Administration reports that 1,162.8 million short tons of coal were produced in 2006. Currently, approximately 50% of the electricity is produced by using coal and there are approximately 600 power plants. Coal is also one of the nation's lowest-cost electric power sources (DOE). Thus, today the electric power sector drives the coal demand for electricity production and is almost responsible for 90% of the coal consumption (EIA). Figure 1.1 shows the comparison of electricity produced from the major energy sources in United States.[1-3]

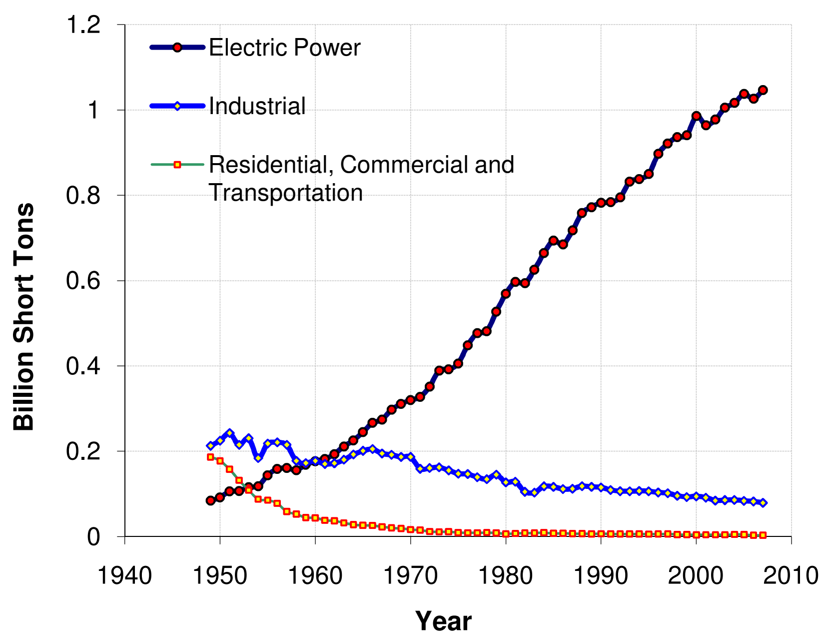


Figure 1.2 Coal consumption by sector (Energy Information Administration)

As seen from Figure 1.1, a large part of the energy production in the United States comes from fossil fuels such as coal, natural gas and crude oil. Until the 1980s, coal was surpassed by crude oil and then by natural gas. By the mid 1980s, coal started to become the leading energy source, and the use of coal increased sharply. In the 1950s, the industrial sector used coal as its major energy source. By the 1960s, the electric power sector started to use more coal. Figure 1.2 shows the coal consumption from 1949 to 2007 by major sectors.[1, 2]

This demand and increased consumption also increased coal production. United States coal production historically surpassed coal consumption. In 2004 and 2005, they were in balance. Figure 1.3 shows the relationship between production and consumption over the years.[1]

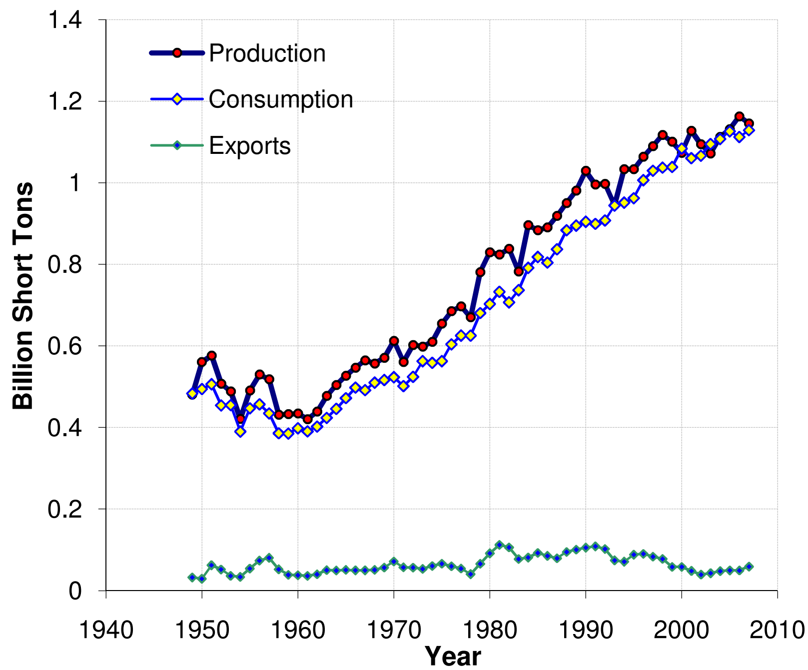


Figure 1.3 Coal production and consumption in the United States (Energy Information Administration)

The increase in demand resulted in tremendous expansions on coal mining and capacity. After the 1970s this demand created more mines, especially above-ground. The National Mining Association reports that, of the two main mining methods, surface and underground, approximately two-thirds of today's coal production is surface mining. The increase in the need of coal also improved mining technologies to produce additional coal. This additional coal is prepared to today's standard specifications dictated by the users. Thus, coal processing technologies is advanced to produce a cleaner and higher heating content end product.[5]

1.2. LITERATURE REVIEW

1.2.1. Coal Processing

Coal preparation includes four major methods, including blending and homogenization, size reduction, grading and handling, and most importantly beneficiation or cleaning for high quality product. Regardless of the intended utilization purposes, there are levels of cleaning to which coal can be economically subjected.[6-10]

Coal preparation is a very important practice for processing industries, and market demand determines the selection of preparation methods. Therefore, as shown in Figure 1.4, there are different methods of economically cleaning coal.[6-11]

In coal processing, it is typical to employ an operation by starting with the crushing and screening of run-of-mine (ROM) coal into coarse, fine, and occasionally intermediate sizes. Being exclusive of fine sizes, the coarse and intermediate size fractions are then upgraded by gravity concentration e.g. dense-medium baths, jigs, dense-medium cyclones, etc. Because the differences in densities between pure coal particles and liberated mineral impurities are sufficient to achieve an ideal cleaning process, these methods are dominantly used. The fine and ultra-fine

Figure 1.4 Levels of beneficiation

coal particles are typically upgraded by froth flotation, the dominant cleaning method for fines.[6-14]

For industries that utilize fine coal processing, froth flotation is a widely used cleaning technology that produces a consistent, high-quality product. Because coal washing is usually done in water, the fraction of fines is mostly in slurry form, which may contain over 80% water. The product is often dewatered by filtration to produce a filter cake, which is then blended with coarse coal products from heavy-medium and gravity concentration units. As a result, the amount of moisture in fine coal filter product and the amount of filter product in a preparation plant control the overall moisture content of the final product.[6, 7, 15-22]

From a materials handling and use standpoint, solid-liquid separation is a vital operation in most mineral processing industries. There are two principal methods of solid-liquid separation: filtration and sedimentation. Filtration is the act of forcing the solid-liquid mixture to pass through a medium. Solids are retained on the surface of the medium, creating a cake formation, which the liquid media flows through. Sedimentation is a separation method that benefits from the differences in phase densities of solids and liquids by allowing solids to sink in the fluid under controlled conditions.[20, 22-29]

In the conventional dewatering processes, thickeners, dewatering screens, vacuum filters (drum, disc, and belt), centrifuges, pressure (hyperbaric) filters are utilized to remove the surface water. Vacuum filtration is one of the dewatering processes widely used in the coal industry, its advantage being a continuous operation that can be utilized under relatively simple mechanical conditions. The removal of free water from the surface of fine particles is difficult and unsatisfactory by mechanical methods. The problems associated with the dewatering of fine particles are complicated. The finer particles have a larger total surface area than the coarse particles, causing very high water retention and smaller capillaries in the filter cake. Eventually, this results in high capillary pressures and slower dewatering rates.[19, 20, 23-26]

In general, the costs of cleaning fine particles are approximately 3 times higher than those for coarse particle. This leaves coal producers only a few options as to what to do with the fines in economical terms. The fine products also contain higher levels of impurities, ash and sulfur, that lead to environmental concerns. Therefore, fine particles smaller than 500μm, and ultra-fine particles smaller than 100μm, are abandoned with the discard streams if they constitute only a small fraction (5% to 10%) of the product stream. This has been the case for many U.S. coal producers. As a result, 30 to 40 million tons of fines have been discarded to waste ponds annually, representing a loss of recognized, exploitable natural resources.[8, 17, 18, 21, 31, 36]

The key reason for not completely exploiting this energy resource is the cost of the cleaning process as well as dewatering the high levels of moisture associated with the fine fraction of coal. The dewatering of fines results in a significant expense reporting to overall cleaning expenses because dewatering costs increase severely when the particle size is smaller than 500μm. The cost also includes thermal drying, which is the only practical method of drying fine coal to further decrease the moisture content. An acceptable level of moisture reduction is usually below 10% (by weight). Even though preferred moisture levels can be achieved by thermal drying, it is a capital-intensive and costly technology compared to mechanical dewatering methods.[31, 33, 40, 44-46]

There have not been any significant technical innovations in fine particle dewatering in decades because most of the fine fraction was sent to waste ponds. This lack of technological knowledge generated approximately two billion tons of fine coal in waste ponds to date, and 500 to 800 million tons of the fines are still in active ponds. On the other hand, in recent times, the industrial demand for coal has increased, and recovery of this size fraction has become more important. Recent advances in the recovery of fine and ultra-fine particles by flotation have also lead to more fine size coal production, creating an incompatibility between efficient, cleaner coal production and insufficient, fine-particle dewatering techniques.[9, 23, 31, 46, 47]

The need for understanding and enhancing fine-coal dewatering will be a considerable contribution to the performance of studies to meet the industry's needs. Studies on fine-coal dewatering will increase the availability of efficient dewatering processes that can provide lower filter cake moisture, resulting in reduced thermal drying costs, reduced transportation cost, improved product quality, increased calorific value, and minimized freezing during winter storage.[31, 36, 48]

This study was carried out to achieve a better understanding to solve the problems associated with fine particle dewatering. For this reason, two new dewatering technologies, which have been developed at Virginia Polytechnic Institute and State University, were tested. The first technology utilizes novel dewatering aids that have revealed promising options to receive significantly lower moisture. The other technology is the utilization of foam-supported dewatering, which is superior when compared to the current, commonly-used technologies.

1.2.2. Coal Dewatering

Because flotation is accomplished in an aqueous solution to produce clean coal, the product contains approximately 80% water. Although, coal fine products may represent only 20% of the weight of a preparation plant feed, this fraction is accountable for almost two-thirds of the final product moisture. From a utility viewpoint, a one-ton decrease in moisture can offset four tons of steam coal. This steam coal can be added to clean coal product, strongly indicating that the success of coal utilization is critically dependent on solid/liquid separation technology.[23, 49, 50]

As the first step in dewatering, large settling tanks can be used to remove the free water, where the slurry is thickened from 35% to 75% solid content. The second step involves subjecting the pulp to filtration methods including vacuum, drum, disc, and belt filters, centrifuges, and pressure (hyperbaric) filters to remove the remaining surface water. Despite all of this, the fine coal, filtered by using these mechanical methods, may still include undesirable amounts of water in their compositions. Thermal drying, the only fully-developed method to lower the moisture to single digits, can further decrease cake moisture contents to attain desired levels; however, the associated high energy intensives, operation costs, and special installation permissions limit the employment of thermal driers.[20, 26, 51-53]

1.2.3. Solid-Liquid Separation Methods

Solid-liquid separation is the splitting of two phases, solid and liquid, from a mixture. Because the separation is accomplished mostly by physical means, the technology to perform this process is often referred to as mechanical separation. This does not include thermal treatment. Mechanical solid-liquid separation can be broadly classified into the following distinct categories, shown in Figure 1.5.[20, 26, 51-53]

1. Sedimentation – clarification and thickening – is most efficient when there is a significant density difference between solid and liquid.
2. Filtration, in which the solid-liquid system is directed to a medium, and solids are constrained by medium either on its surface or within, where the liquid flows through.

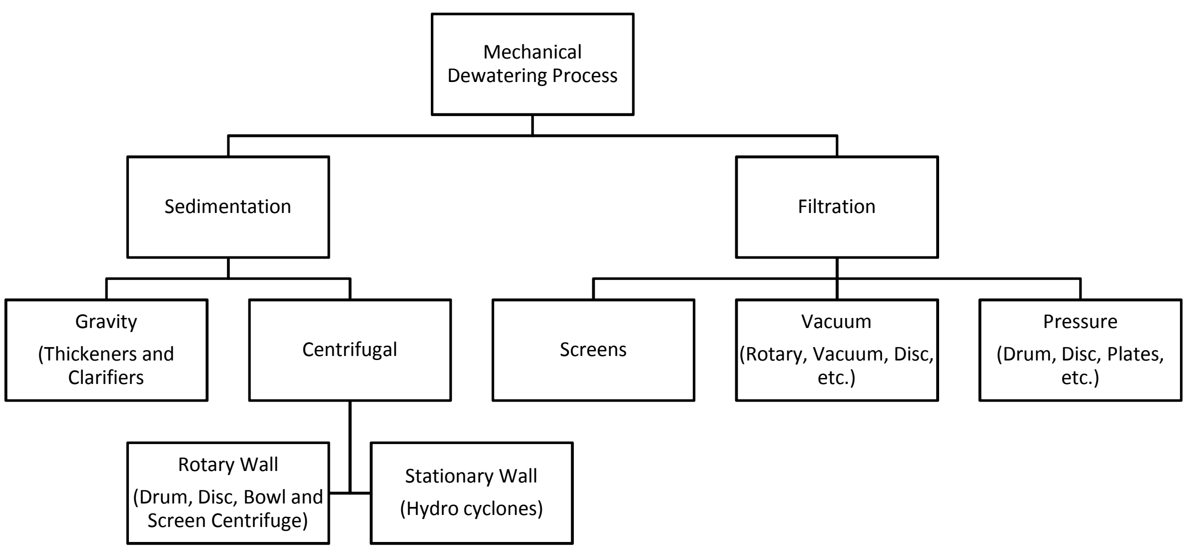


Figure 1.5 Classification of mechanical coal dewatering methods

Thermal drying is the last operation conducted on dewatered solids when these two mechanical solid-liquid separation methods are not sufficient.[22, 24, 54]

At processing plants, dewatering is practiced normally in a combination of the above methods. First, the bulk of the water is removed by sedimentation methods, followed by filtration methods. If needed, thermal driers are used to produce the desired final moisture content. Several common factors influence all solid-liquid separation steps and equipment selection, such as solid concentration, particle shape, specific gravity, surface characteristics, liquid viscosity, and, most importantly, particle size and size distribution. Figure 1.6 describes

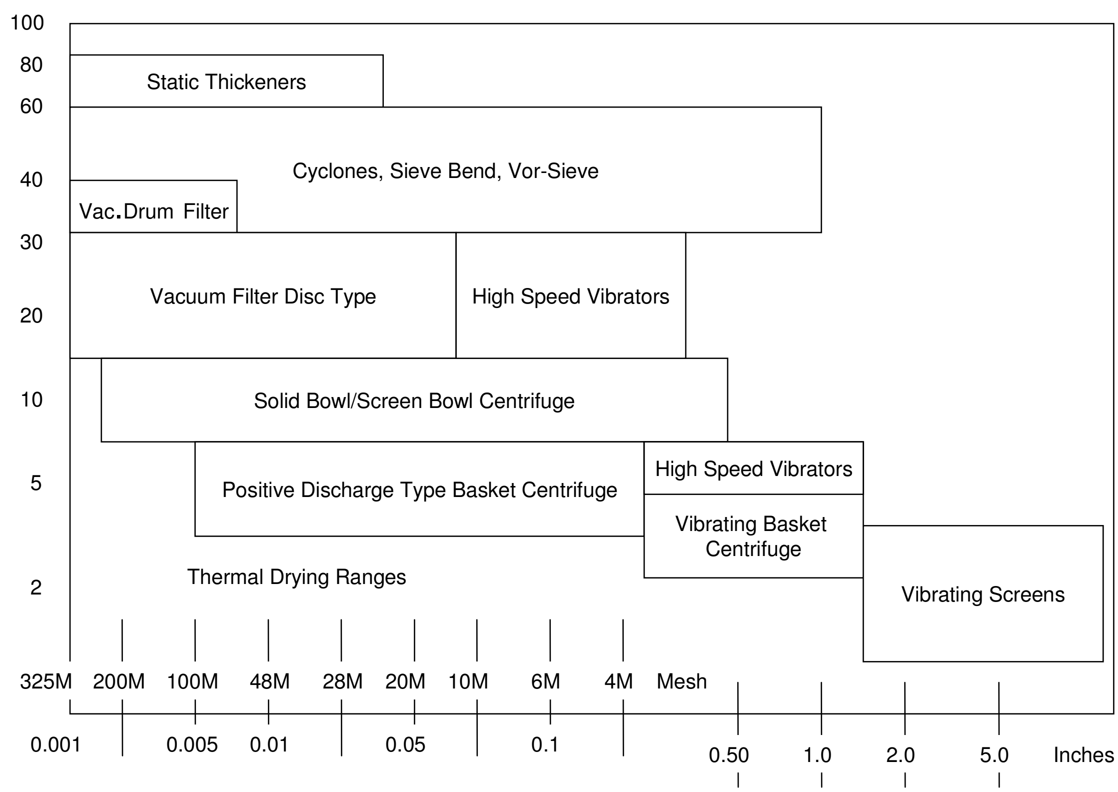


Figure 1.6 Commonly used dewatering equipment in coal industry for various size ranges and corresponding approximate moistures

the tools that are generally employed for different size fractions of coal to be dewatered.[8, 9, 20, 26, 51-53]

In general, sizes larger than 1.5 inches do not exhibit any particular/serious dewatering problems since very basic types of shaker screens can produce products with low moistures. For intermediate to 0.02inch particle size dewatering, high-speed vibrators may be used. Usually, for dewatering of 3/8 to ¼inch size particles, centrifuges of various types and designs are used, where the centrifugal force can be employed to promote/aid dewatering. For sizes smaller than 0.02 inch, solid bowl and screen bowl centrifuges can be used, even though higher moisture values may result than what is desired. Vacuum filtration becomes increasingly important with finer sizes (-30 mesh) for further dewatering.[8-10, 15, 41, 55, 56]

a) Sedimentation Techniques

Sedimentation is a collective term describing the gravity separation of the fine solids, usually under quiescent conditions, resulting in the formation of a sedimentary layer of solids and a relatively clear supernatant liquid. It is mainly used at the very early stages of the dewatering process to increase the solid content of the slurry in large capacity thickeners. Depending on the particle size and the solid percent, sedimentation processes involve the settling of solids in slurry by either employing gravitational or centrifugal force. Because the settling velocity of the very fine particles is extremely slow by gravity alone, the centrifugal force will have greater affect on settling time and velocity. Alternatively, the particles may be agglomerated into larger lumps to facilitate the dewatering. The settled solids are then collected, removed, and introduced to filtration to further reduce the moisture content of the cake before thermal drying.[20, 26, 41, 44, 52] *b) Centrifugation Techniques*

Centrifugal Separation can be considered an extension of gravity separation because the equipment creates high gravity forces to increase the settling rate of particles for the purpose of solid-liquid separation. Compared to thickeners, centrifugal separation does not require a density difference between the solid and the liquid. Even though high maintenance costs exist, centrifugal dewatering is commonly the most effective mechanical method as a result of these high forces' ability to dewater the particles in a short time and a continuous manner. Centrifugation is primarily utilized for mineral and coal processing industries because it can collectively dewater a wide range of sizes (normally with a 37.5mm upper limit, where fine is 0.5mm x 0).[8, 9, 29, 41, 61]

Centrifugal separation can be executed by using cyclones or centrifuges. Cyclones are very simple and cheap, but they suffer from limitations such as low efficiency when dealing with fine particles and their inability to use flocculants due to high shear forces. Therefore, cyclones are considered more like classifiers than thickening tools.

Centrifuges are generally classified into two groups.

a) perforate-basket type

- without transporting device
- with positive discharge system
- vibrating basket

b) bowl type

- co-current solid bowl
- countercurrent solid bowl
- screen bowl

In coarse coal dewatering, perforated basket centrifuges are most commonly used, and bowl type centrifuges are most generally used for fine particles. Bowl type centrifuges are commonly used to dewater coal at sizes from approximately 10mm to 1.0 mm. Two types of bowl centrifuges may be used, solid bowl centrifuges and screen bowl centrifuges.[8, 9, 29]

c) *Filtration Techniques*

Filtration is a widely utilized dewatering application in mineral processing industries and generally occurs after thickening. In filtration, there are four types of driving forces employed to obtain flow through the filtering medium: gravity, vacuum, pressure, and centrifugal forces. There are basically two types of filtration used in practice, surface filters, in which the solids are deposited in the form of cake on the upstream side of the relatively thin filter medium, and depth filters, in which particle deposition takes place inside the medium. In coal-preparation applications, most filters are surface filters, employing vacuum and pressure forms of driving force.[26, 29, 51, 52, 62, 63]

Vacuum filtration can be categorized into two groups: batch and continuous. In coal dewatering, where continuous filters are widely employed, batch vacuum filters are not practical. There are several types of vacuum filters that are used for fine particle dewatering. Three types of vacuum filters are rotary drum, rotary disc, and horizontal belt (HBF), or disc, filter.[26, 44, 52]

Rotary vacuum drum filters are the most widely used continuous filters for fine coal and mineral particles. They utilize a drum partially submerged into a tank of agitated slurry. Once the vacuum is applied, cake is deposited on the drum surface and discarded by various types of mechanisms, such as fixed knife and air blowing. Effluent is drained by different methods, depending on the manufacturer's design. Adjustable operating parameters, such as drum rotation speed (rpm), applied vacuum pressure, and submergence, dictate the performance of the drum filter. Changes in these conditions affect the cake formation, drying, throughput, and the degree of dewatering achieved.[26, 44, 52, 64]

The key advantages of drum filters are i) effective washing and dewatering properties, ii) low labor and operating costs, iii) wide operation variations, and iv) easy maintenance and clean operation. The main disadvantages are i) high capital cost, ii) large space requirements, iii) incompatible for fast-settling slurries, and low efficiency with ultra-fine particles that blind the filter cloth.[26, 44, 52, 64]

Rotary vacuum disc filters have almost the same fundamental design as the drum filters. Disc filters consist of a number of flat filter elements, mounted on a central shaft and connected to a normal rotary vacuum filter valve. As the unit rotates, the discs are submerged in slurry contained in a slurry tank and agitated. Gradually, cake is formed and dewatered as the unit rotates out of submergence. The filter cake is usually removed by a combination of scraper blades and a blowback mechanism. The disc filters have a low capital cost per unit area, and they supply large filter areas in smaller floor areas; however, blowback systems may cause higher moisture, and cake washing cannot be done efficiently.[26, 44, 52, 64]

Horizontal belt filters are continuous filters and consist of an endless reinforced perforated rubber belt with drainage channels, where the vacuum is applied. The filter medium/cloth sits on the rubber belt and moves along with it. The suspended slurry is fed from one end of the filter to produce cake, and filtrate is collected in a tank to be pumped out as effluent. The horizontal belt filters occupy large floor areas, and the installation costs per filter area are high; however, being fully automatic, flexible, and having relatively high speeds of operation offset these weak points.[22, 24, 29]

The pressure filters normally perform in a batch-wise manner under positive (air or hydraulic) pressures to remove water and retain solids in the form of a cake. Pressure filters are utilized very often in process industries that deal with fine, slow-settling particles exhibiting low filterability and suspensions that contain higher solid contents. Pressure filters have advantages over vacuum filters due to higher pressures and vertical incompressibility of solids. In these units, high pressure creates an increased dewatering rate and lower filter cake moisture. On the other hand, the discharge of the cake in a continuous manner from the inside of the unit is difficult and therefore pressure filters are usually employed as batch units. The high capital costs and inefficient returns associated with batch units create an additional economic disadvantage. Batch, chamber filter presses and continuous, belt filter presses are two distinct types of pressure filters most frequently utilized in coal dewatering.[25, 26, 52, 65]

d) Thermal Drying

Thermal drying of minerals and coal is the last and the most expensive unit operation performed on the dewatered materials before transportation. For that reason, the surface area of particles increases proportionally with the fineness of size and the final cake moisture. The coal's ultimate dewatering cost is strongly related to the amount of the fines. Thermal dryers are utilized to generate low moisture, maintain high coal-pulverized capacity in power plant applications, reduce heat loss, prevent freezing, and ease handling, storage and transportation.[66-68]

There are different types of dryers available, but only a small number of them are used in coal preparation. Coal thermal dryers can be categorized into two main groups, direct or indirect heat exchange. The most common dryer in use today for coal preparation industries is the direct-heated, fluid-bed type dryer. This type of dryer is generally used for fine particle drying, where hot gas passes through the fine particles inside the dryer and removes water from the unit. In the indirect heat exchange method, the fine particles in the chamber of the dryer are externally heated by hot gas to obtain dry product. It is mostly employed when environmental concerns arise. [66-68]

1.2.4. Dewatering Parameters

The performance of most of the dewatering tools that are used today depends strongly on several parameters of particle-aqueous systems. Parameters which affect the dewatering process include equipment properties, mineral type, particle size and distribution, physical and chemical properties of the mineral surface, cake structure and thickness, impurity content, surface oxidation, solid/liquid ratio, and the presence of chemical additives.[10, 14, 25]

a) *Effects of Physical Properties*

To improve dewatering to a large extent, understanding the characteristics and properties of coal and their effects on dewatering behavior is of utmost importance. Coal is the most abundant resource of fossil fuel available. It is 20 times more abundant than crude oil and over 1.5 times more than other fossil fuels and crude oil combined. Coal is an inherently heterogeneous material, possessing organic matter, mineral matter, and has an extensive pore structure. This is shown below in Figure 1.7.[1, 6, 10]

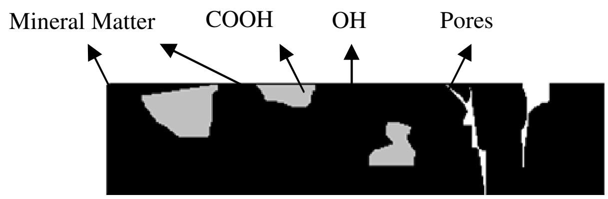


Figure 1.7 Coal structure

Depending on the rank, coal consists of various types of carbon (C-C, C=O-C, and CH_3), oxygen (O, OH), sulfur (S), nitrogen (N), and other inorganic materials. Rank is known as the category into which a particular coal can be placed with regard to its degree of coalification and/or the stage of alteration. Figure 1.8 shows simplified classifications of coals by rank.[10, 41]

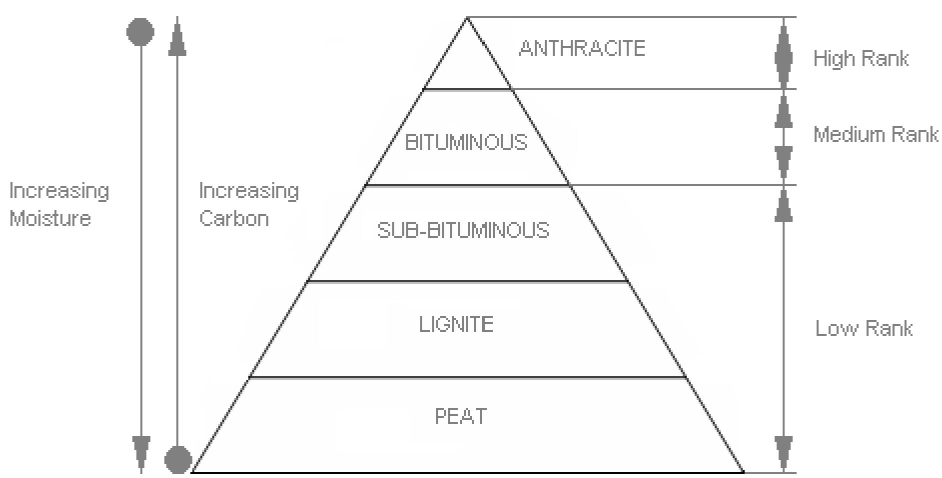


Figure 1.8 Simplified classifications of coals by rank

Understanding the different types of water associated with the fine particles is important to improve the performance of the dewatering process. Water exists in coal in different states: surface, free, inherent, or capillary water. The inherent water moisture is often defined as the water that is not readily removable by mechanical methods, chemically bound to the particle, and which is a part of the particle. It is also generally known as intra-particle moisture. This is extensively seen in the structure of low rank coals, such as lignite, and it can only be removed by thermal drying methods (over 100^oC). The capillary water is trapped in small channels of the filter cake, which, again, requires more complex and intensive methods for removal. Free water, which is not associated with solids and behaves thermodynamically as pure water occupying the bulk of the slurries, can be removed by any means of mechanical dewatering. Screens, thickeners, cyclones, and centrifuges are widely-used tools to remove this type of water.[23, 24, 26, 38, 41, 56]

The existing relationship between water and particles is of interest in the dewatering process, as well. Essentially, particle-water interaction has three main states. Figure 1.9 shows a brief description of these 3 stages.

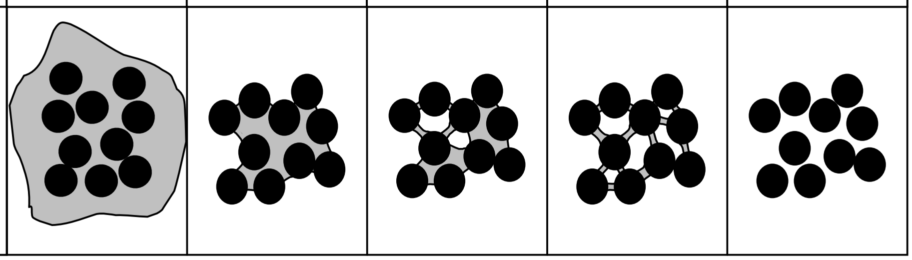
Liquid Content	More	Almost Saturated	Middle	Small	None
State	Slurry	Capillary -First- Stage	Funicular -Second- Stage	Pendular -Third- Stage	Dry
Schematic Diagram					

Figure 1.9 Schematic of particle-water relationship in cake structure

In the saturated, or capillary, state, where all the pores and voids (or capillaries) of the particles are filled with water, the liquid pressure is lower than the air pressure, and the surface tension, capillary radii, and the contact angle of the system determine the magnitude of capillary forces. Only if the applied forces of vacuum, pressure, centrifugal, and gravity are greater than the capillary forces, can water be removed. This is called the funicular state, where the remaining water starts to create bridges around the contact points of the particles. If the applied force is further increased, it will lead to the formation of lenses of water between the particles. Most of the time, this determines the final cake moisture. Figure 1.10 shows three main stages of the water content of the particles under applied forces.[24, 30, 31, 66, 69, 70]

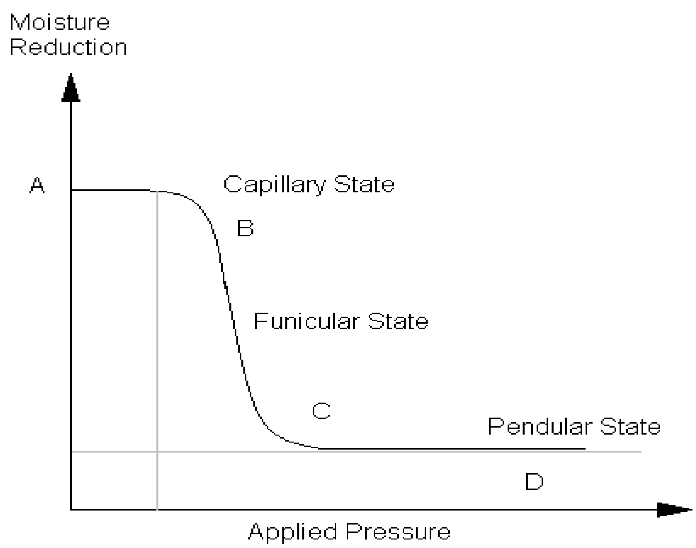


Figure 1.10 Relationship between applied pressure and moisture reduction

The deposition of solids on a filter medium is achieved by applying vacuum, pressure, or centrifugal forces to a suspension. Throughout this process, several stages occur starting from cake formation through the end of the drying cycle. The stages of the dewatering process are illustrated in Figure 1.11. Figure 1.11(A) represents an enlarged slice of the filter medium and slurry, where the initial bridging of particles begins. In this stage, the filtration commences, and the first few layers of the cake start to emerge. Figure 1.11 (B) shows the formation of cake on the filter media, and Figure 1.11 (C) shows the cake after it is formed. The time that passes during these first three stages is called the cake formation time. Once the cake is formed, it gets compacted and air starts to enter the cake structure, displacing the water from the largest pores (shown in Figure 1.11 (D) and Figure 1.11 (E)). Finally, macropore and micropore channels (capillaries) are formed where the air breakthrough is achieved, draining more water from the cake. These three stages represent the dry cycle time.[23, 30, 70]

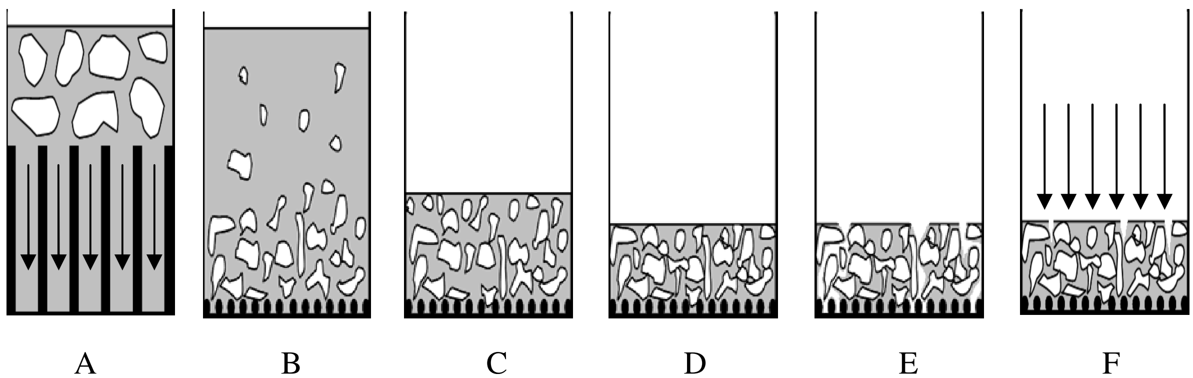


Figure 1.11 Cake formations during dewatering stages

During the drying time, applied pressure, ΔP, is not capable of removing water horizontally from the fine capillaries in the cake. Cheremisinoff, et.al, suggested a model with zones between the particles in which water is located (shown in Figure 1.12).

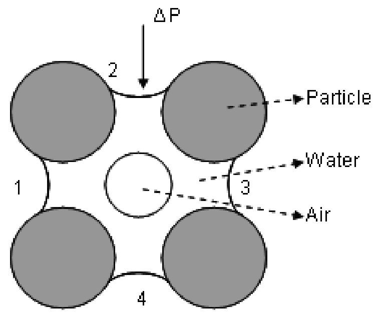


Figure 1.12 Schematic of the moisture zones

Once the pressure is applied, the water in Zones 2 and 4 can be removed and displaced with air. On the other hand, the water in Zones 1 and 3 will not be affected by the airflow, which determines the final moisture, and water will still be located between the particles. To displace water with air in these zones, chemicals can be used to increase the hydrophobicity of particles and lower the surface tension of water, which, in turn, increases water removal.[24, 26]

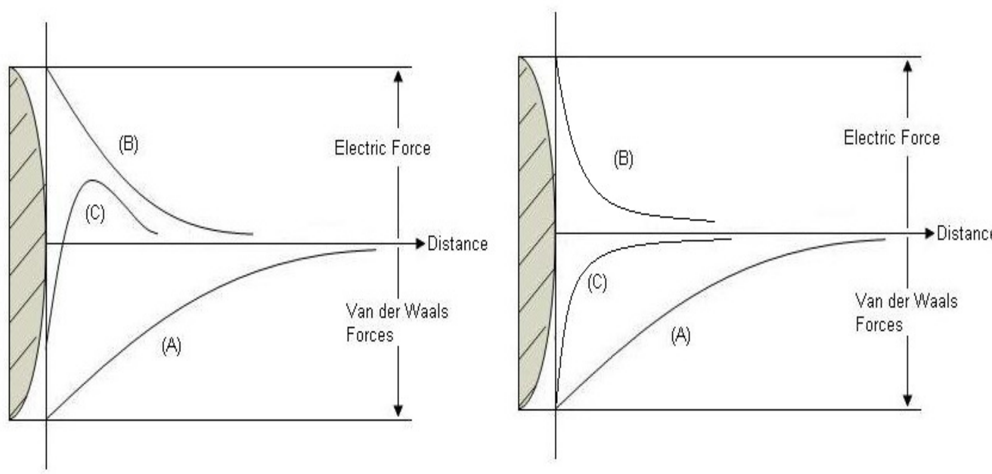
b) Effects of Chemical Properties

The use of chemical additives as dewatering aids to increase the mechanical filtration efficiency has become more crucial in mineral processing industries. No matter what dewatering equipment is employed, it is almost a standard procedure to pre-treat the slurry with the addition of chemicals. The applications show that using the appropriate chemicals may provide significant improvements in dewatering efficiency as a means of reducing moisture content in the filter cake and increasing the dewatering kinetics. Practically, chemical additives can be fitted into two main categories, flocculants/coagulants and surfactants.[25, 44, 71-73]

Flocculants and coagulants are the chemicals that change the packing density and inter-particle separation distances in the particle structure, modifying the compressibility and the drainage features of the formed cake. Surfactants, on the other hand, are long-chain polymers that are absorbed between two surfaces in order to change the surface properties. [25, 44, 71-73]

Coagulants or inorganic salts (electrolytes), such as aluminum, copper, chromic, and ferric and calcium sulfates (or chlorides) affect the composition and the extent of the electrical double layer surrounding the particles. They also change the zeta potential of the particles, as well as the inter-particle electrostatic repulsion, which in turn lead to coagulation.[20, 21, 74, 75]

The balance between the repulsive, double-layer force and the attractive Van der Waals force determines whether coagulation will occur. If the particle surfaces are not charged, particles come closer to one another, which in turn help the attractive forces bring them together to create small agglomerates. Conversely, the surfaces of the particles may be charged electrically, creating repulsive forces between the particles and preventing the spontaneous agglomeration brought about by Brownian motion. This phenomenon is shown in Figure 1.13.



A (Stabilized Particles) B (Coagulated Particles)

Figure 1.13 Potential energy curves

The curve (A) in Figure 1.13 A represents Van der Waals energy. It is an attractive force having an increasingly negative value, which is effective in small distances between particles. Curve (B) represents the repulsive electrical force and curve (C) is the outcome of these two forces, showing the maximum energy barrier for the colloidal system to become steady. At this point, because the resultant force is repulsive, coagulation does not occur. To allow the agglomeration to take place, chemical additives can be used to change the surface charge in favor of Van der Waals forces. Figure 1.13 B shows the changes on the particle surface when coagulant is introduced to the system. It reduces the electrical force and brings the curve (B) to lower values. This causes the resultant curve (C) to fall below zero and allows the particles to coagulate – if they come close enough – and the Van der Waals forces can be effective.[20, 31, 58, 71, 73, 74, 80]

The magnitude of the repulsive forces at the interface of the particles and the liquid determines the colloidal stability. This stability can be explained by Stern's double-layer theory.

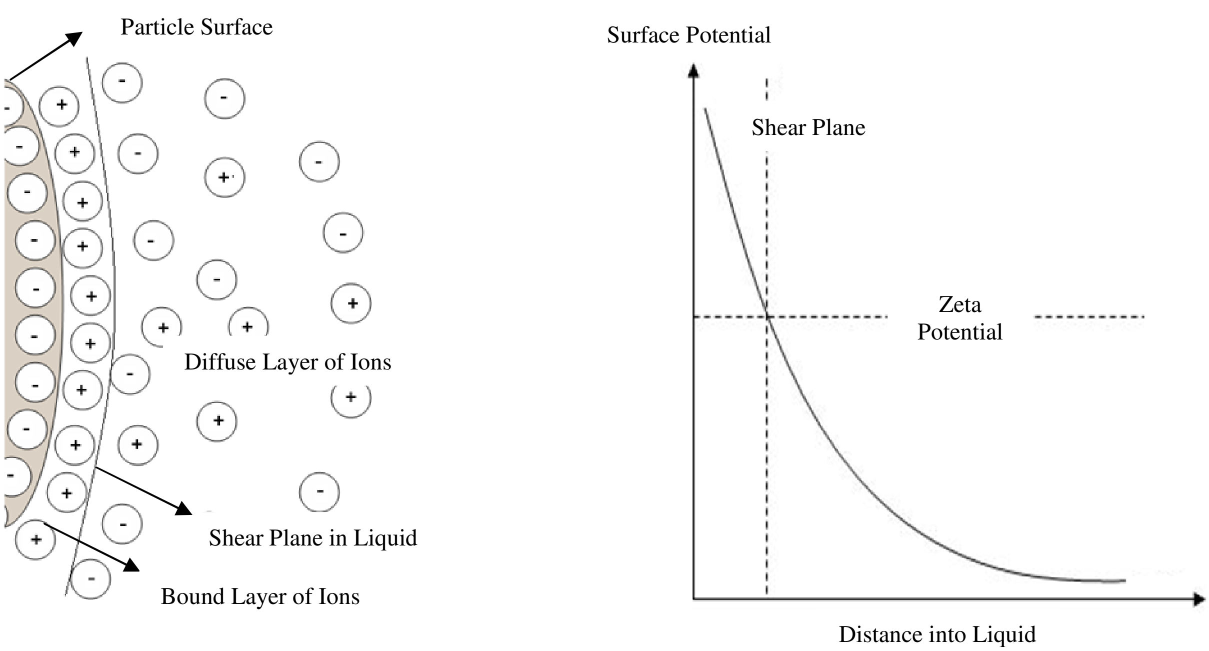


Figure 1.14 Double layer model

The suspended particles, having a certain type of charge in an electrolyte solution, attract the ions of opposing charges from the liquid, repel the ions that have similar charges, and develop an electrical double layer of ions as shown in Figure 1.14. The ions that are located in the inner stratum of the double layer are more strongly bound, and the ions that are located in the outer stratum are weaker and more diffuse. The electrical potential between these two stratums is called the zeta potential.[21, 50, 58, 71, 72, 80]

The ions of a coagulant compress the double layer by bringing the charges within the plane of shear between the bound and the diffuse layers. This compression becomes more effective when multivalent ions are in solution because they have greater charge concentration. As a result, the zeta potential is decreased. The minimum ionic concentration required to produce coagulation and its overall effect is described by the Shulze-Hardy rule. According to the Shulze-Hardy rule, to start coagulation the minimum ionic concentration must be proportional to the sixth power of reciprocal counter ion charge. The Al^{3+}, Fe^{3+}, and Ca^{2+} ions are the commonly used ions as coagulants. The pH level of the slurry is important for the hydrolysis of these salts. pH values of a slurry that are above or below the effective pH value range for a given specific salt, or coagulant, may not allow hydrolysis, and chemical dosage requirements may be higher than the ideal dosage.[21, 50, 58, 71, 72, 80]

Flocculants are long-chain polymers, or electrolytes, that cause the particles to aggregate by forming bridges between particles. Typically, flocculants are categorized into two groups: natural and synthetic. [71, 81]

Natural polymers, such as starches, gums, alginates, and polysaccharides, are mostly short-chain, neutral, organic compounds. The effectiveness of these polymers is dependent on the pH of the slurry as well. At alkaline and neutral conditions, polysaccharides are more effective while, gums, alginates, and starches are more effective at acidic conditions. Because natural polymers that are used as flocculants have short, rigid chain structure and low bonding strength, their shear strength is very low. Thus, for flocculation applications, excessive amounts of these polymers may be needed. In recent times, synthetic polymers, or polyelectrolytes, have displaced these natural materials, as these polymers can be designed to give desired behaviors, such as providing more durable flocs and more economical dosages when applied to a particular problem.[25, 58, 77, 78, 80, 83]

The synthetic polymers are based on polyacrylamide or one of its derivatives that may have very large polymer chains. These chains consist of anionic, cationic, or neutral groups, causing the polymer to uncoil and bond to the surface of the coal or clay minerals. This may result in a desired or undesired selectivity. Polymers with long chains have more contact with particles and produce large and open floccules. However, they have low shear strength and contain high residual moistures. In contrast, shorter and lesser-charged polymers generate more compact granular floccules and improve the filtration characteristics. The high shear forces due to dewatering methods might reduce the effectiveness of flocculation on fine particle dewatering. When the flocculants/polymers in solution are introduced to the slurry, they work in two stages: ion, or charge, neutralization and bridging. Although, the exact absorption mechanism is still not fully understood, initial adsorption occurs by strong bonding between the polymer and the solid particles. It involves a molecular bridge, or a series of bridges, between polymer and the solid particles. The polymer chain from the solution adsorbs onto the solid particles and, when the extended part of the chain or particles come close enough, creates bridges and continues to adsorb onto the other particles. These basic floccules grow by bridging with other solid particles until a most favorable floc size is formed. This is a quick and, unlike coagulation, irreversible reaction, which again, needs low heat rates to avoid breaking the molecular bridges in order to initiate a fast settling of solid. The use of flocculants can increase the filtration rate, as well as the cake thickness, by several multiples. The combined use of anionic and cationic flocculants exhibits further improvements. It is also reported that anionic flocculant is more effective in promoting fine coal dewatering than cationic flocculant in vacuum filtration. On the other hand, cationic flocculant was more effective in high shear centrifugal filtration. The positive increase in kinetics is, however, accompanied by an increase in moisture content resulting from water being trapped in the agglomerate structure. In addition, it was observed that the use of flocculants increased final cake moisture due to increased kinetics, thicker filter cake, and water trapped in the agglomerate structure.[25, 30, 50, 54, 58, 72, 73, 77, 78, 80, 82, 83]

Surfactants, also referred to as surface-active agents, are the chemicals that modify and control interfacial interactions by adsorption. This can take place between any two phases or immiscible components, including solid-vapor (S/V), solid-liquid (S/L), solid-solid (S/S), liquid-vapor (L/V) and liquid-liquid (L/L) interfaces in a system. Surfactants consist of two compounds, hydrophobic tail and hydrophilic head, and they are characterized by the chemical structure of their hydrophilic groups as anionic, cationic, non-ionic, and amphoteric (shown in Figure 1.15 below).[31, 80, 84]

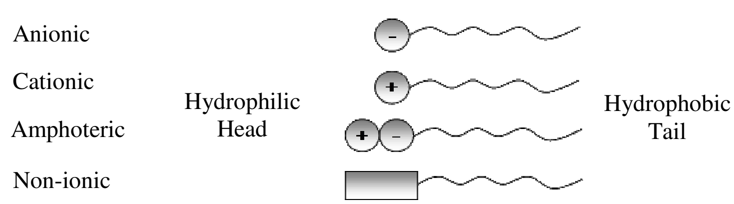
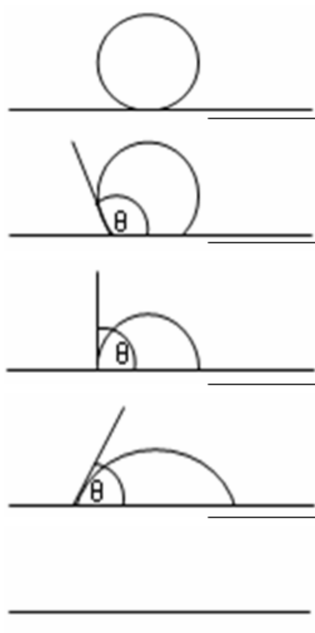


Figure 1.15 Surface active agents

In the filtration process, these surfactants are used to control the characteristics of the interface between a solid and a liquid by building molecular films, resulting in a reduction of the viscosity and the surface tension of the liquid and an increase in the contact angle. This supplies a better water drainage from the filter cake capillaries by simply lowering capillary retention forces and increasing hydrophobicity, which provides lower final cake moisture. Changes in the surface chemistry of the particles and filtrate by adding surfactants increases the contact angle and makes the surface more hydrophobic. This relationship is shown in Figure 1.16.[31, 80, 84]



Contact Angle	Hydrophobicity
θ=180°	hydrophobic
180°>θ>90°	good
θ>90°	Poor/moderate
90°θ>0°	poor
θ=0°	hydrophilic

Figure 1.16 Contact angle and hydrophobicity

If the surface potential of a particle in a liquid has the same sign as the surfactants, repulsive force is, in effect, between the particles and the surfactants which stabilizes the additives in solution, preventing any absorption on the surface of the solid. When counter ions are added to the system, these ions populate between particle surfaces and the head of the surfactant, allowing adsorption. The adsorption occurs in two ways: ion exchange and ion pairing. Ion exchange happens when the surfactant molecules are placed on the charged side of particles which have not been occupied by the counter ions. In ion pairing, the surfactant molecules displace the charge of the same counter ions and stay on the surface. Based on the orientation of the surfactant molecules, the surface chemistry of the particles changes, and becomes hydrophobic or hydrophilic. Depending on the surfactant concentration, the formation of the additives forms either a neutralized monolayer or reversed, secondary bilayer (shown in Figure 1.17).[24, 25, 31, 71, 77, 81, 86]

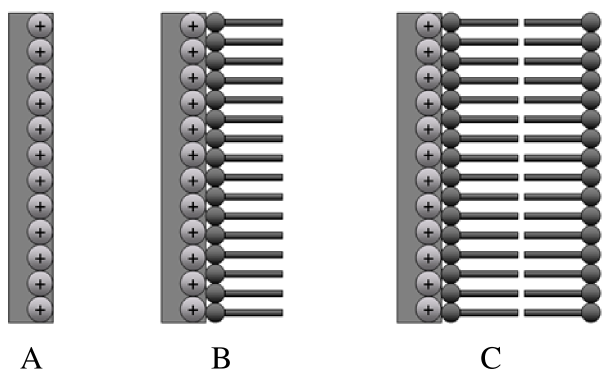


Figure 1.17 Schematic illustration of the layer formations

When the surfactant dosage is low, few or no surfaces may be coated, not causing a change in hydrophobicity (Figure 1.17-a), while higher dosages form a close-packed monolayer, representing neutralized surface charges (Figure 1.17-b). If the amount of chemical is further increased or overdosed, there will be a bilayer or reverse orientation (Figure 1.17-c), which, in dewatering, will decrease the hydrophobicity of the particles. [24, 31, 71, 77, 81, 84-86]

Polyelectrolytic polymers are highly charged, short-chain polymers that are used when the solid content is very low. These types of polymers are used when the flocculants cannot be effectively employed.[24, 31, 38]

1.2.5. Dewatering Theory

a) Dewatering Kinetics

Filtration kinetics is an important characteristic of fine particle dewatering that determines the volumetric flow rate of liquid to be removed through a porous media created by the particles and the filter throughput. Darcy first described the filtration kinetics in 1856 as the rate equation for the dewatering process. Darcy's basic filtration equation relating the flow rate is [20, 26, 44, 88] :

$$\frac{dV}{dt} = K\frac{A}{\eta}\frac{\Delta P}{L} \qquad [1]$$

where V is the volume of fluid, t is the filtration time, ΔP is the pressure drop across the cake, and L is the thickness. A is the cross-sectional area of the cake, η is the absolute viscosity of liquid, and K is the rate constant referred to as the permeability of the cake. The equation reveals a basic relationship; the rate of dewatering is proportional to the pressure gradient and the cross-sectional area and is inversely proportional to the viscosity. Equation [1] is also written in the form:

$$\frac{dV}{dt} = \frac{A}{\eta}\frac{\Delta P}{R} \qquad [2]$$

where R is the medium resistance (the medium thickness is divided by the permeability of the cake). If the suspension does not include any solid particles, all the parameters in Equations [1] and [2] will be independent of filtration time, t. This will result in a constant filtration rate for a constant pressure drop. It also represents a linear increase of the cumulative filtrate volume; however, if the suspension contains particles, resistance of the cake will increase gradually with time and lead to a drop in the flow rate.[20, 22, 26, 39, 44, 88]

In batch filtration, resistance has two components, medium resistance, R, which can be assumed constant, and cake resistance, R_c, which increases with time as a result of increase in cake thickness (shown in Equation [3]).

$$\frac{dV}{dt} = \frac{A}{\eta} \frac{\Delta P}{(R + R_c)} \qquad [3]$$

If the resistance of the cake is assumed to be directly proportional to the amount of cake deposited (w), then

$$R_c = \alpha w \qquad [4]$$

where α is the specific cake resistance.

The mass of cake deposited is a function of time (t) and can be related to the accumulated filtrate volume, V by

$$wA = cV \qquad [5]$$

where c is the concentration of solids in the system (mass per unit volume of the filtrate). By integrating and rearranging Equations [3], [4] and [5], the general filtration equation can be reached. The general filtration equation, Equation [6], is shown:

$$\frac{dt}{dV} = \frac{\alpha \eta c}{2A^2 \Delta P} V + \frac{\eta R}{A \Delta P} \qquad [6]$$

The fundamental filtration parameters, such as α and R, can be determined to evaluate the effects of different conditions on filtration kinetics. The filtration kinetics can be altered using chemical additives, which change filter cake properties, such as permeability (K), cake porosity (ε), and the specific surface area of particles (S). [20, 22, 26, 39, 44, 88]

According to Kozeny, the factors that affect permeability are the size of the particles making up the porous medium and the porosity. The Kozeny equation is related to the specific cake resistance α by

$$K = \frac{\varepsilon^3}{kS^2(1-\varepsilon)^2} = 1/\alpha \qquad [7]$$

Normally, the Kozeny constant has the value of 5 in fixed or slowly moving-beds and 3.36 in settling or rapidly-moving beds, but for many industrial applications, k is often greater than 5. Equation [7] suggests that the cake permeability decreases with decreasing cake porosity and specific surface area of particles, both of which decrease with decreasing the particle size. Substituting Equation [7] into Darcy's law gives the Kozeny-Carman equation:

$$\frac{dV}{dt} = \frac{A\varepsilon^3}{kS\eta(1-\varepsilon)^2}\frac{\Delta P}{L} \qquad [8]$$

The filtration process is also related to the flow of liquid through a bundle of capillary tubes, which is defined by the Poiseulle's equation (1846):

$$\frac{dV}{dt} = \frac{\pi r^4}{8\eta}\frac{\Delta P}{L} \qquad [9]$$

where r is the radius of the capillary tubes.[26, 39, 44, 88] *b) Dewatering Thermodynamics*

To remove the liquid located in these capillaries, it is important that the applied pressure be larger than the capillary pressure, (p), which is defined by Laplace:

$$p = \frac{2\gamma_{23}\cos\theta}{r} \qquad [10]$$

where, γ_{23} is the surface tension of liquid, and θ is the water contact angle. The Laplace equation suggests that decreasing surface tension leads to a lower capillary force and improved cake dewatering. Increasing the solid/liquid contact angle, θ, lowers the $\cos\theta_s$, and as a result, it decreases the capillary force. This controls the water removal from the cake.[25, 31, 39]

By using an appropriate surface-active reagent, it is possible to hydrophobize the surface, decrease γ_{23}, and increase contact angle by absorption in such a way that its hydrophobic part is oriented away from the surface. The Young's Equation describes the relationship between the solid/liquid contact angle and the work of adhesion, a measure of how strongly water is bound by the solid surface: [44, 71, 89]

$$\gamma_{13} = \gamma_{23}\cos\theta + \gamma_{12} \qquad [11]$$

where θ is the solid/liquid contact angle, γ_{13} is the surface free energy of the solid/air interface, γ_{23} is the surface free energy of water/air interface, and γ_{12} is the surface free energy of the solid/water interface. The free energy change per unit area, ΔG/dA, is determined by the following equation in a solid/liquid/air system. This is also known as the Dupre equation.[31, 44, 71, 89]

$$W_{adh} = \frac{dG}{dA} = \gamma_{13} + \gamma_{23} - \gamma_{12} \qquad [12]$$

By substituting Equation [11] into [12], the following relationship can be obtained (Young-Dupre Equation):

$$W_{adh} = \gamma_{23}(1+\cos\theta) \qquad [13]$$

The Young-Dupre equation suggests that the work of adhesion between solid and liquid can be calculated from the liquid surface tension and the contact angle. The spontaneous hydrophobization and effect of contact angle is illustrated in Figure 12, where A is the solid, B is the liquid, and C is the air. [31, 44, 71, 90]

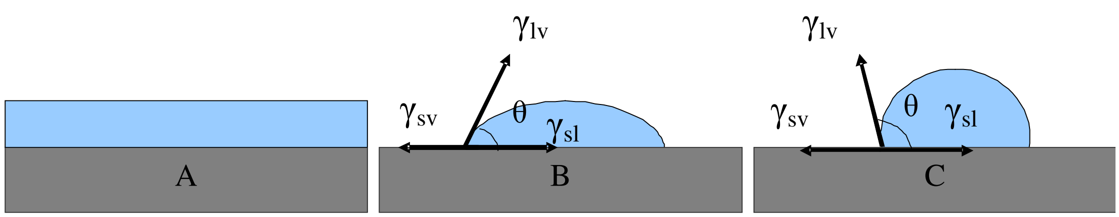


Figure 1.18 Schematic representation of solid/liquid/air interfaces

For a completely hydrophilic surface, $\theta_{12}=0^o$, $\cos\theta_{12}=1$, and W_{adh} will be at their maximum values. On the contrary, for a completely hydrophobic surface, $\theta_{12}=180^o$, $\cos\theta_{12}=0$, and W_{adh} will be at their minimum values. This, in summary, suggests that for a dewatering process, the work done on a hydrophilic solid surface can be minimized by simply decreasing water/air surface tension and increasing the contact angle, as seen in Figure 12.[31, 44, 71, 89]

Novel dewatering aids introduced in this study are non-ionic, low hydrophile-lipophile balance (HLB) number surfactants. The surfactant molecules adsorb on a hydrophobic solid surface, such as coal, as a result of hydrophobic attraction and, thus, increase its contact angle.

They also are capable of lowering surface tension, and enlarging capillary radius, all of which should contribute to decreasing capillary pressure and improving dewatering.

1.2.6. Research Objectives

The objectives of the work herein are to:

- Determine the technical and economic effectiveness of using novel dewatering aids to reduce the moisture content of fine-particle coal samples from various sites;
- Compare the results of laboratory- and pilot-scale experiments utilizing the aids to investigate if scale-up is possible;
- Use laboratory test results to formulate an empirical model for vacuum filtration systems, and then simulate filtration under a variety of operating parameters;
- Demonstrate the capabilities of the novel dewatering aids at full-scale to evaluate the feasibility of implementation; and
- Evaluate the effectiveness of two new dewatering methods – two-step hydrophobization in the presence of the novel aids, and foam-aided dewatering – to reduce the moisture of other fine-particle industrial minerals.

REFERENCES

1. Administration, E.I. Official Energy Statistics from the U.S Government. 2009 Available from: http://www.eia.doe.gov/.
2. Energy, U.S.D.O., Clean Coal and Natural Gas Power Systems. 2009 Available from: http://www.energy.gov/energysources/coal.htm.
3. Journal, Oil & Gas. 2009 Available from: http://www.ogj.com/index.cfm.
4. British Petroleum. Statistical Review of World Energy. 2007 Available from: http://www.bp.com
5. Association, National Mining. 2009 Available from: http://www.nma.org/.
6. Wills, B.A., Coal Preparation Technology: D.G. Osborne Graham & Trotman Ltd, 1988. p. 374-375.
7. Rice, N.M., Mineral Processing Technology: by B.A. Wills, 1984 Second Edition Volume 29, p. 134-135.
8. 96/03533 Coal preparation. Fuel and Energy Abstracts, 1996 37(4), p. 252.
9. P. Somasundaran, Advances in Mineral Processing. 1986 Littleton: The Society of Mining Engineers of AIME. 756.
10. D.R. Mitchell, Coal preparation, 1950 New York: Institute of Mining and Metallurgical Engineers
11. Gy, P., Mineral Processing 1979: Proceedings of the Thirteenth International Mineral Processing Congress, Warsaw Edited by Prof. J. Laskowski. 9(3): p. 299-300.
12. Pradip, Selected Topics In Mineral Processing, 1995 New Delhi:
13. 95/00081. The Future of Coal Preparation. Fuel and Energy Abstracts, 1995. 36(1): p. 6.
14. Y.A. Liu, The Physical Cleaning of Coal: Present and Developing Methods. 1982 Volume 17 p.551. 1982, New York

15. Xu, Z., Mineral Processing Technology (5th edition) 1994.
7(2-3): p. 427-428.
16. Wakeman, R.J., Vacuum Dewatering and Residual Saturation of Incompressible Filter Cakes. International Journal of Mineral Processing, 1976. 3(3): p. 193-206.
17. A survey of coal cleaning technologies (Coal Preparation). Fuel and Energy Abstracts, 2002. 43(1): p. 9.
18. B.K. Parekah, J.D. Miller, Advances in Flotation Technology, 1999
Society for Mining, Metallurgy, and Exploration. p.463.
19. 99/02473 Dewatering/Reconstitution of Fine Clean Coal Slurry
Fuel and Energy Abstracts, 1999. 40(4): p. 260.
20. Solid-Liquid Separation. Filtration & Separation, 1994. 31(1): p. 51.
21. Coal Surface Control for Advanced Physical Fine Coal Cleaning Technologies. 1990, Chemical and Petroleum Engineering Department, University of Pittsburgh
22. J. Gregory. Solid-Liquid Separation, New York: Ellis Horwood Limited 1984
23. D.J. Condie, M.H., and C.J. Veal, Modelling the Vacuum Filtration of Fine Coal. Filtration & Separation, 1996.
24. Cheremisinoff, N.P., David S. Azbel, Liquid Filtration. 1983: Ann Arbor Science. 520.
25. Akers, R.J., Solid-Liquid Separation : ed. by L. Svarovsky, Butterworths, London, 1977
The Chemical Engineering Journal, 1978. 15(1): p. 77-79.
26. R. Wakeman, S. Tarleton. Solid-Liquid Separation: Principles of Industrial Filtration 2005. Oxford: Elsevier Advanced Technology. p.339.
27. R.A. Bott, T. Langeloh Solid-Liquid Separation Lexicon, 2002
28. Classification and Dewatering of Fine Coal Concentrates. Fuel and Energy Abstracts, 2002. 43(1): p. 10.
29. H.S. Muralidhara Advances in Solid-Liquid Separation, Columbus: Battelle Press 1986
30. Condie, D.J. and C.J. Veal, Modelling the Vacuum Filtration of Fine Coal: Part 2. Filtration & Separation, 1997. 34(9): p. 957-963.
31. Asmatulu, R., Advanced Chemical-Mechanical Dewatering of Fine Particles, in Material Science and Engineering. March 12, 2001, Virginia Polytechnic Institute and State University: Blacksburg.

32. Ranj an, S. and Hogg, R.The Role of Cake Structure in the Dewatering of Fine Coal by Filtration: Coal Prep., 1996, 17, (1-2), 71-87. Fuel and Energy Abstracts, 1998. 39(3): p. 172.

33. Ivatt, S. and Shah, Dewatering of Fine Coal 138 pp. Fuel and Energy Abstracts, 1998. 39(3): p. 170.

34. Pechoc, J. Acta Mont., Ser. B, Kinetics of Filter Cakes Dewatering: 1996 Fuel and Energy Abstracts, 1998. 39(4): p. 250.

35. Improving the Vacuum Filtration of Fine Coal Using Simultaneous Dewatering and Size Enlargement. Fuel and Energy Abstracts, 1998. 39(5): p. 334.

36. Yoon Roe-Hoan; Luttrell Gerald H Blacksburg, Method for Dewatering Particles: VA, United States assigned to Virginia Tech Intellectual Property Inc. Journal of Cleaner Production, 1997. 5(3): p. 232.

37. J. Yang. X. H. Wang and. K. Parekh Improved Approaches for Dewatering of Fine Coal. Fuel and Energy Abstracts, 1996. 37(5): p. 334.

38. C. Orr, Matteson, Michael. Filtration: Principles and Practices. Chemical Processing and Engineering Vol. 1. 1987

39. C. Orr. Filtration: principles and practices (in two parts). Chemical Processing and Engineering, Vol. 2. 1977

40. Roe-Hoan Yoon, R.A., Ismail Yildirim, William Jansen, Jinming Zhang, Brad Atkinson, and Jeff Havens, Development of Dewatering Aids for Minerals and Coal Fines, Final Report to U.S.D.O.E 2004, Virginia Tech. Blacksburg, VA.

41. G., O.D., Coal Preparation Technology. London: Graham & Trotman. 1988,

42. Lockhart, N. C.; Kern, R. Research and Development Needs in Filtration and Dewatering. 1997. 38(3): p. 194.

43. Palowitch, Eugene R. Coal Preparation in Appalachia. 1996. 37(4): p. 252.

44. Smith, M.R., Solid-Liquid Separation Process and Technology, 1985 42(11): p. 2802-2803.

45. Wakeman, R.J., Selection of Equipment for Solid-Liquid Separation Processes. 1995 Filtration & Separation, 32(4): p. 328.

46. El-Shall, H. and P. Zhang, Process for Dewatering and Utilization of Mining Wastes. Minerals Engineering, 2004. 17(2): p. 269-277.

47. Evaluation of the influence of coal properties on dewatering results. Fuel and Energy Abstracts, 1997. 38(3): p. 134.
48. Cake dewatering reduces transport costs. Filtration & Separation, 2001. 38(5): p. 10.
49. Fonseca, A.G., Challenges of Coal Preparation. Fuel and Energy Abstracts, 1996. 37(3): p. 171.
50. D. J. Condie and C. J. Veal, Modeling the Vacuum Filtration of Fine Coal: part 2. Fuel and Energy Abstracts, 1998. 39(1): p. 7.
51. Holdich, R.G., Solid-Liquid Separation Equipment Selection and Modelling. 2003. 16(2): p. 75-83.
52. R. Wakeman, S. Tarleton, Solid-Liquid Separation: Equipment Selection and Process Design, 2007,
53. Rahul Konnur, S.R, Parameter Estimation and Simulation of Dependence of Constant Pressure Batch Dewatering on Initial Solids Concentration. International Journal of Mineral Processing, 2006.
54. Apling, A., Mineral Processing Technology: B. A. Wills, Pergamon Press, Oxford Corrosion Science, 1994. 36(4): p. 743-744.
55. Dickenson, T.C., Filters and Filtration Handbook. 1997: Elsevier
56. Society, T.S.A.C.P., Coal Preparation for Plant Operators. 1980:
57. R. Wakeman, S. Tarleton, Solid-Liquid Separation: Scale-up of Industrial Equipment. 2005
58. Hogg, R., Flocculation and Dewatering. International Journal of Mineral Processing, 2000. 58(1-4): p. 223-236.
59. Dewatering Equipment Database. Filtration & Separation, 2001. 38(8): p. 12.
60. Demir, I., Characterization of the Surface Properties of Illinois Basin Coals, in DOE Technical Report, K.K. Ho, Editor. 1992.
61. Advanced Centrifuges for Solid-Liquid Separation and Liquid-Liquid Extraction. Filtration & Separation, 1993. 30(2): p. 104-106.
62. Machado, A.F., Mineral Technology and Mineral Processing & Hydrometallurgy Meetings. Filtration & Separation, 1993. 30(2): p. 123.
63. Lee, C.P.C.a.D.J., Experimental Analysis of Centrifugal Dewatering Process of Polyelectrolyte Flocculated Waste Activated Sludge. 2001. 35(10).

64. Osborne, D.G., Rotary-Vacuum Filtration of Coal Flotation Concentrates. International Journal of Mineral Processing, 1976. 3(2): p. 175-191.
65. A. Patwardhan, Y.P.C., Dewatering Ultrafine Clean Coal in a T.H. Filter Press. Coal Preparation, 2006. 26.
66. B.K. Parekh, a.Z.C., Dewatering and Handleability. 2000, University of Kentucky: Lexington, KY.
67. Mahidin, et al., The Advantages of Vacuum-Treatment in the Thermal Upgrading of Low-rank Coals on the Improvement of Dewatering and Devolatilization. Fuel Processing Technology, 2003. 84(1-3): p. 147-160.
68. A.H. Mwale, P.M., D.M. Fraser, The Influence of Particle Size on Energy Consumptions and Water Recovery in Comminution and Dewatering Systems. Minerals Engineering, 2005. 18.
69. Bergins, C., Kinetics and Mechanism During Mechanical-Thermal Dewatering of Lignite. 2003. 82(4): p. 355-364.
70. Parekh, D.P.P.a.B.K., Determination of Free and Bound Water in Fine Coal Filter Cake. Coal Preparation, 2004(24): p. 217-232.
71. Yu, X. and P. Somasundaran, Colloid Chemistry in Mineral Processing : Edited by J.S. Laskowski and J. Ralston, Development in Mineral Processing, 1994. 83(2): p. 183-184.
72. Tao, D., J.G. Groppo, and B.K. Parekh, Enhanced Ultrafine Coal Dewatering Using Flocculation Filtration Processes. Minerals Engineering, 2000. 13(2): p. 163-171.
73. Moss, N.a.D., Brian, Flocculation: Theory & Application.
74. D.R. Skuse. Specialty Chemicals in Mineral Processing, 2002, Cambridge:
75. 99/00817 To enhance the dewatering of vacuum filter by adding surfactant reagent. Fuel and Energy Abstracts, 1999. 40(1): p. 81.
76. L. Besra, D.K.S., S.K. Roy, P. Ay, Flocculation and Dewatering of Kaolin Suspensions in the Presence of Polyacrylamide and Surfactants. International Journal of Mineral Processing, 2002. 66.
77. Hogg, R., Colloid Chemistry in Mineral Processing : J.S. Laskowski and J. Ralston (Editors), Vol. 12. Elsevier, Amsterdam, 1994. 41(1-2): p. 161-163.
78. S.R. Rao, Surface Chemistry of Froth Flotation.Vol. 1. 2004, New York:

79. Bimal P. Singh, The Role of Surfactant Adsorption in the Improved Dewatering of Fine Coal., 2000. 41(4): p. 195.
80. R. Laskowski, Colloid Chemistry in Mineral Processing, 1992, New York:
81. Singh, B.P., The Influence of Surface Phenomena on the Dewatering of Fine Clean Coal. Filtration & Separation, 1997. 34(2): p. 159-163.
82. B. P. Singh, L. Besra, P. S. R. Reddy and D. K. Sengupta, Use of surfactants to aid the dewatering of fine clean coal. 1999. 40(1): p. 11.
83. Hans Saveyn, G.P., Rik Timmerman, Paul Van der Meeren, Effect of Polyelectrolyte Conditioning on the Enhanced Dewatering of Activated Sludge by Application of an Electric Field During the Expression Phase. Water Research, 2005. 39.
84. B.P. Singh, L.B., P.S.R. Reddy and D.K. Sengupta, Use of Surfactants to Aid the Dewatering of Fine Clean Coal. 1998. 77(12).
85. Wheelock, J.D.a.T.D., Air-promoted Oil Agglomeration of Moderately Hydrophobic Coals. DOE/PC/93209--T10, 1996.
86. Pearse, M.J., Historical Use and Future Development of Chemicals for Solid-Liquid Separation in the Mineral Processing Industry. Minerals Engineering, 2003. 16.
87. Mwaba, C.C., Surfactant-enhanced Dewatering of Graphite and Hematite Suspensions. Minerals Engineering, 1991. 4(1).
88. Svarovsky, Solid-Liquid Separation. 1991, London: Butterworth.
89 Yoon, R.H., Advanced Mineral Processing-Fundamentals of Flotation Chemistry, in MinE 5045 Notes. 2002.

CHAPTER 2 LABORATORY AND PILOT SCALE EVALUATION OF DEWATERING AIDS

2.1. INTRODUCTION

The fine size coal production has increased as a result of several factors. These factors can be broadly explained as follows[1, 2];

i) The mechanized ROM coal production methods have increased the fine coal production

ii) The coal produced, for both coking and power industries, has to meet with certain quality constraints, such as calorific value, ash, sulfur and moisture content. To meet these requirements and produce a higher quality product, the raw material should be crushed to a lower top size for improved liberation.

iii) The possibility of economical recovery of fine size fraction of coal that reports to plants' discard streams

iv) Recovery of the fine/ultrafine coal from the waste ponds which represents an economically viable resource

Froth flotation is the widely used and the most effective separation method for fine coal cleaning. It is a wet process and the separation results in two products in the form of slurry, i.e. the concentrate (coal) and the tailing (ash-forming minerals). Of the products, the tailings are discarded to the waste ponds and the coal is further processed before it reaches to the final consumer.[2-4]

In processing plants, clean coal is dewatered by means of filters, such as vacuum disc, horizontal belt and drum filters, and centrifuges. However, the larger surface area of fine/ultra fine particles per mass results in more water adsorption. Additionally, the fine particles create smaller size capillaries in the filter cake from which water cannot be removed easily during the application of vacuum or air pressure. As a result, these dewatering methods cannot produce the desired moisture levels if there is not a thermal drier. Therefore, fine coal dewatering is the most difficult and costly operation in preparation plants.[5, 6]

Previous studies on fine coal dewatering addressed the problem and a number of dewatering aids were developed to lower the final cake moisture to the extent that is not achievable by mechanical means. In this study, laboratory and pilot scale tests were conducted on a variety of fine coal samples to do engineering evaluation of these dewatering aids; and possible industrial applications have been investigated.

2.2. GENERAL EXPERIMENTAL DETAILS

2.2.1. Samples

Laboratory- and pilot-scale dewatering tests were conducted on various samples comprised of different mixtures of coarse and fine coal. Following are the samples that were used in these tests:

i) Flotation feed (cyclone overflow or underflow)

ii) Flotation product

iii) Filter feed (blend of flotation and spiral products)

These samples were received from different preparation plants located in North America region. Table 2.1 shows the selected coal samples for the dewatering tests.

Table 2.1 Names of the operations and their locations from where the samples received for dewatering aid evaluation tests

Operation	**Location**
Mingo Logan	West Virginia
Coal Clean	West Virginia
Concord	Alabama
Buchanan	Virginia
Moatize	Mozambique
Elkview	BC, Canada
Pinnacle Reclamation	Virginia

These samples were tested

i) as received, if the sample was directly collected from filter feed stream,

ii) after upgraded with flotation, if it was cyclone overflow or cyclone underflow,

iii) after blended at different ratios.

For laboratory dewatering tests, the samples were collected or received as coal slurry in 5 gallon buckets. To minimize the adverse effects of superficial oxidation of the coal samples, the dewatering tests were conducted within 24-48 hours of receipt. The samples were first subjected to solid content determination (% solid) and particle size distribution determination. When flotation step was carried out, Denver D-12 flotation equipment was used, where kerosene and MIBC were added as collector and frother, respectively. After dewatering aid addition, each sample was conditioned with a stand-alone mixer to ensure a proper dispersion and adsorption of the dewatering aids.

2.2.2. Methods and Procedures `

a) Laboratory Scale Dewatering Test Equipment and Procedure

The laboratory-scale dewatering tests were conducted using the following equipments:

i) 2.5 inch diameter Buchner funnel

ii) 2.5 inch diameter pressure filter

Buchner Funnel was used in the bulk of the tests fitted with various sizes of filter cloth depending on the samples treated. It was mounted on a vacuum flask, which in turn was connected to a larger vacuum flask to protect the pump itself and stabilize the vacuum pressure. Before initiating the filtration, first, a known volume of coal slurry was transferred to container, to which a known amount of a dewatering aid (or a mixtures thereof) was added by means of a microliter syringe. The coal slurry was then subjected to mixing with a three-blade propeller type conditioner for a given time to ensure that proper chemical dispersion and adsorption were achieved. After conditioning, the slurry sample was poured into the funnel before opening the vacuum valve. Filtration started when a vacuum was applied to the slurry. After the cake formation, the vacuum pressure was kept on for a desired length of time to remove the remaining water trapped in the capillaries. This period was called the dry cycle time. The amount of volume added to the Buchner funnel determined the cake thickness. After the pump was stopped a representative sample was removed from the cake and dried for a give time. The filter cake was weighed before and after drying; and moisture content was determined from the dry-wet weight differences. In each experiment, the cake thickness, set up and actual vacuum pressure and cake formation time were recorded. Figure 2.1 shows the basic experimental set-up used for the Buchner funnel filtration tests.

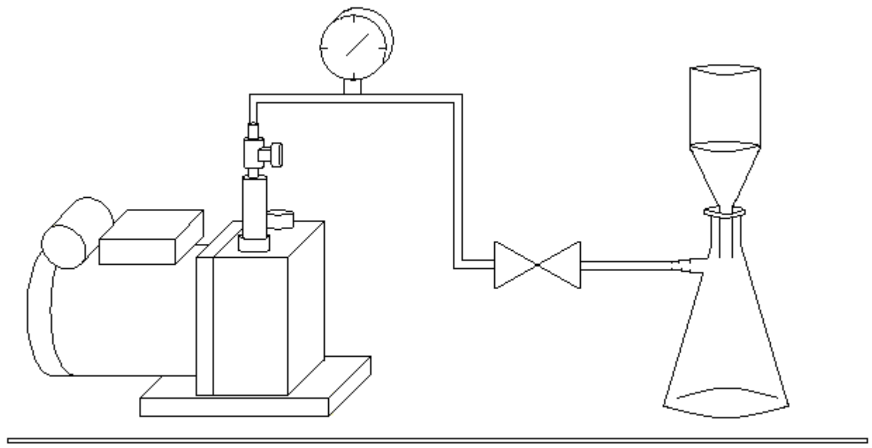


Figure 2.1 Experimental setup for laboratory vacuum filtration tests

A limited number of tests were also conducted using a stainless steel pressure filter with dimensions of 2.5 inches in diameter and 8 inches in height. Both the top and the bottom of the filter were covered with stainless steel lids. The filter cloth was placed on the bottom lid where filter cake was formed which was also the discharge spot. Compressed air was introduced from the upper side of the chamber and could be adjusted to a desired level.

b) Pilot Scale Dewatering Test Equipment and Procedure

The pilot-scale filtration equipment used in tests consisted of two types of filter units and additional equipment. The pilot scale dewatering equipments and the ancillary units were:

- 24-inch diameter Peterson Vacuum Disc Filter
- 12" wide x 7' long Wes-tech Horizontal Belt Filter
- Pilot-scale Flotation Column Module
- Conditioner Module

Vacuum Disc Filter was manufactured by Peterson Filter Company, Salt Lake City, Utah

(Figure 2.2). The vacuum disc filter consisted of a disc mounted on a horizontal shaft. The disk had interchangeable elements which could be changed for fitting and removing filter cloths. The disc rotated in a sump into which the suspension was fed (the sump also has two agitators to provide even cake formation), and a vacuum was applied through the core of the shaft. The submerged sector of the disc collected the cake and then removed it by blow-back air cake discharge system utilized in conjunction with a scraper just before re-entering to the sump. The specifications of the equipment were as follows:

- 2 ft diameter with 10 removable sectors
- 0.2 - 2.0 ft^2 of adjustable filter area by varying number of filter sectors used
- Peterson "Syncro-Blast" air cake discharge system
- 0.5 - 12 minutes per revolution
- 29 inches Hg vacuum pressure at 2.5 cfm.
- Dual filtrate sumps: 25 gal capacity each
- Connected HP: 2.25
- Dimensions: 5ft. High x 5ft wide x 4ft deep.
- Weight: 1,800 lb.

The rotational speed of the disc filter could also be adjusted which in turn enabled the operator to have a better control over the cake thickness and moisture content. The disc filter also incorporated "dual vacuum system" by splitting the vacuum manifold into two separate lines. One of these lines was directed to the submerged filter sectors while other to those that were open to the air. The line which was directed to the submerged section was equipped with a pressure reducer, so that a lower vacuum pressure could be applied to the submerged section while the increased pressure was sustained on the upper sectors (Figure 2.3).

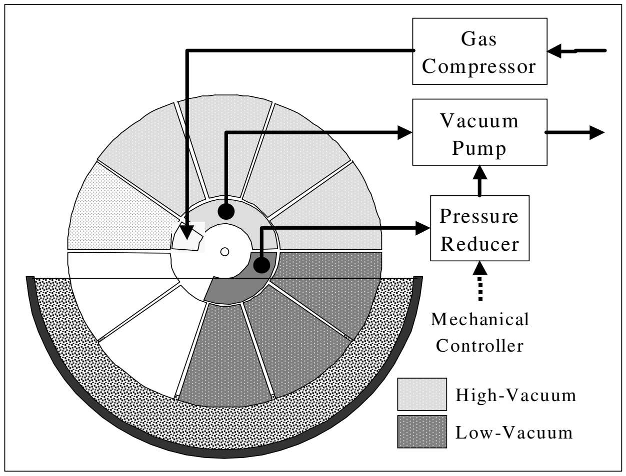


Figure 2.3 Figure illustrating the principle of the dual vacuum system

The Horizontal Belt Filter (Figure 2.4) was rented from Wes-tech. As the name implies, the area where solid-liquid separation takes place is horizontal. The unit had a continuously moving, grooved rubber belt which supports and transports the filter cloth. There were holes drilled in the center of each groove of the belt located directly over the vacuum box -running the length of the filter- from where the vacuum was applied. The vacuum also kept the cloth on the rubber belt and helped them to move forward at the same speed.

Figure 2.4 Photograph of a horizontal belt filter

Some of the specifications of the equipment were as follows:

- 12"wide x 7’ long
- 5.9 ft2 of filter area
- Variable speed Belt Drive with Motor
- Vacuum Pump with Silencer and 5 HP Motor
- Three (3) 12" diameter x 4’ high Filtrate Receivers
- Three (3) Filtrate Pumps and Motors
- Weight: Approximately 6,500 lbs

The schematic diagram of the Column Module, shown in Figure 2.5 (a), consisted of a 30-cm diameter by 3-m tall flotation column. The column was equipped with the Microcel sparging system that circulated a portion of the slurry from the bottom of the column through an in-line static mixer. Up to 100 liters/minute of air was supplied to the static mixer by a rotary air compressor. Coal slurry was fed to the column from an agitated tank using a variable-speed centrifugal pump. Pulp level in the column was maintained by adjusting the tailings flow rate using a pneumatic control valve. The valve actuated based on readings from a pressure transducer mounted in the side of the column. Wash water was added to the froth to minimize the entrainment of fine mineral matter. Chemical metering (reagent) pumps were used to add the desired dosages of frother and/or collector to the feed slurry.

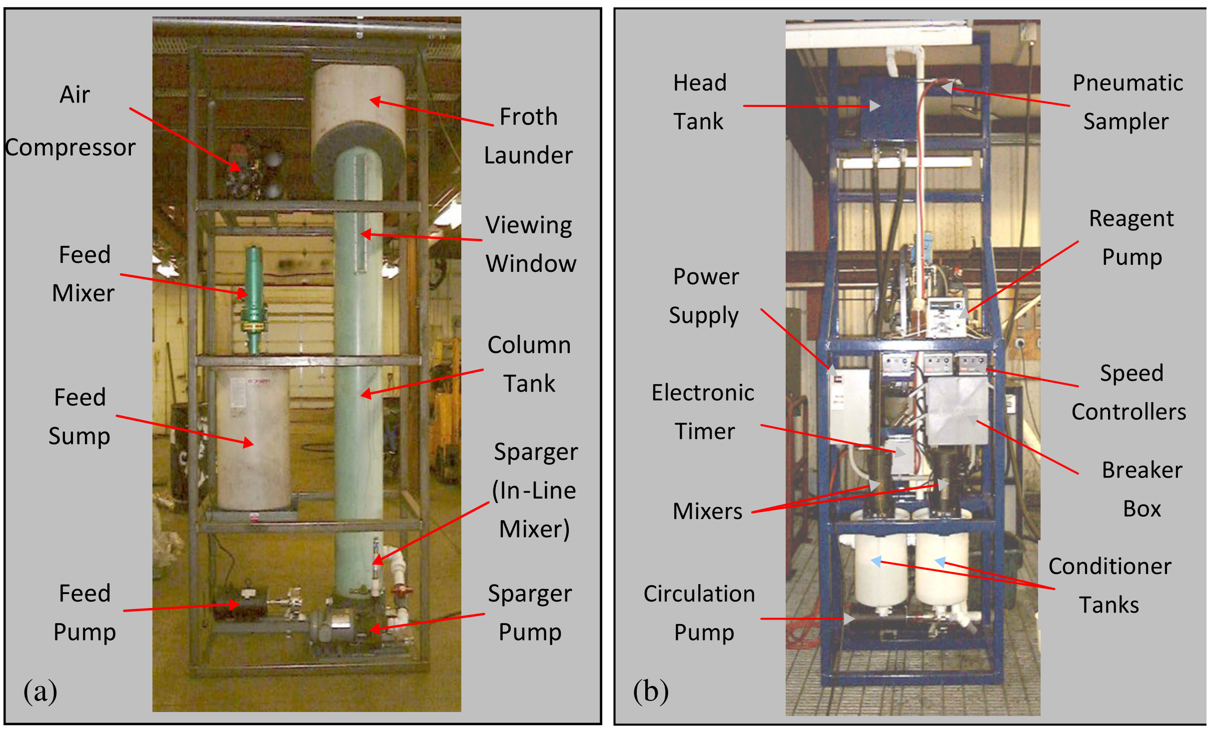


Figure 2.5 Photographs of (a) the column unit, (b) the conditioner

A schematic diagram of the Conditioner Module is shown in Figure 2.5 (b). The module incorporated two 20-liter conditioning tanks that were operated in series to provide up to 10 minutes of conditioning time. The conditioning tanks were equipped with single-impeller mixers that could be varied in speed from 0 to 2500 rpm using electronic controllers. To ensure that coarser particles did not settle when low feed rates were used, the slurry in the conditioning tanks was continuously circulated through a head tank using a centrifugal pump. The head tank was equipped with an automated sampling system that consisted of an electronic timer and a pneumatic sample cutter. During operation, the sampling system was used to divert a defined portion of circulated slurry to the Filter Module (or any other downstream operations). A chemical metering (reagent) pump was used to add the proper dosage of dewatering aid to the feed slurry as it enters the conditioning tanks. To obtain a consistent feed rate, a small peristaltic pump was installed at the plant to pump feed slurry from the plant's filter feed box to the conditioning module.

2.3. FIELD TESTING

2.3.1. Test Program Overview

A series of laboratory dewatering tests were conducted to identify the best chemicals and optimize the dewatering parameters. After successfully completing the bench-scale, a diverse set of field tests were undertaken using the mobile test modules described previously. The on-site field tests were required in order to provide site specific operational information and scale-up data for the development of the POC plant. The field tests were conducted using samples comprised of different mixtures of coarse and fine coal obtained as flotation feeds (either cyclone overflow or underflow), flotation froth products or filter feeds (blends of flotation and spiral products). As a result, the various samples were tested as-received (if the sample was directly collected from filter feed stream), after upgrading with flotation (if it was cyclone overflow or cyclone underflow), or after blending at different mass ratios. In support of the field testing effort, numerous laboratory tests were also performed. The specific procedures used for conducting these laboratory dewatering tests have been described previously.

2.3.2. Mingo Logan Site

a) Site Description

The Mingo Logan Coal Preparation Plant is located in the counties of Mingo and Logan, WV. This is in Central Appalachia Region. As Arch's principal source of metallurgical coal, the plant produces approximately 3.9 million tons (2006) at a quality of 13,000 Btu/lbs. and, 1.1 lbs. SO_2/MM Btu. The preparation plant consists of two independent 800-ton circuits, each provided with separate surge bins to ensure uniform splitting of the plant feed. Each of the circuits consists of heavy media vessels for the 160 x 7.7 mm (6 x 0.3 inch) material, heavy media cyclones for the 0.3 x 16 mesh material, spirals for the 16 x 65 mesh material, and froth flotation for the 65 mesh x 0 material. Dewatering is accomplished through centrifuging for the various size fractions. However, the dewatering efficiency was not satisfactory due to (i) the fine particle size of flotation product, (ii) the screen bowl centrifuges were not capable of capturing the fines, and (iii) the accumulation of excessively stable foam over the screen bowl centrifuges. Also, the issues resulted in high final cake moistures and loss of considerable amount of valuable material that might be recovered by other means. If recovered and dewatered more efficiently, such losses could be turned into profits.

An extensive laboratory and pilot-scale dewatering test program was conducted to study the feasibility of new approaches on more efficient dewatering methods and the recovery of the fine particle size material which reports to the plant's discard stream as a result of centrifuge's lack of fine particle capturing capability. To meet the objectives, exploratory laboratory dewatering tests were conducted to evaluate the performance of various types and dosages of newly developed fine particle dewatering aid technology. Data from the laboratory studies were used to provide technical justification for the pilot-scale test program undertaken at a coal preparation plant using a horizontal belt vacuum filter. Investigations incorporated the study for the possible utilization of these technologies in industrial applications for fine coal beneficiation.

b) Reagents

The investigation, both laboratory and pilot-scale, was performed with varying amounts of three types of dewatering aids, namely RW, RU, and RV. Since these dewatering aids are insoluble in water, they were dissolved in a solvent. In both laboratory and pilot-scale experiments diesel was used as the solvent. The ratio of reagents-to-solvent was optimized in previous studies by varying the individual dosages (0.5 to 3lb/ton), while maintaining the total blend dosage constant. For this test program, the optimum combination for a given dewatering aid and solvent is one to two (1:2) dewatering aid-diesel ratio. The flocculants, Nalco-9822 and Nalco-9806, were prepared at a 0.09 % solution and used both alone and in conjunction with dewatering aids. After examining various dewatering aid and flocculant combinations and their orders of additions, the best combination was found to be to add dewatering aids first, then to add flocculants. As it is going to be discussed in the following sections, the agitation and mixing intensity play an important role in determining the chemical performance. The dewatering aids require certain amount of mixing intensity; however, it may be excessive shear for a given flocculant. Excessive shear may lead to chain breakage, which reduces the effectiveness of a flocculant for bridging. Therefore, flocculant was added after mixing the dewatering aids; and the new combination was conditioned at a very low intensity for 15-20 seconds.

Table 2.2 Size distribution of Mingo Logan flotation product

Size (mesh)	Weight (%)
65	10.95
65X100	9.25
100X150	11.91
150X325	22.25
325	45.65
Total	100

c) *Coal Samples*

The dewatering tests were conducted on feeds comprised of three different mixtures of coarse and fine coal feeds. The first series (Series A) was conducted using a fine coal sample from a classifying cyclone overflow that had been cleaned by flotation. The product solid content was 23% by weight. Approximately 45%of the material was finer than 325 mesh size. Table 2.2 shows the sieve analysis results on flotation product sample.

In some of the experiments, the cyclone overflow sample was also used to produce clean flotation product for dewatering tests. The second series (Series B) was conducted using a mixture of fine and coarse coal. The flotation product sample was mixed with a portion of the spiral product at a ratio of 3:1 (i.e., 75% fine coal and 25% coarse coal) to prepare the dewatering feed slurry for dewatering tests. The solid content of the combination of

Table 2.3 Size distribution of Mingo Logan mixture sample

Size (mesh)	Weight (%)
35	17.80
35x100	22.25
100x325	23.25
325	36.71
Total	100

spiral/flotation product slurry was approximately 27% by weight. The third and final series (Series C) was performed with a feed comprised of 50% coarse coal from the spiral circuit and 50% fine coal from the conventional flotation circuit. Table 2.3 shows the size distribution of mixed clean spiral and flotation product (at 1:1 ratio) sample. The solid concentration of the slurry was about 25% by weight, while 36% of the solids were minus 325 mesh.

d) Laboratory Procedures

All the samples including the flotation product, obtained in the lab using cyclone overflow sample or collected directly from the preparation plant, and the mixture sample were conditioned with the dewatering aid before the dewatering tests. As stated earlier, a conditioning system is very important for the chemical dispersion and adsorption. The surfactants adsorb on the surface of the solid leading to an increase in the solid/liquid contact angle and a decrease in liquid surface tension. Each slurry sample subjected to mixing to ensure that proper chemical dispersion and adsorption was achieved. The sample was first agitated in a 300 ml Plexiglas cell equipped with a three-blade propeller-type mixer. The mixer was designed to control the mixing strength by adjusting the input current and voltage of its motor. Once the sample was mixed, it was transferred to a Buchner vacuum filter and subjected to dewatering tests. When flocculants were used, proper mixing was supplied. This ensured that high shear conditions neither degraded nor rendered the flocculant ineffective. Essential test parameters affecting the final cake moisture and filtration performance, such as pressure level, specific cake weights, and filtration time were recorded. Upon receipt, all the samples were subjected to dewatering tests within 24 hours to minimize the effects of aging and artificial oxidation that can change surface properties rapidly and affect the filtration behavior.

e) Pilot-Scale Procedures

Pilot-scale dewatering tests were conducted on feeds comprised of mixtures of coarse and fine coal feeds. Feeds specifically used, for example, are fine flotation product or combinations of coarse spiral and fine flotation products. The coal feeds, flotation product/spiral product coal sample, were reasonably consistent for most test work. The blend sample contained approximately 28% of solids by weight and 12% ash on dry basis. Table 2.4 provides the particle size and ash analysis results. The particle size is almost uniformly distributed over the range of 18 mesh x 0, but obviously, the fine particles contain more ash. As shown, 68% ash is included in the fine fraction (325 mesh x 0). Table 2.5 shows the particle size and the ash contents of each size class of the flotation product. Of the sizes, 43% of the particles was passing 325 mesh (-44 μm), and more remarkably, 81% ash was distributed in this fine fraction.

Table 2.4 Particle size and ash content analysis of Mingo Logan plant spiral/flotation product mixture (on-site samples)

Size (mesh)	Weight (%)	Ash (%)	Ash Distribution (%)
35	19.66	3.74	6.09
35x100	22.54	4.75	8.87
100x325	23.83	8.55	16.87
325	33.98	24.23	68.18
Total	100.00	12.08	100.00

Table 2.5 Particle size and ash content analysis of Mingo Logan plant flotation product (on-site samples)

Size (mesh)	Weight (%)	Ash (%)	Ash Distribution (%)
35	1.77	2.53	0.48
35X100	15.06	2.08	3.45
100X325	40.22	3.38	15.00
325	42.96	17.1	81.07
Total	100.00	9.06	100.00

The on-site pilot-scale test work was conducted using a conditioning tank and the pilot-scale horizontal belt filter built by WestTech/Delkor. The spiral/flotation product mixture intercepted from the screen bowl feed pipe was fed to a 35 gallon conditioner. Flotation sample was taken from the distribution box. After conditioning with the dewatering aids under test, a desired amount of coal slurry was then fed to the filter. The total feed rate to the conditioning tank was generally controlled at 13gal/min. This allowed approximately 3 minutes of conditioning time for dewatering aids.

The dewatering aids, RU and RV were used. The flocculant, R9822 from Nalco, diluted to 0.09% solution, was directly pumped to the horizontal belt filter's feed pipe. The belt speed of the filter was adjusted to allow the materials travel over the vacuum zone in the time interval between 65 to 184 seconds, which enables to investigate the effect of different cake thicknesses under certain filter feed rate, and of different dry cycle times. The cake sample was taken for moisture analysis periodically from the ending point of the belt when a steady-state was achieved after the test parameters were changed.

f) Laboratory Test Results (Series A – Fine Coal Only)

In this series of experiments, Mingo Logan's clean coal product from the conventional flotation circuit and cyclone overflow samples were used. The first dewatering tests were conducted to determine whether the flotation product coal samples provided by Mingo Logan Preparation Plant would respond well to the addition of the novel dewatering aids. The preliminary results showed that with the addition of dewatering aids, it is possible to reduce the final cake moisture content of fine coal cake by about 20%, while also increasing the rate of dewatering. The surface moisture was reduced down to about 19% using 3 lb/ton RW, where it was approximately 23% for control tests at about 8-11mm cake thickness. As many factors influence the filtration performance, particle size distribution and corresponding ash contents were the dictating parameters. Considering the amount of material under 325 mesh size in the flotation product sample, a portion of ultra-fine particles was removed from the sample via desliming. The tests results on the deslimed sample showed that moisture was further decreased to 16.7 % at the same RW dosage. This is another significant moisture reduction from a baseline of 21%. This moisture reduction in both baseline and with reagent tests is achieved by desliming of the ultrafine fraction that results in more freely draining filter cake capillaries. This also prevented the fines from forming an impermeable layer which might be positioned on the top of the cake.

The screen analysis results also showed that 81% of the ash was in the minus 325 mesh size fraction. This material, which consists of clays and slimes, is hydrophilic in nature and this affects the dewatering performance negatively. The use of flocculants may be another way to compensate the negative effects of the ultrafine particles. The principle of using of flocculants is to bring the fine particles together in the coal slurry and create looser packing in the filter cake. This loose packing results in larger capillaries between the aggregates and a more porous, permeable cake. This allows a rapid drainage of water from these voids, which, in turn, increases the filtration rate. To investigate the effects of flocculants on dewatering kinetics, a series of tests was conducted. To optimize the dosage, various amounts of flocculant were tested, from 5 g/ton to 75g/ton. It was determined that 25 g/ton was the most appropriate flocculant amount when being used alone or in conjunction with other chemicals. The test results showed that in most cases, the addition of flocculant did not improve the final cake moisture, due to the water trapped inside the flocculants. Instead, the dewatering kinetics increased significantly i.e., 30% to 75%.

Another way of lowering the moisture may be to clear out the excessive amount of ash-forming minerals in the fines. To investigate this idea, prior to laboratory dewatering tests, the plant's clean coal product was subjected to another step of flotation using 1 lb/ton (454g/t) RV (in Diesel (1:2)) as collector and 100g/t MIBC as frother. The particle size analysis results showed that the 37.6% of the re-floated sample was minus 325 mesh. Dewatering test results showed after its addition, RW, at about 3 lb/ton, can reduce the moisture to 15.5%, where the baseline was approximately 20%. In addition, the dewatering kinetics of the coal sample was also increased by 50% as a result of the increased hydrophobicity.

As stated earlier, the surfactant adsorption is very important so that it can lead to an increase in solid/liquid contact angle and a decrease in liquid surface tension. For this reason, it is very important to have an effective conditioning system. To investigate and optimize the effectiveness of the dewatering aids and their performances, two series of filtration tests were conducted at various mixing intensities and times. Table 2.6 gives the laboratory test results obtained on the flotation product sample using RW at 3 lb/ton at about 20-24 mm cake thicknesses. As shown, the moisture reduction was substantially improved when mixing intensity was increased. The use of RW at 3 lb/ton reduced the filter cake moisture from baseline value of 30.02% to 25.98% and 25.45% when using low and medium-energy agitation at one minute, respectively. The cake moisture was further reduced to 20.26% moisture when

Table 2.6 Effect of mixing intensity and conditioning time on Mingo Logan flotation product (20 inHg vacuum and 3 lb/ton RW)

Speed Level	Moisture @ specified conditioning time	
	1 min	2 min
0	30.02	30.02
Low	25.98	25.29
Medium	25.45	23.91
High	20.26	18.66

using high-energy agitation at the same reagent dosage of 3 lb/ton. The moisture reduction showed a similar trend, but improved final cake moistures were obtained when the agitation time was increased to two minutes. The final cake moisture was reduced to 23.91% and 18.66% using medium and high-energy conditioning, respectively, at the same dewatering aid dosage. It was found out that two minutes of conditioning at high intensity mixing was optimum. Beyond two minutes of conditioning, there was no change in the residual moisture of the cake. These results clearly demonstrate that the importance of proper conditioning when using the novel dewatering aids.

In the light of the results obtained in the initial laboratory filter tests, using dewatering aids were believed to be a promising method because it could not only remove water but also increased the filtration kinetics. To evaluate the filtration performance in detail as a function of cake formation time, dry cycle time, and specific cake weight in the absence and presence of different types of dewatering aids, a series of vacuum filtration tests was carried out. It is very informative to know the effects of these physical parameters for scale up of using chemicals, optimization of total filtration time, production rate, and, consequently, the final cake moisture. These parameters would also provide general suggestions to meet the desirable filtration efficiencies.

The dewatering tests were carried out using a fixed amount of dewatering aids (3lb/ton) and flocculants (25 g/ton) after the optimum dosage was determined. The tests were conducted at a fixed setup vacuum pressure and pre-measured amount of slurry was added to the Buchner funnel for dewatering tests. The dry cycle times were changed randomly, varying from 14 to 120 seconds. The cake weights were changed by increasing the slurry volume from 50 ml to 300 ml. The cake formation time, dry cycle time, cake weights, amount of filtrate, and solid contents for each test were recorded for production rate calculations. The filter cake production rates were investigated in the absence and presence of dewatering aids and flocculants.

Figure 2.6 is a plot of the final cake moisture percent as a function of pounds of dry solids per hour per square foot, or the production rate. The filtration kinetics without the surfactants and flocculant addition were observed to be very slow, and the residual cake moisture of the cake was found to be in the range of 31-36% at about 60-100 lb/hr/ft^2 throughput, which corresponds to approximately 5-20 mm cake thickness. The filtration time used for the calculations was the sum of cake formation time and dry cycle time. In the absence of dewatering aids, a production rate greater than 100lb/hr/ft^2 was found to be impractical, as achieving a dry cake was no longer possible. There was also segregation of particles in the filter

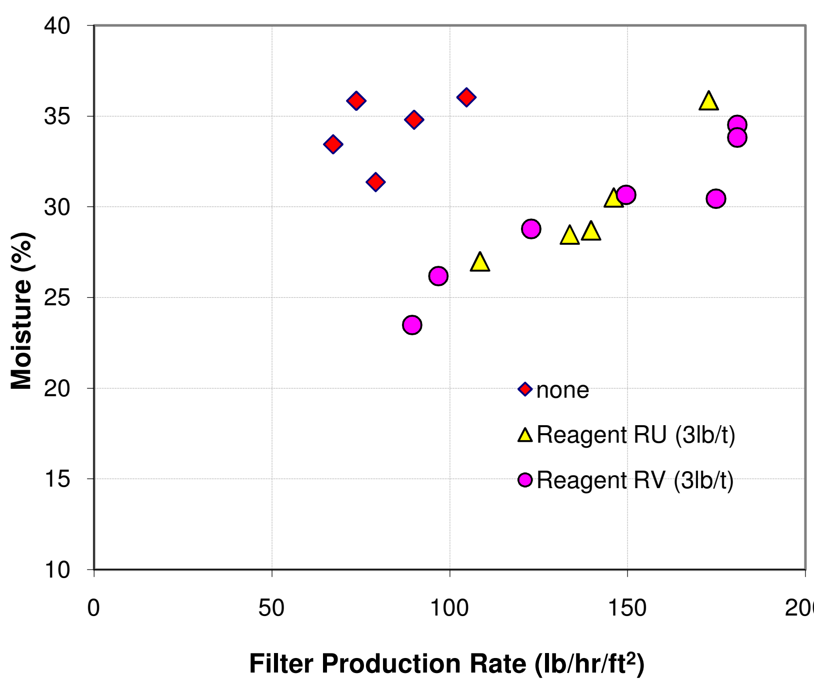


Figure 2.6 Normalized production rate (lb/hr/ft^2) versus cake moisture

cake and a considerable amount of solid loss – up to 10% - in the filtrate. As expected, the cake moisture increased proportionally with increased production rate; however, the results showed that using dewatering aids lead to very significant reductions in the final cake moistures and increase in the filtration rate by several times. The addition of reagents outperformed baseline to a great extent because it gave the same or lower cake moisture at a higher production rate with almost 97% to 99% solid recovery. Also given in the Figure 2.6, RU and RV produced very similar results.

The results obtained with the plants flotation product showed that moisture values as low as 25-30% at improved throughputs – as high as 100-150 lb/hr/ft^2 – could be achieved. When the production rate was further increased at the expense of final cake moisture, it became possible to increase the throughput almost 2.5 times (compared to baseline) in the presence of dewatering aids. The test work demonstrated that the use of dewatering aids provided an outstanding dewatering performance on Mingo Logan's flotation clean coal product and produced significantly lower final cake moisture values, as well as a higher rate of dewatering, and improved the filtration efficiency.

If the other operating parameters are kept constant, the effectiveness in productivity of filtration is related to the time required to complete a full solid-liquid separation cycle, consisting of cake formation time and dry cycle time. The correlation between the cake moisture and total filtration time, normalized with cake weight, is shown in Figure 2.7. Test results showed that when the dewatering aids and flocculant were used in a combined manner, the time consumed for filtration of a given amount of material was reduced, which, in turn, increased the production rate by several factors. There was also significant moisture reduction in the presence of dewatering aids; however, it appears that even when the filtration time was increased in the absence of dewatering aids, the reduction in final cake moisture content was very small. The

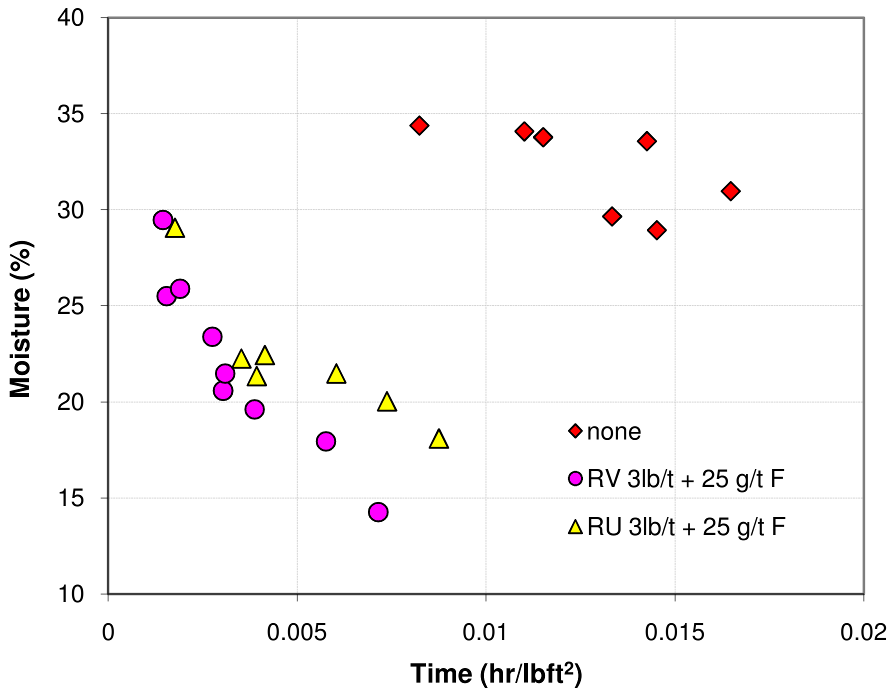


Figure 2.7 Cake moisture versus normalized drying cycle time

results suggest that there is an increase of 100% in solids throughput at a fixed moisture value or 5-10% moisture reduction at approximately the same solid throughput.

Figure 2.8 shows the effect of chemicals on cake formation time and, eventually, the filtration kinetics. A strong correlation was observed between cake formation time and the throughput of the filter. In the presence of RU and RV, even at higher specific cake weights, the formation times were much shorter than what was seen in the baseline tests. Approximately 100-140 seconds were required to produce 3.5-4.5 lb/ft^2 of coal in baseline tests; however, for the same coal production, using dewatering aids, approximately 30-60 seconds was needed. Above 5 lb/ft^2, obtaining a baseline value was impossible because the filter time was impractically prolonged, while 6-7lb/ft^2 of coal could be produced when using dewatering aids. The test data indicated that cake formation time was a significant parameter in throughput and residual cake moisture; however, formation time can be altered by using dewatering aids. In addition, in daily practice, decreased cake formation time allows a longer dry cycle time to complete the full filtration operation, increasing the throughput of the filter at lower final cake moisture.

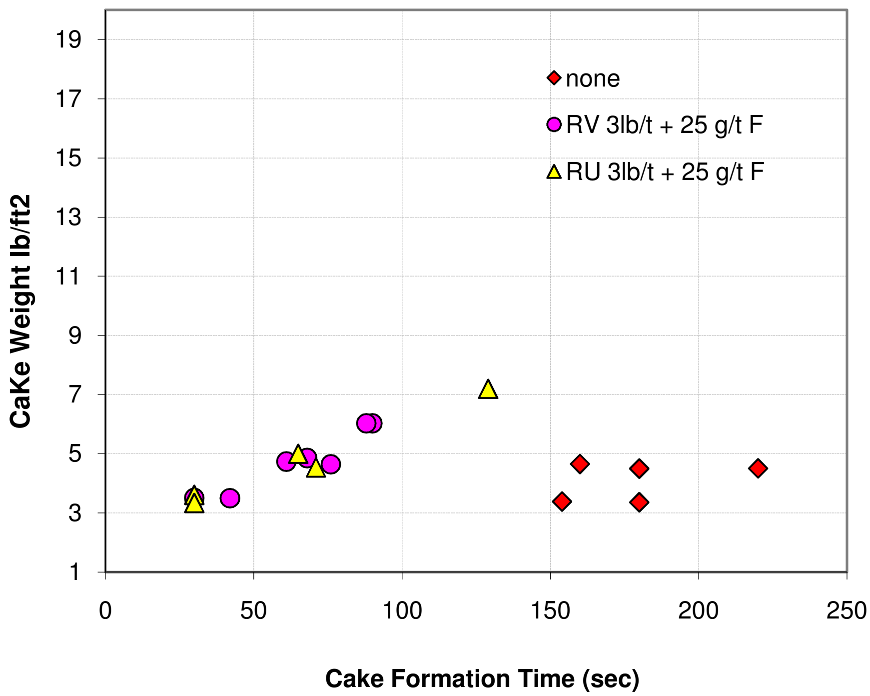


Figure 2.8 Cake moisture versus cake formation time

After completing the tests using the fine clean coal sample from Mingo Logan, a second set of tests were performed using a fine cyclone overflow sample that had been subjected to froth flotation to minimize the adverse effects of high ash content. The tests were conducted by floating the cyclone overflow sample using diesel (0.66 lb/t) as collector and MIBC (0.33 lb/ton) as frother. Both conventional laboratory-scale flotation and bench-scale column flotation equipment were used to produce ash-free clean coal. Both products were tested in the absence and presence of dewatering aids.

The first series of clean coal samples were produced using lab scale Denver cell. The solid concentration of the product was about 15% by weight, while maintaining approximately 44% of minus 325 mesh particle size. The test results on this sample showed that the control cake moisture was 25.31%, and with RW addition at about 3 lb/ton, the final cake moisture was reduced approximately to 19.5% at 7-10 mm cake thickness. Another set of flotation tests was run on the plant's flotation product to discard ash-forming minerals, and the product of the flotation tests was subjected to the same type of dewatering tests. The results indicated that the floated cyclone overflow and re-floated samples, using the laboratory-scale Denver flotation machine, gave considerably better results compared to the plant's as-is flotation product sample. This may be the result of discarding ash-forming minerals and removing ultra-fine particles more efficiently from the coal slurry compared to plant operation.

Using a laboratory scale column flotation unit produced another series of clean coal samples. In this set of tests, the flotation feed/cyclone overflow sample was floated using 0.66 lb/ton diesel as collector and 0.22 lb/ton MIBC as frother. The solid concentration of the column flotation product was about 10% by weight, and approximately 39% of particles were minus 325 mesh size. The dewatering tests were conducted using a 2.5inch-diameter Buchner funnel at 20 inch Hg set-up vacuum pressure with 2 minutes drying cycle time and about 11 mm cake thickness. RU and RV were used as dewatering aids with several of dosages ranging from 0.5-3 lb/ton. Vacuum filtration results on both column and Denver unit clean coal products showed that RV alone was capable of both reducing the moisture and cake formation time significantly; therefore, flocculant addition was unnecessary. The cake formation time for control tests was 94 seconds, whereas it was approximately 10 seconds in the presence of the dewatering aid, RV. The presence of dewatering aids also made it possible to reduce the final cake moisture content of the cake by about 35%. The dewatering test results from the column flotation's clean coal product are represented in Table 2.7. The final cake moisture for control tests was approximately 22.80%. With RV addition, the moisture was reduced to 14.91%, and with RU addition, the cake moisture was lowered to 14.55%, giving approximately 36% overall moisture reduction. When the dry cycle time was lowered to one minute, the baseline was 26.9%, and with about 1 lb/ton RU and RV addition, the moisture was lowered down to 20.44% and 17.95%, respectively.

The influence of cake thickness on final cake moisture was also investigated. To study the effect of dewatering aids on a thicker filter cake, the thickness was increased to approximately 20 mm by increasing the slurry volume that was added to Buchner funnel. As expected, thicker cake resulted in increased baseline moisture and cake formation time. The addition of RU and RV at 3 lb/ton lowered the cake moisture to 17.81% and 15.08%, respectively, from the baseline moisture 24.8% with 2 minutes of dry cycle time. These results correspond to 22% and 33% total moisture reduction. The test results indicated that even with thick cake, dewatering aids showed improvements with final cake moisture.

Another set of tests was performed to investigate the effect of flocculant on the dewatering of the laboratory column flotation product (Table 2.8). The same procedure was applied, and the same types of collector, frother, and dosages were used to produce clean coal.

Table 2.7 Effect of reagent addition on Mingo Logan flotation feed sample (cleaned using bench-scale column).

Reagent Dosage (lb/ton)	Moisture (%)	
	RV	RU
0	22.80	22.80
0.5	16.51	19.20
1	16.99	17.02
3	14.91	14.55

For this series of dewatering tests, the moisture results for the baseline tests were considerably higher than what was obtained previously. This might be attributed to the excessive amount of fine material associated with this fine fraction. The baseline values for the tests were around 27.30% at approximately 10 mm cake thickness. The addition of flocculant alone at about 25 g/ton was capable of reducing the final cake moisture to 20% and decreasing the cake formation time from 120 seconds to 25 seconds at about 7-9 mm cake thickness and 20 inch Hg vacuum pressure. For lower vacuum pressure, about 15 inch Hg, flocculant additions were capable of reducing the moisture to 23% from 28.1%, and significant improvement in dewatering kinetics was present, as well. The cake formation time was lowered significantly to 26 seconds from 310 seconds. The final cake moisture was also lowered down to 15.52% and 17.02% at addition of 5 lb/ton RU and RV, respectively, in conjunction with flocculant addition. These results show that, in terms of producing a cleaner product, column floatation is superior to the Denver test. The results illustrate how dewatering aids can lower the final cake moisture significantly.

g) Laboratory Test Results (Series B – Mixture of 75% Fine and 25% Coarse)

A limited number of dewatering tests were conducted using a mixture of 75% fine coal and 25% coarse coal as a function of various dosages of RW, RU and RV. The solid content of the combination of spiral/flotation product slurry was around 27% by weight and it was subjected to dewatering tests using novel dewatering aids at various dosages. The test results showed that the final cake moisture was about 16% to 17% when dewatering aids were added at about 1lb/ton dosage. The moisture content for the control test was approximately 20% at about 8-11 mm cake thickness. Tests were also performed in the presence of flocculant alone and in conjunction with dewatering aids; however, increases in the dewatering kinetics were close to

Table 2.8 Effect of flocculant and dewatering aid addition on Mingo Logan floation feed sample (cleaned using bench-scale column)

Reagent (lb/ton)	Flocculant (lb/ton)	Moisture (%)	
		RV	RU
0	0	27.30	27.30
0	25	19.99	19.99
0.5	25	18.78	18.55
1	25	19.66	17.41
3	25	19.99	16.55
5	25	17.02	15.52

those obtained with reagents. Even though the preliminary dewatering test results were promising, due to the Plant's request dewatering tests were focused more on flotation or flotation/spiral mixture samples. Thus, no further tests were done.

h) Laboratory Test Results (Series C – 50% Mixture of Fine and Coarse Coal)

The fine and ultra-fine size fraction of a stream to be dewatered is very influential on dewatering, which affects the filtration performance. In the dewatering of such particles, lower moisture percentages are always desirable; however, mechanically, there is a limit to the level of moisture achievable, regardless of operating parameters, such as the length of the filtration time or the applied pressure. In filtration, most of the water is held between and on the surfaces of the particles. Finer particles will create a larger overall surface area and smaller inter-particle openings, which, in turn, keep more water than coarser size distributions do. As the capillary filtration model suggests, a filter cake consists of numerous capillaries with a range of diameters. When the capillary radii are increased, the filtration rate should also increase.[5-8] It can be achieved either by de-sliming or coarse particle addition into the stream or, in this case, into the slurry. As mentioned earlier, when the coal slurry was partially or fully deslimed the drainage of the filter cake was much more efficient, thus lowering the final cake moisture and increasing the kinetics. Blending coarse particles with fine particles will also increase the capillary radii and improve the filtration performance. In some coal preparation plants, this is already applied to increase the dewatering efficiency as an alternative to other means. In fact, currently, Mingo Logan Coal Preparation Plant has been blending spiral clean coal (18 X 100 mesh) with the finer flotation clean product (smaller than 100 mesh) at about 1:1 ratio before filtration in dewatering centrifuges.

The preliminary dewatering test results using RW, RU and RV on one-to-one spiral/flotation blend product showed that significant cake moisture reduction can be obtained using novel dewatering aids by about 20.5-28%, while also increasing the rate of dewatering so much that the formation time could be reduced by as much as 50%. A set of three preliminary tests was conducted to investigate the effectiveness of the dewatering aids (Table 2.9). The dosages used were 1, 3 and 5 lb/ton. The baseline tests produced an average of 19% final cake moisture at about 8mm cake thickness. Even at low dosage, 1lb/ton, when RW, RU and RV was used, the cake moisture was decreased to 15.6%, 14.4%, and 14.8%, respectively, and the filtration kinetics were increased by 30-50 %. Of the reagents and dosages being tested, RU was the most effective for the cake moisture reduction. In this case, the addition of 5 lb/ton RU reduced the cake moisture from 19.00% to 13.7% which corresponds to 20-30 % overall moisture reduction.

Table 2.9 Effect of reagent addition on dewatering of Mingo Logan mixture sample (50% flotation product and 50% spiral product)

Reagent Dosage (lb/ton)	Moisture Content (%)		
	Reagent RW	Reagent RU	Reagent RV
0	19.0	19.0	19.0
1	15.6	14.4	14.8
3	15.7	13.8	14.9
5	15.1	13.7	14.3

Similar dewatering tests were conducted to evaluate the filtration performance with different types of dewatering aids as a function of filtration time and specific cake weight. Filtration tests were done at the same vacuum level; however, the dry cycle times were varied from 15 to 100 seconds. RU and RV were prepared at 2:1 active solvent ratio, and the dosage amount was fixed at 3 lb/ton. The coal slurry was conditioned with the dewatering aids for 2 minutes. Then, flocculant was added at a 25g/ton dosage and conditioned at a very low intensity for 15-20 seconds. Tests were conducted on a pre-measured amount of slurry to differentiate the specific cake weight. This, in turn, varied the cake thickness from 5 to 20 mm. The cake formation time, dry cycle time, and cake weights were recorded for each test. The filter production rate was plotted as pounds of dry solids per hour per square foot, and the filtration time used for filtration rate calculations was the sum of cake formation time and dry cycle time. Figure 2.9 shows the relationship between production rate and cake moisture for the spiral/flotation mixture sample.

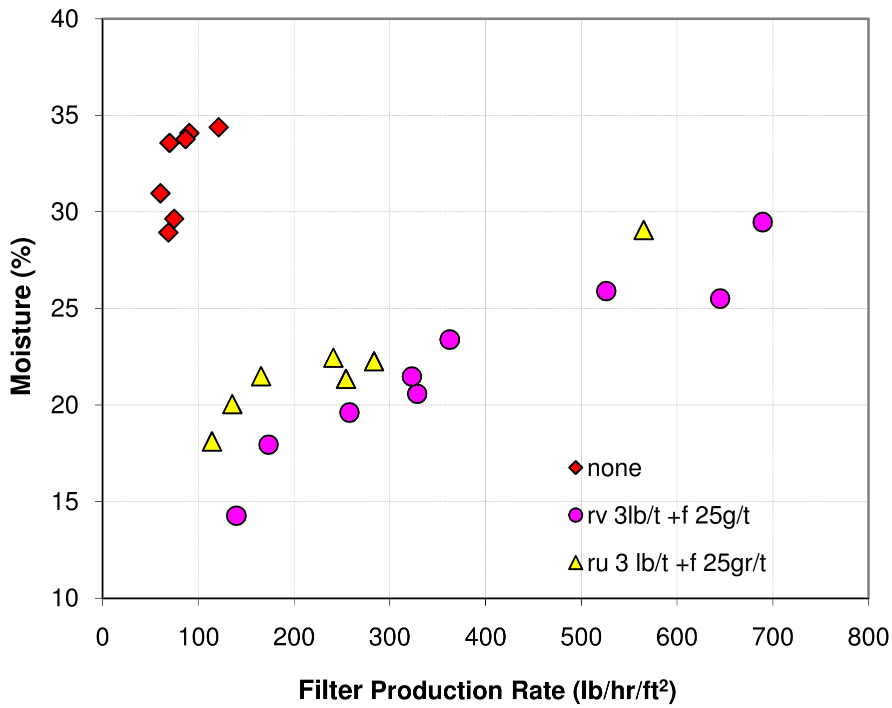


Figure 2.9 Cake moisture versus normalized filter cake production rate

The baseline tests were conducted without reagents and produced a cake of 29-34% moisture at about 5-20 mm thickness, yielding a production rate in the range of 60-120lb/hr/ft^2. Throughput greater than 120 lb/hr/ft^2 was found to be impossible because the cake formation time was being prolonged to impractical limits. The additions of RV and RU, along with 25 g/ton flocculant, surpassed the baseline throughput to a great extent. The usage of RV reduced the final cake moisture to 14-18%, and the usage of RU reduced the final cake moisture to 18-21%. The production rates were also increased by multiples of 1.5 to 2.5, corresponding to 150-190 lb/hr/ft^2. The results also showed that the throughput could be increased at the expense of cake moisture. When the production rate was increased to the 200-400 lb/hr/ft^2 range, the cake moisture increased to 20-25%; however, final cake moistures were still 5-15% lower compared to baseline. Further increases in production rate were also achieved i.e., the 550-650 lb/hr/ft^2 range was achieved at a 25-30% moisture range. It was obvious that the addition of reagent outperformed baseline, and it was possible to increase the production rate with significantly lower final cake moisture values. The results also showed that the use of RV represented better performance than RU because it gave higher filtration rates at the same, or lower, final cake moisture values.

Further data evaluation was carried out on the effect of filtration time on final cake moisture. Figure 2.10 shows the correlation between the cake moisture and total filtration time (normalized with cake weight) in the absence and presence of dewatering aids. In the baseline tests the total filtration time varied between 130 and 360 seconds. On the other hand, the total filtration times in the presence of RU and RV were 65 to 120 seconds and 50 to 118 seconds, respectively. As seen, the dewatering aids that were tested decreased the filtration time, which, in turn, produced higher production rates. It is also noteworthy that these outstanding times were achieved while maintaining very low moisture levels.

The test data also indicated that cake formation time is an important factor in determining dry cycle time and, thus, cake moisture and throughput, as well. Figure 2.11 shows the effects of dewatering aids on cake formation time and, eventually, the filtration kinetics. The formation times were found to be much shorter in the presence of RU and RV. When used with a flocculant, even at higher specific cake weights, the formation times were shorter than those that were recorded during the baseline tests.

As seen in Figure 2.11, only 10-15 seconds were required to produce 4.5 lb/ft^2 of coal when dewatering aids were used; however, in baseline tests, approximately 100-120 seconds were needed for the same coal production. When the cake weight was increased to 7 lb/ft^2, the cake formation time was 20 seconds using dewatering aids, where it was 340 seconds for baseline test. Cake weights above 5 lb/ft^2 were found to be impractical for dewatering in the absence of dewatering aids. Conversely, it was still possible to produce a cake weight of 11 lb/ft^2 while maintaining a short cake formation time. The results showed that the use of dewatering aids decreased the cake formation time by several multiples. The usage of dewatering aids also substantially improved the throughput. This is a strong indication that in the presence of dewatering aids, more material can be treated.

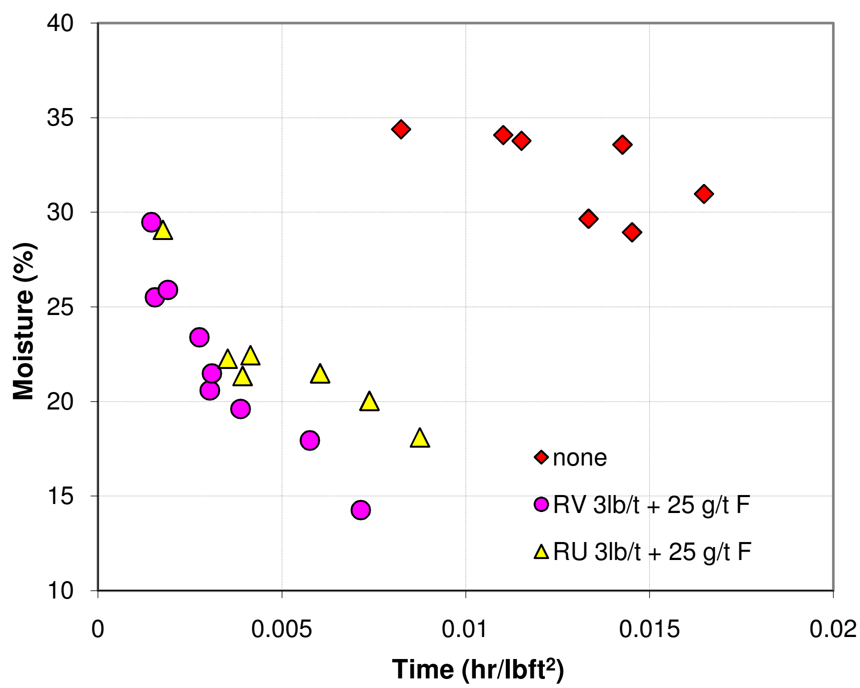


Figure 2.10 Cake moisture versus normalized filtration time

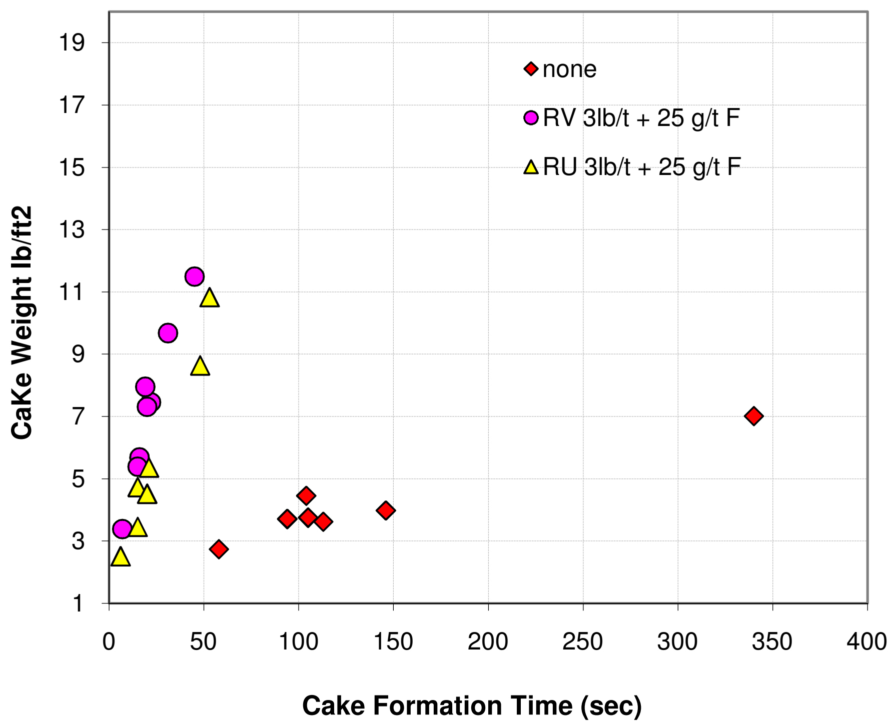


Figure 2.11 Cake moisture versus total cake formation time

i) Pilot-Scale Test Results (Series A – Fine Coal Only)

The pilot-scale dewatering tests were conducted using two feeds intercepted from the plant's main stream (i.e., flotation concentrate and blended products). The first set of tests focused on dewatering of the flotation product, which is very difficult to dewater because of the amount of fine size particles and high ash content. As mentioned earlier, nearly 43% particles of this product was passing 325 mesh (-44 μm) and contained 81% ash. The tests were conducted over a range of filter feed rate that typically varied at 2, 3, 4 and 7 gal/min of coal slurry. The preliminary analysis showed that of the feed rates, 4-7 gal/min were at optimal ranges for the filtration tests and also adequate to produce 0.33 to 1.0 inch cake thicknesses. The pilot-scale

Table 2.10 Effect of RV on Mingo Logan flotation product sample

Filter Time (sec)	Cake Thickness (mm)	Cake Moisture (%)					
		Control	25 g/t Floc	0.5 lb/ton RV	0.5 lb/ton RV*	1.0 lb/ton RV	1.0 lb/ton RV*
184	25	31.49	34.61	27.11	26.13	-	-
120	15	31.59	34.13	25.92	25.95	31.73	23.59
85	10	31.18	-	28.49	-	28.12	24.45
65	8	31.63	36.2	-	-	28.9	26.67

*With Roller

test procedure was same as the mixture sample tests except that the flotation product was intercepted from the distribution box pipeline.

The first set of dewatering tests was conducted with a pilot-scale horizontal belt filter (HBF) at the feed rate of 4gal/min. Baseline tests provided a filter cake with 31~32% moisture. With 25 g/ton flocculant addition to the feed, even though a slight decrease in cake formation time was observed, the cake moisture increased to 34~36%. On the other hand, at 0.5 lb/ton RV addition, the cake moisture was reduced down to approximately 26~28% (see Table 2.10). The results also showed that, in this set of tests, the belt speed had a slight effect on the dewatering when flocculant and reagent were added; however, it did not appear to influence the cake moisture for the control tests (it only caused changes in cake thickness and formation time).

During the tests, it was observed that even though the filter feed rate was kept constant; using RV increased the cake thicknesses. This can be attributed to increased porosity, which in turn creates more permeable cake. This phenomenon also caused a cake-cracking problem. As a result, the vacuum pumps also lost pressure. To overcome this problem, a roller was attached on the belt to press down the cake, presumably to prevent the cracking. The test results clearly showed that the cake moisture was further reduced to 23~26% when a roller was applied to the

Table 2.11 Effect of RV on Mingo Logan flotation product sample (with 5 g/t of flocculant)

Filter Time (sec)	Cake Thickness (mm)	Final Cake Moisture (%)				
		Control	1 lb/ton RV	1 lb/ton RV*	3 lb/ton RV	3 lb/ton RV*
184	25	31.49	28.65	26.32	25.69	26.93
120	15	31.59	29.05	25.61	29.03	25.19
85	10	31.18	28.63	28.01	29.13	27.12
65	8	31.63	24.85	26.81	27.37	28.25

*With Roller

dry cake to help seal the pores inside the cake. Overall, the use of roller resulted in additional moisture reductions from 3-4% to 15% at 0.5 lb/ton and 1 lb/ton of RV addition, respectively.

Table 2.11 shows another series of tests were conducted to investigate the effect of the flocculant addition. The effect of roller was also tested in the presence of dewatering aid. The cake moisture was in the range of 24 to 29% with combined use of 5 g/t of flocculant and 1 lb/ton of RV. Yet again, increased cake porosity and the cracking was a problem; however, when the roller was applied in some cases moisture was lowered.

j) Pilot-Scale Test Results (Series B – 50% Mixture of Fine and Coarse Coal)

The next set of tests was conducting using an equal mixture of fine and coarse coal. The fine coal was obtained from the froth concentrate, while the coarse coal was intercepted from the screen-bowl feed containing approximately 28% solids by weight and 12% ash on dry basis. The tests were conducted over a range of filter feed rate that typically varied from 2, 3, 4 to 7 gal/min of coal slurry. Unlike the flotation product, it was found out that the 3 gal/min feed rate was optimal for the blend filtration tests, which would be sufficient to produce 1/3 to 1 inch cake thicknesses. Table 2.12 shows a summary of the pilot-scale test data obtained at the Mingo Logan plant spiral/flotation product mixture sample using RU at 3lb/ton and flocculant at 25g/t dosage at the feed rate of 3gal/min. Each series of tests were conducted as a function of total filtration time.

This reagent reduced the cake moisture from about 30% down to 21% at 3 lb/ton dosage. Meanwhile, the cake formation time decreased by 20~50% with the addition of RU as dewatering aids. When used alone, it could produce a low-moisture cake (approximately 21~22% moisture), and the cake formation time was significantly reduced down to 14~75 seconds from approximately 85 seconds. The flocculant alone was not capable of reducing the moisture content to the level that RU achieved; although, they could reduce the formation time more significantly. The use of flocculants resulted in loss of vacuum pressure in the pump, an indication of increased cake porosity, but did not help to remove the surface water that was entrapped inside the flocs. However, the combined use of flocculants and RU could achieve a very short cake formation time. When used together with 25 g/t flocculants, RU reduced the cake formation time further to 7~24 seconds, while the final cake moisture remained at almost the same level. In this case, the cake formation time was reduced by 50~75% over the belt speed range under test, and the moisture was reduced from 30-34% down to 21~24%. Test work performed using RU showed that a filter cake with good handling characteristics could also be produced.

Table 2.12 Effect of RU and flocculant on Mingo Logan flotation product sample

Filter Time (sec)	Cake Thickness (mm)	Final Cake Moisture (%)			
		Control	25 g/t Floc	3 lb/ton RU	25 g/t Floc & 3 lb/ton RU
184	25	34.38	26.6	25.05	24.96
120	15	29.62	26.9	21.8	24.66
85	10	30.78	29.1	21.3	18.7
65	8	29.99	28.81	21.5	21.7
Vacuum (Inch Hg)		16-14	11-5	15-11	7-5

In comparison, RV was tested and the results given in Table 2.13 showed that RV was less effective in reducing moisture and increasing the kinetics of dewatering. However, the difference between RU and RV was narrowed when each of these two reagents were used together with low dosage (25 g/t) of flocculants. It was also noticed that in the presence of dewatering aids, change in belt speed made a slight difference in the moisture reduction, especially while operated at lower speeds. When the filter feed rate was fixed, the short retention time of the materials over the vacuum zone was compensated by the thin cake thickness at higher belt speed, and thick cake at lower belt speed.

Table 2.13 Effect of RV and flocculant on Mingo Logan flotation product sample

Filter Time (sec)	Cake Thickness (mm)	Final Cake Moisture (%)			
		Control	25 g/t Floc	3 lb/ton RV	25 g/t Floc & 3 lb/ton RV
184	25	34.38	26.6	28.11	24.56
120	15	29.62	26.9	28.9	24.74
85	10	30.78	29.1	29.78	22.63
65	8	29.99	28.81	26.34	21.4
Vacuum (Inch Hg)		16-14	11-5	16-15	8-4

2.3.3. Coal Clean Site

a) Site Descripton

The Coal Clean Panther Preparation Plant is located in Dry Branch, approximately 15 miles south of Charleston, West Virginia. The Panther Preparation Plant currently processes 63 mm x 0 raw coal for ultimate use in the metallurgical and steam markets. The minus 325 mesh size fraction material reports to the discard stream without any type of processing method. However, if this size fraction is recovered and dewatered, loss of valuable source could be turned into profit. For this reason the plant management looked for alternatives for the recovery of the minus 325 mesh coal that is presently discarded to refuse. To evaluate the feasibility of the recovery and dewatering of this size fraction, extensive pilot scale flotation and dewatering tests were conducted. A number of tests with pilot scale centrifuge unit were conducted for comparison reasons.

b) Reagents

In this investigation, majority of the pilot scale dewatering tests were performed with varying amounts of three types of dewatering aids, namely RW, RU and RA. Diesel was used as solvent at one to two (1:2) ratio (dewatering aid: solvent). Each of the reagents were tested over a range of dosages typically ranging from 0 to 20 pounds per ton for the ratios of the ultra-fine and fine products in the feed to the Filter Module and the Centrifuge Module.

c) Coal Samples

For this particular test site, pilot-scale dewatering tests were conducted on feeds comprised of different mixtures of coarse and fine coal feeds. These included (i) 100 mesh x 0 feed stream from the overflow of the primary classifying cyclones, (ii) 325 mesh x 0 feed raw stream, and (iii) blends of 100 mesh x 325 and minus 325 mesh product from the flotation column. The

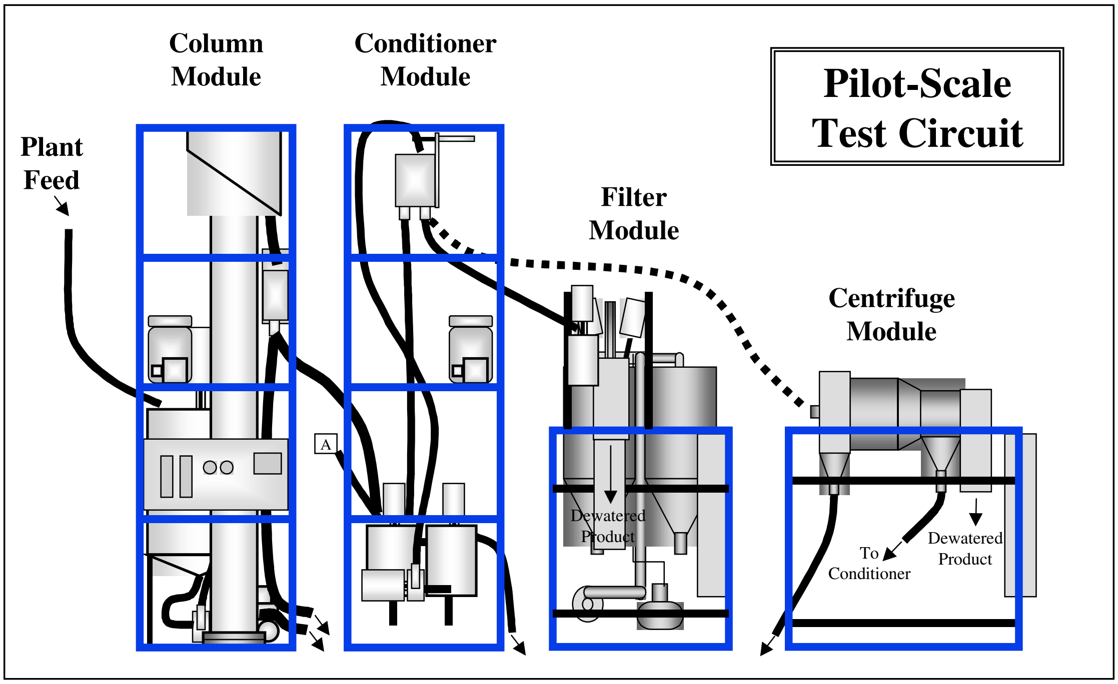


Figure 2.12 Configuration of the mobile unit test modules evaluated at the Panther Preparation Plant site

blends were established at ratios of 1:1 and 3:1 of minus 325 mesh material to the 100 x 325 mesh product.

d) Pilot-Scale Procedures

The column flotation, conditioner, disc filter, and centrifuge modules were set up at the Coal Clean Plant to accommodate the pilot-scale testing program. A total of 115 tests were run over a one month period to establish optimum conditions for the proposed plant upgrades that were under consideration to accommodate the proposed POC-scale test circuit. The arrangement of the modules for the tests is presented in Figure 2.12. For the majority of the test work, the plant supplied a relatively consistent feed stream of minus 325 mesh material to the Column Module from the overflow of the secondary classifying cyclones. The secondary classifying cyclones were fed from the overflow of the primary classifying cyclones which were separating a

16 mesh x 0 slurry stream at a nominal size of 100 mesh. The underflow from the secondary classifying cyclones reported to a bank of conventional flotation cells and the overflow was discarded to the refuse thickener. A series of tests was also conducted on the 100 mesh x 0 feed stream from the overflow of the primary classifying cyclones. The minus 325 mesh raw feed was cleaned in the Column Module, with the clean coal routed to the Conditioner Module. The conditioned slurry was then dewatered in either the Filter Module or the Centrifuge Module.

The Panther Preparation Plant offered a unique opportunity to conduct a series of tests with various blends of 100 mesh x 325 mesh conventional flotation product added to the minus 325 mesh product from the Column Module. The projected blends were established at ratios of 1:1 and 3:1 of minus 325 mesh material to the 100 x 325 mesh product. Maintaining the blend at the various ratios proved somewhat difficult due to the changing flows for the products, but most of the tests were conducted within 5% of the projected blend. Although the Filter Module and the Centrifuge Module were not operated at the same time, the tests included almost the same sweep of reagent tests for both modules.

e) Pilot-Scale Test Results

Table 2.14 presents the results for dewatering tests with various reagents while dewatering the minus 325 mesh product in the Filter Module. The sample was first floated using a mixture of Nalco 01DU113 (0.25 unit) + 01DU145 (1.0 unit) as collector and Nalco-948 as frother (9 units) in a flotation column. The solids content in the filter feed ranged from 4-5% by weight. For dewatering tests, RW, RU and RA were used as dewatering aids at various dosages. The vacuum pressure was approximately 20.5 to 23.0 inches Hg; and the cake thickness was around 6-7 mm. The results indicate that the baseline moisture content of the filter cake (with no reagent) was very consistent and it was approximately 29.0%. This value was reduced to 27.5%,

Table 2.14 Coal Clean coal (minus 325 mesh) dewatered using RW, RU, and RA dispersed in Nalco 01DW110

Reagent Dosage (lb/t)	Moisture Content (%)		
	RW (1:4 Ratio)	RU (1:2 Ratio)	RA (1:2 Ratio)
0.00	29.10		
3.08	27.01		
6.08	26.04		
9.75	24.66		
0.00		29.28	
2.80		27.28	
5.48		24.71	
9.43		23.42	
0.00			29.05
1.62			28.47
2.19			27.49
7.50			27.49

24.7% and 23.4% with RA, RW, and RU, respectively. As such, RU provided the best overall performance for dewatering of the minus 325 mesh product. Considering the amount of the minus 325 size fraction in this feed, RU was capable of reducing the final cake moisture by 20%.

Table 2.15 presents the results for dewatering tests using the Filter Module for a blend of the minus 325 mesh product with the 100 x 325 mesh product from the conventional flotation cells at a ratio of 1:1. For flotation tests, same procedure and collector and frother dosages were employed. As shown, the addition of the coarser material had a dramatic effect on the moisture content of the product. With no reagent added, the addition of coarser material reduced the baseline moisture from 29% (Table 77) to 20% (Table 78). As the reagent dosage was increased, the 20% moisture content was further reduced to 17.8% with RW and to 17.6% with RU.

Table 2.16 presents the results for the dewatering tests using the Filter Module for a blend of the minus 325 mesh product with the 100 mesh x 325 mesh product from the conventional flotation cells at a ratio of 3:1. The solids content in the filter feed ranged from 5% to 6%. As

Table 2.15. Coal Clean coal (50% minus 325 mesh and 50% 100x325 mesh) dewatered using RW and RU dispersed in Nalco O1DW110

.

Reagent Dosage (lb/t)	Moisture Content (%)	
	RW (1:2 Ratio)	RU (1:2 Ratio)
0.00	19.98	
1.98	19.82	
4.47	17.76	
0.00		19.98
2.80		18.26
5.40		17.93
7.95		17.65

seen, when the amount of the minus 325 size material was increased, the baseline moisture value was also increased. The results, as expected, lie between those obtained in the two test series noted above. Of the dewatering aids, RU was the most effective and capable of reducing the cake moisture from 24.78% to 20.77 % which correspond to 17 % overall moisture reduction.

Table 2.17 presents the results for dewatering tests for a minus 325 mesh product in the Filter Module with the filter disc speeds ranging from 4 to 1.5 min/rev. The sample was floated using a mixture of 01DU110 as collector and Nalco 01DU009 as frother in a flotation column. The solids content in the filter feed ranged from 9-10% by weight. The product rate increased from 43.78 to 59.18 lb/h with only a slight increase in the moisture content from 27.5 to 28.5% when using RU at approximately 2.0 lb/ton.

Table 2.16 presents the results for pilot scale dewatering tests using RW for a blend of the minus 325 mesh product with the 100 x 325 mesh product at a ratio of 1:1 using the Centrifuge Module. The sample was floated using mixture of 01DU110 as collector and Nalco 01DU009 as frother in a flotation column. The solids content in the centrifuge feed ranged from 9-10% by weight. The 150 mm diameter centrifuge operated at approximately 480 rpm with a differential of 95:1 for the conveyor. The moisture of the product increased from 33.2 to 34.5% as the solids yield increased from 87.2 to 93.4% with the increase in reagent dosage. The higher

Table 2.16 Coal Clean coal (75% minus 325 mesh and 25% 100x325 mesh) dewatered using RW, RU and RA dispersed in Nalco 01DW110

Reagent Dosage (lb/t)	Moisture Content (%)		
	RW (1:2 Ratio)	RU (1:2 Ratio)	RA (1:2 Ratio)
0.00	24.78		
2	23.57		
4.36	24.83		
7.4	22.49		
0.00		24.78	
1.85		23.01	
3.78		22.55	
7.7		20.77	
0.00			24.78
2.0			23.00
3.56			23.82
8.47			23.68

Table 2.17 Effect of disc speed on product rate and moisture content for Coal Clean coal (minus 325 mesh) from the filter module

Disc Rotation Speed (min/rev)	Filter Cake Production (lb/hr dry)	Moisture Content (%)
4.0	43.78	27.49
3.0	44.86	27.76
2.0	52.94	28.63
1.5	59.16	28.47

Table 2.18 Effect of Reagent 01DW133 on Coal Clean coal (50% minus 325 mesh and 50% 100x325 mesh) from the centrifuge module

Reagent Dosage (lb/t)	Nalco 01DW133 in 01DU110 (1:2 Ratio)	
	Moisture Content (%)	Solids Yield (%)
0.00	33.24	87.20
2.31	33.94	89.60
4.51	33.36	90.30
6.16	33.86	86.00

moisture and improved yield appear to be mostly due to an increase in the recovery of the ultra-fine material.

Table 2.18 presents the results for dewatering tests using Reagent 01DW133 for a blend of the minus 325 mesh product with the 100 x 325 mesh product at a ratio of 1:1 using the Centrifuge Module. The sample was again floated using a mixture of 01DU110 as collector and Nalco 01DU009 as frother in a flotation column. The solids content in the centrifuge feed ranged from 9-10%. The moisture of the product increased slightly from 33.2 to 33.8% and the solids yield increased from 87.2 to 90.3% with the increase in reagent dosage. Once again, the increases in moisture and yield were attributed to increases in the recovery of ultra-fine material.

During the continuous test work conducted at the Panther Preparation Plant site, a series of timed samples were collected periodically from various points around the pilot plant so that complete mass balances could be established for the different unit operations. The samples included representative splits for the column cell (feed, product and tails), filter (feed, product and filtrate), and the screen bowl centrifuge (feed, product, effluent and drain). The series of tests for the minus 325 mesh feed showed that the 300 mm (1 ft) diameter Column Module produced an average of 68.2 lb/hr of concentrate. Depending on the particular operating conditions, the clean coal capacity ranged from a low of 37.4 lb/hr to a high of 127.6 lb/hr. The grade of the concentrate averaged 10.4% ash and ranged from 3% to18% ash, while the combustible recovery averaged 83.0% and ranged from 35.7% to 92.9%. The feed ash averaged 40.8% ash and ranged from 32.6% to 53.6% ash. The 0.2 m^2 (2 ft^2) Filter Module produced from 33 to 59.4 lb/hr of cake and the 150 mm diameter Centrifuge Module produced from 10.12 to 52.36 lb/hr of product from feed streams containing between 2.1-16.7% solids by weight.

2.3.4. Concord Site

a) Site Description

The Concord Coal plant is located near Birmingham, Alabama. This facility processed 50 mm x 0 coal for the metallurgical and steam markets, with a design plant feed rate of 1,000 tph and typical clean coal yields of 55%-60%. The intermediate/fine coal circuit consists of primary classifying cyclones (PCC), spirals, secondary classifying cyclones (SCC), froth flotation, and screen bowl centrifuges. The overflow from the PCCs is fed to the SCCs; the SCC underflow is the feed stream to flotation (4 banks of five 180-ft^3 cells), while the SCC overflow is piped to a refuse thickener. The coal being processed is very soft and fine in size consist and the feed to the flotation cells can be as much as twice that of the design flowsheet rate (design 54 t/hr vs. actual 80-100 t/hr). The feed to the flotation cells is approximately 80% minus 325 mesh (0.045 mm). The flotation and spiral clean-coal products are combined and then dewatered via four 44" x 132" screen bowl centrifuges with a total design feed rate of 2,200 gal/min and 242 t/hr.

The primary objectives of the test program were to determine whether (i) a thick and low-moisture filter cake and (ii) a filter cake with good material handling characteristics could be produced from the minus 100 mesh flotation feed stream (primary cyclone overflow) that is currently processed in conventional flotation cells and centrifuges at this plant. To meet these objectives, extensive laboratory and pilot scale flotation and dewatering test program was conducted at Virginia Tech Mineral Processing Facility in Virginia, and at the Concord coal preparation plant in Alabama. The laboratory tests included the performance evaluation of various types and dosages of dewatering aids to collect dewatering data that could be used in direct support of pilot-scale test work. The data obtained from the laboratory tests were used to provide a technical guidance for the pilot scale test program.

b) Reagents

The laboratory and pilot scale tests included both the flotation and dewatering tests. For laboratory flotation tests, diesel and MIBC were used as collector and frother, respectively. The floated sample was then subjected to dewatering tests using RW3 and RV. For pilot-scale flotation tests RW, RU, RV and diesel as collectors and Nalco DU009 as frother were used. The same reagents were used as the dewatering aids in filter tests. Each of the reagents was tested over a range of dosages that typically varied from 0 to10 lb/ton of dry coal. As mentioned elsewhere, these dewatering aids are insoluble in water. For all the tests, flotation and dewatering, diesel was used as solvent at one to two (1:2) dewatering aid-to-diesel ratio.

c) Coal Samples

The coal slurry samples for the flotation and dewatering tests were collected from the minus 100 mesh flotation feed stream (primary cyclone overflow) that was processed. The same coal slurry feed was used for the pilot-scale flotation and dewatering tests.

d) Laboratory Procedures

The flotation feed sample (minus 100 mesh) from the Concord plant was first floated in a laboratory mechanical flotation cell using 0.66 lb/t of diesel and 0.33 lb/t of MIBC to remove ash-forming minerals. The floated product was then subjected to dewatering tests at about 13.5% solids by weight. In these tests, RV and RW3 (both diluted to 33.3% in solvent) were used as dewatering aids. Immediately prior to each filter test, a known volume of slurry was conditioned for 5 minutes with the reagent in a mechanical shaker and then poured into a

Buchner funnel before applying vacuum of 20 inches Hg. A constant drying cycle time of 2 minutes was used in all tests. The thickness of the filter cake varied from 7 to 9 mm.

e) Pilot-Scale Procedures

The on-site test work was conducted using the Column, Conditioner, and Filter Modules. In each test, the cyclone overflow from the plant was first upgraded using the Column Module (a 305 mm or 12-inch diameter Microcel column) using diesel and RW, RU and RV as collectors and Nalco DU009 as frother. The plant feed was reasonably consistent for most of the test work and contained 3.5-5.5% solids by weight and 23.3-26.5% ash. The column produced a clean coal product with 7-11% ash and 84-87% combustible recovery, depending on the reagent type and dosage used. The froth product was then dewatered using the Filter Module (i.e., 0.186 m^2 (2 ft^2) filter area, 10 sector Peterson disc filter). The same reagents listed above, namely RW, RU and RV, were also used as the dewatering aids in the filter tests. Each of the dewatering aids were tested over a range of dosages that typically varied from 0 to 10 lb/ton of dry coal. Timed samples were collected periodically from various points around the test modules to establish typical material balances and reagent addition rates for this particular coal.

f) Laboratory Test Results

Several series of laboratory dewatering tests were conducted to collect data that could be used as guidance for the pilot scale dewatering tests. Table 2.19 shows the laboratory test results obtained on the Concord Plant flotation feed sample using RV and RW3 as the dewatering aids. A 62.5 mm diameter Buchner funnel was used. The solid content of the flotation feed sample was 6%. It was increased up to 13.5% after flotation. 0.66 lb/t diesel and 0.33 lb/t MIBC was used during flotation. Vacuum setup point was 20 inches Hg. Cake thickness: 7-9 mm. Conditioning time: 5 minutes; drying cycle time: 2 minutes. The volume of the slurry was 100

Table 2.19 Effect of reagent addition on Concord flotation feed (100 mesh x 0) using RV and RW3

Reagent Dosage (lb/t)	Moisture Content (%)	
	RV	RW3
0	24.6	24.6
0.5	21.3	23.2
1.5	19.6	21.0
2.5	18.1	19.4
10.0	18.2	18.6

ml. As shown, the moisture content of the filter cake was reduced as the reagent dosage increased. At dosages of 0.5 lb/t and 2.5 lb/t of RV, the moisture contents of the filter cake were reduced from 24.6% down to 21.3% and 18.1%, respectively. Similar results were obtained when RW3 was used as the dewatering aid. These values correspond to a 10-26% moisture reduction in the filter product.

g) Pilot-Scale Test Results

The first series of dewatering tests were conducted at various levels of vacuum pressures applied. The feed slurry was first floated using diesel collector and Nalco DU009 frother in flotation column. As shown in Table 2.20, there was a strong correlation between the cake moisture and vacuum pressure. The results showed that when the vacuum pressure was increased the cake moisture was decreased from 31.12% down to 24.18%.

Table 2.21 provides an overall summary of the pilot-scale test data obtained at the Concord plant. In the first series of tests, RW, RU and RV were evaluated over a wide range of reagent dosages at a constant disc speed of 4 min/rev. The test data show that RW was the most effective of the three dewatering aids (Table 2.21a). This reagent reduced the cake moisture from 25.5% down to 20.2% at the highest reagent dosage of 2.7 lb/t. RU, which was rather less effective than RW, was not capable of reducing the moisture content to less than 21.5% (Table 2.21b). However, this moisture level was achieved at a very low reagent dosage level of just 0.88 lb/t. In fact, higher dosages of RU did not appear to be as effective in reducing moisture in this particular series of tests. RV was generally the least effective in reducing moisture (Table 2.21c).

Table 2.20 Pilot-scale test results obtained on the Concord flotation feed sample (100 mesh x 0) using various vacuum pressures

Vacuum Pressure (Inch Hg)	Moisture Content (%)
10	31.12
15	26.83
20	24.18

Although less effective in reducing moisture, RV generally provided the best overall cake thicknesses (up to 12 mm) when compared to RW and RU. The thick cakes produced using RV also possessed the best material handling characteristics and were cited by plant personnel as the most suitable for their particular needs. Therefore, several additional tests were conducted using RV to determine whether lower moistures could be achieved by adding the reagent directly to the flotation column feed in place of the diesel collector. As shown in Table 2.22, this strategy significantly improved the moisture reduction and greatly reduced the total reagent requirement. More importantly, the lower moistures were obtained at relatively large cake thicknesses (i.e., 9-10 mm). In one test run, the cake moisture was reduced from 25.1% down to 20.9% (a moisture reduction of 16.5%) while maintaining a cake thickness of 9 mm. The total reagent dosage

Table 2.21 Pilot-scale test results obtained on the Concord flotation feed sample (100 mesh x 0) using various reagent combinations

Flotation Diesel (lb/t)	Disc Filtration Reagent Type	Disc Filtration Dosage (lb/t)	Feed Solids (%)	Cake Thickness (mm)	Cake Moisture (%)
(a)					
1	None	0.0	4.22	2.5	25.47
1	1:2 RW/Diesel	0.66	6.01	5.5	23.10
1	1:2 RW/Diesel	1.1	6.01	4.5	21.70
1	1:2 RW/Diesel	5.94	6.01	5.5	20.24
(b)					
0.84	None	0.0	6.24	2.5	24.63
0.84	1:2 RU/Diesel	0.88	6.24	5.0	21.50
0.84	1:2 RU/Diesel	2.42	6.24	7.5	21.89
0.84	1:2 RU/Diesel	4.18	6.24	4.0	22.79
(c)					
1	None	0.0	4.22	2.5	25.47
1	1:2 RV/Diesel	1.98	6.01	6.0	23.97
1	1:2 RV/Diesel	5.28	6.01	8.0	25.49
1	1:2 RV/Diesel	9.68	6.01	12.0	25.79
1	1:2 RV/Diesel	11.44	6.01	8.0	22.83

*4 min/rev disc speed

required to achieve this moisture was just over 2.2 lb/t (i.e., 0.98 lb/t for flotation and 1.32 lb/t for filtration). Further improvements in moisture content were obtained by reducing the disc speed from 4 min/rev to 6 min/rev (Table 2.23). The addition of reagent to the flotation feed combined with the slower disc speed made it possible to achieve moistures of 20.3%, 19.3% and 18.7% for dewatering aids RV, RW and RU, respectively. These low moisture values were obtained at very low total reagent dosages of less than 1.1 lb/t and with relatively large cake thicknesses of 12-15 mm.

Table 2.22 Pilot-scale test results obtained on the Concord flotation feed sample (100 mesh x 0) using various reagent combinations

Column Flotation		Disc Filtration		Feed	Cake	Cake
Reagent Type	Dosage (lb/t)	Reagent Type	Dosage (lb/t)	Solids (%)	Thickness (mm)	Moisture (%)
Diesel	0.95	None	0.0	5.23	2.5	25.05
1:10 RV/DU009	0.51	None	0.0	16.12	9.0	23.27
1:2 RV/Diesel	0.97	None	0.0	7.10	6.0	21.04
1:2 RV/Diesel	0.97	1:2 RV/Diesel	0.66	5.71	10.0	21.30
1:2 RV/Diesel	0.97	1:2 RV/Diesel	1.32	8.09	9.0	20.92
1:2 RV/Diesel	0.97	1:2 RV/Diesel	2.64	6.46	7.0	21.36

*4 min/rev disc speed

Table 2.23 Pilot-scale test results obtained on the Concord flotation feed sample (100 mesh x 0) using various reagent combinations

Column Flotation		Disc Filtration		Feed	Cake	Cake
Reagent Type	Dosage (lb/t)	Reagent Type	Dosage (lb/t)	Solids (%)	Thickness (mm)	Moisture (%)
Diesel	0.86	None	0.0	8.55	3.0	21.78
1:2 RV/Diesel	1.3	None	0.0	7.34	12-15	20.37
1:2 RW/Diesel	0.92	None	0.0	11.42	12-15	19.25
1:2 RU/Diesel	0.99	None	0.0	13.07	12-15	18.66

*6 min/rev disc speed

2.3.5. Buchanan Site

a) Site Description

Consolidation Coal Company's Buchanan Mine #1 is an underground coal mine located two miles south of Route 460, adjacent to State Route 632, at Mavisdale, Buchanan County, Virginia. Consol Energy Inc., located in Pittsburgh, Pennsylvania, is the parent company of Consolidation Coal Company. The Buchanan preparation plant processes approximately 5 million tons of 37.5 mm x 0 raw coal (2006) from the Pocahontas 3 Seam for use in the metallurgical and steam markets.

The objectives of this study were to (i) identify the best possible reagents and combinations thereof for this specific coal and (ii) identify the conditions under which a given dewatering aid can give the best performance. To meet the objectives, laboratory and pilot scale tests were conducted to evaluate the performance of various types and dosages of dewatering aids.

b) Reagents

The investigation, both laboratory and pilot-scale, was performed with varying amounts of three types of dewatering aids, namely RW, RU, and RV. Since these dewatering aids are insoluble in water, they were dissolved in a solvent. In both laboratory and pilot scale experiments diesel was used as the solvent. The ratio of reagents-to-solvent was optimized in previous studies by varying the individual dosages (0.5 to 3lb/ton), while maintaining the total blend dosage constant. For this test program, the optimum combination for a given dewatering aid and solvent is one to two (1:2) dewatering aid-to-diesel ratio.

c) Coal Samples

The test work was conducted on different samples taken from Consolidation Coal Corporation's Buchanan Preparation Plant in Mavisdale, Virginia. These samples included (i) a flotation feed sample, (ii) a grab sample of current flotation product, and (iii) a slip-stream sample of filter feed (all taken on various dates).

d) Laboratory Procedures

Prior to experiments sieve analysis was conducted for each sample by wet screening using 600, 300, 150, 75 and 45 micron sieves. Dewatering tests were mostly conducted on filter feed and flotation product samples. The solid concentration of Buchanan's flotation product was about 25% by weight while maintaining approximately 25-30% of minus 45 micron material. Table 2.24 shows the sieve analysis results on flotation product sample. The flotation product sample was occasionally mixed with a portion of the spiral product at the plant at a ratio of 1:1 to have better dewatering kinetics. Table 2.25 shows the sieve analysis results of spiral/flotation product slurry collected from the plant.

For the laboratory-scale batch dewatering tests, the samples were collected in 5-gallon buckets and to be able to receive a representative sample, samples were homogenized by mixer. When the plant flotation feed sample was used in the dewatering tests, the sample was first floated in a laboratory mechanical flotation cell using 0.66 lb/t of kerosene and 0.33 lb/t of MIBC to remove ash-forming minerals. The floated product was then subjected to the dewatering tests at about 20% solids by weight. Immediately prior to each filter test, a known volume of slurry (whether flotation product or mixture sample) was conditioned with the reagent in a mechanical shaker and then poured into a Buchner funnel before applying vacuum. The dewatering aids RW and RU, diluted to 33.3% in solvent, were used in these tests. The

Table 2.24 Screen analysis of the Buchanan flotation product used for dewatering tests

Particle Size (Mesh)	Pipe 1 Weight (%)	Pipe 2 Weight (%)	Pipe 3 Weight (%)
Plus 28	3.6	3.4	7.7
28x45	19.0	15.8	27.3
45x100	23.0	18.0	24.1
100x200	14.9	13.2	13.2
200x325	8.1	25.3	6.9
Minus 325	31.4	24.3	20.7

following conditions were kept constant during the tests: 20 inches Hg of vacuum, 2 minutes of drying cycle time, 10-15 mm of cake thickness, 100 ml volume of feed slurry and 5 minutes of conditioning time.

e) Pilot-Scale Procedures

In the pilot-scale dewatering tests, the Conditioner Module and Filter Module were required since the feed slurry for the tests was supplied directly from the Buchanan Preparation Plant. The pilot-scale disc filter tests were conducted on flotation product samples.

Table 2.25 Screen analysis of the Buchanan filter feed used for dewatering tests

Particle Size (Mesh)	Pipe 1 Weight (%)	Pipe 2 Weight (%)	Pipe 3 Weight (%)
Plus 28	6.0	6.3	5.5
28x45	23.4	23.3	21.8
45x100	21.7	22.5	19.9
100x200	14.7	14.8	14.6
200x325	8.2	6.9	8.9
Minus 325	26.0	26.2	29.3

Table 2.26. Effect of reagent addition on dewatering of Buchanan's filter feed

Reagent Dosage (lb/t)	Moisture Content (%)	
	RW	RU
0	18.1	18.1
1	17.58	17.5
3	15.2	16.78
5	14.9	16.58

f) Laboratory Test Results

Table 2.27 gives the laboratory test results obtained on the Buchanan plant flotation product using RU and RW as the dewatering aids at 20 inches Hg vacuum. As shown, the moisture content in the filter cake was reduced with increasing reagent addition. At RU additions of 1 and 5 lb/t, the moisture contents of the cake were reduced from 17.6% to 16.1% and 14.4%, respectively. Similar results were obtained when RW was used as dewatering aid. These values corresponded to a 15-20% moisture reduction in the filter product.

Similar dewatering tests were conducted on the filter feed sample (which contains approximately 10 g/t of Nalco 9806 polymer flocculant as the dewatering aid) using RW and RU as dewatering aids. Results for the filter feed sample are summarized in Table 2. The results show that the moisture content of the filter product again decreases with increasing RW and RU additions from 1 to 5 lb/ton. In this case, the addition of 5 lb/ton of RW reduced the cake moisture from 18.1 to 14.9%, giving a percentage moisture reduction of about 20%. The

Table 2.27. Effect of reagent addition on dewatering Buchanan's flotation product

Reagent Dosage (lb/t)	Moisture Content (%)	
	RW	RU
0	17.6	17.6
1	16.2	16.1
3	16.2	14.9
5	15.0	14.4

moisture reduction is quite similar to that obtained for the flotation product, except that the moisture content of the filter cake product obtained using RU was almost 2 percentage units lower, i.e., 14.4% vs. 16.6% moisture in the filter cake (see Table 2.27 and Table 2.26). The reasons for the relatively poorer behavior of RU may be related to the presence of flocculant in this particular sample. Apparently, the polymer flocculant has an adverse effect on the performance of RU during dewatering. Those poor results are also due to the Ca^{2+} ions present in Buchanan plant water.

Two series of laboratory filtration tests were conducted to determine the effects of agitation intensity on the dewatering performance of the Buchanan filter feed. The first series of tests were conducted using a laboratory shaker to condition the feed samples. The shaker was a low-energy conditioner that uses reciprocating motion (similar to wrist-action shaking) to gently mix slurry contained in a 100-ml glass conditioning flask. A second series of tests were conducted using a 100-ml Plexiglas cell equipped with a three-blade propeller-type mixer at 1000 rpm. The rotary mixer provided an intense agitation that is necessary for high-energy conditioning. The feed slurry was conditioned for 5 minutes in both series of tests.

As shown in Table 2.28, the moisture reduction was substantially improved when high-energy conditioning was used. For example, the use of 5 lb/ton of dewatering aid reduced the filter cake moisture from a baseline value of 18.2% (no reagent added) down to 14.6% when using low-energy agitation. The cake moisture was further reduced to 11.7% moisture when high-energy agitation was used at the same reagent dosage of 5 lb/ton. Similar results were obtained using RW as the dewatering aid. With this reagent, the final cake moisture improved from 14.9% to 13.4% at 2 lb/t of dewatering aid and from 14.3% to 13.1% at 5 lb/t of dewatering aid. These results clearly demonstrate the importance of proper conditioning when using the novel dewatering reagents. The

Table 2.28. Effect of mixing intensity on dewatering of Buchanan's flotation product

Reagent Dosage (lb/t)	Moisture Content (%)			
	RU		RW	
	High Energy Mixing	Low Energy Mixing	High Energy Mixing	Low Energy Mixing
0	18.2	18.2	18.2	18.2
1	15.4	17.1	13.5	15.5
2	13.2	14.9	13.4	14.9
3	11.9	14.6	13.1	14.7
5	11.7	14.6	13.1	14.3

results also indicate that the high-intensity conditioning increases the adsorption density of dewatering reagents; as a result, lower moisture filter cake product can be obtained.

g) Pilot-Scale Test Results

Table 2.29 gives the results of pilot scale dewatering tests which were obtained using various dewatering reagents at different addition rates. The data indicate the moisture content of the filter cake decreased from a baseline (no reagent) value of 16.9% to 14.5% with 5 lb/ton of RU and to 15.0% with RW. Likewise, Table 2.27 gives the laboratory test results obtained on the Buchanan plant flotation product using RU and RW as the dewatering aids at 20 inches Hg vacuum. As it can be seen from the table, the moisture content in the filter cake was decreased with increasing reagent addition. At RU additions of 1 and 5 lb/t, the moisture contents of the cake were reduced from 17.6% to 16.1% and 14.4%, respectively.

Similar results were obtained when RW was used as a dewatering aid. These values correspond to a 15-20% moisture reduction in the filter product.

Table 2.30. Effect of pilot-scale filter disc speed on filter cake production rates and moisture content

Filter Disc Speed (min/rev)	Product Rate (lb/hr)	Moisture Content (%)
3.0	182.6	16.9
2.0	238.9	16.7
1.0	256.5	17.3

The effect of filter disc speed was also investigated in the pilot-scale tests. Table 2.30 summarizes the results obtained by increasing the filter disc speed from 3 to 1 min/rev for dewatering of the plant flotation product. The product rate increased from 182.6 to 256.5 lb/hr, with a small change in the moisture content of the filter cake. The results show that it would be possible to increase the filter capacity by 28% without adversely impacting the moisture content of the filter cake. Tests were conducted on Buchanan plant flotation product without dewatering reagents.

A series of pilot-scale dewatering tests were conducted to study the effects of different

Table 2.29 Effect of reagent addition on the pilot-scale dewatering of Buchanan's flotation product

Reagent Dosage (lb/t)	Moisture Content (%)	
	RU	RW
0.0	16.9	
1.43	15.7	
3.08	15.2	
6.05	14.5	
0.00		16.9
1.47		15.8
3.19		15.5
6.31		15.0

vacuum levels on moisture reduction. The initial results, which are presented in Table 2.31, were obtained without dewatering aid addition. The data show that it is possible to reduce the product moisture from 19.7% to 17.1% by simply increasing the vacuum level from 5 to 15 inches Hg. It seems that a further increase in vacuum is not advantageous in terms of further lowering the moisture contents of the filter products. It should be mentioned here that the disc filters in the Buchanan preparation plant are currently operated at vacuum levels of only 10.5-11.0 inches Hg. Because of such low vacuum levels, the plant filter product typically contains 21-22% moisture. The present work shows that by increasing the vacuum levels from 5-11 in Hg, the plant could probably obtain a filter product with 17-18% moisture. Besides dewatering aid addition, vacuum level is one of the important operating conditions determining the final product moisture in the filter cake.

Table 2.32 gives the pilot-scale test results obtained on the Buchanan plant flotation product using RW as the dewatering aid at various vacuum pressures. As shown, the moisture content in the filter cake was reduced with increasing vacuum pressure. At vacuum pressure increasing from 10-20 in Hg, the moisture contents of the cake were reduced from 18.2 to 16.8% and 16.3%, respectively.

The test results given in Table 2.33 indicate that a further improvement in filter cake moisture (about two percentage points) was obtained when using RU as dewatering aid. As the vacuum levels increased from 10 to 20 in Hg, the moisture content of the cake were reduced from 16.8 % to 14.5% and 14.3%, respectively.

Table 2.31. Effect of pilot-scale filter vacuum level on filter cake moisture contents

Applied Vacuum (in Hg)	Moisture Content (%)
10	19.7
15	16.7
20	17.1

Table 2.32. Effect of pilot-scale filter vacuum level on filter cake moisture contents (4 lb/t RW)

Applied Vacuum (Inch Hg)	Moisture Content (%)
10	18.2
15	16.8
20	16.3

Table 2.33. Effect of pilot-scale filter vacuum level on filter cake moisture contents (4 lb/t RU)

Applied Vacuum (Inch Hg)	Moisture Content (%)
10	16.8
15	14.5
20	14.3

2.3.6. Elkview Site

a) Site Description

The Elkview coal cleaning plant, B.C Canada, is processing 1,400 metric tons per hour (t/hr) of run-of-the-mine (ROM) coals. The materials that are floated and dewatered are classifying cyclone products. The cyclone overflows (O/F) are fed to five banks of mechanically agitated flotation cells, while the underflows (U/F) are fed to sieve bends (60 mesh). The sieve bend U/F joins cyclone O/F and are fed to the flotation cells, while the sieve bend O/F bypasses the flotation cells. The froth product and the sieve bend O/F's are combined and fed to vacuum disc filters to reduce the moisture to approximately 21.5%. The filter cake is then fed to a thermal dryer to further reduce the moisture to 8.4%. Typically, the thermal dryer is operating at its maximum capacity, i.e., 65 t/hr of water evaporated, and cannot handle additional froth product. Under this condition, operators cannot pull the flotation cells hard, causing a significant loss of fine coal.

The primary objective of the project was to develop appropriate methods of reducing the filter cake moisture to the level that can eliminate the situation where the thermal dryer is acting as a bottleneck for increased production. These novel dewatering aids are designed to increase hydrophobicity. As such, the dewatering aids can be added to a flotation cell displacing some, or perhaps even all, of the conventional collector (kerosene) that is currently used. This can result in a higher flotation recovery while at the same time improving dewatering. However, the novel dewatering aids would work better if they were added to a separate conditioner with a strong agitation since the energy dissipation in a flotation cell is generally less than that in a well-designed conditioner. Therefore, the extent of moisture reduction may be less when the froth cell is used for conditioning. Adding the dewatering reagent in place of collector for flotation may be sufficient since relatively small moisture reduction may be sufficient in eliminating the bottleneck at the thermal dryer and thereby allowing operators to pull the flotation cell hard and increase the recovery. To meet this objective, a series of dewatering tests have been conducted at Virginia Tech. The present work was limited to testing the novel dewatering aids to reduce the moisture of the filter cakes produced from the vacuum disc filters at Elkview.

b) Reagents

The investigation, both laboratory and pilot-scale, was performed with varying amounts of three types of dewatering aids, namely RW, RU and RV. Since these dewatering aids are insoluble in water, they were dissolved in a solvent. In both laboratory and pilot scale experiments diesel was used as the solvent. The ratio of reagents-to-solvent was optimized in previous studies by varying the individual dosages (0.5 to 3 lb/ton), while maintaining the total blend dosage constant. For this test program, the optimum combination for a given dewatering aid and solvent is one to two (1:2) dewatering aid-to-diesel ratio. When flotation feed was used, the sample was subjected to laboratory flotation tests using 0.66 lb/t of kerosene or RV as collectors and 0.44 lb/t of MIBC as frother.

c) Coal Samples

Two types of samples were received from the Elkview site, i.e., a standard metallurgical coal (Std-Met) and a medium-volatile metallurgical coal (Mid-Vol Met). In each case, both flotation feed (minus 60 mesh, 2-3% solid by weight) and vacuum filter feed (67% froth product and 33% sieve bend overflow, 25% solids by weight) samples were received.

d) Laboratory Procedures

Most of the laboratory filtration tests were conducted using a 2-inch diameter Buchner vacuum filter at 20-inch vacuum pressure (68 kPa) and 2 minutes of drying cycle time. To compare the effect of pressure drop on filtration, a few tests were also conducted using a 2-inch diameter pressure filter at a 30 psi of compressed air. In each dewatering test, a coal sample was conditioned in a mixing tank for 2 minutes. The cake thicknesses were varied in the range of 15 to 25 mm by varying the slurry volume.

To prepare the test samples, a series of flotation tests were conducted using a Denver laboratory flotation cell. In each test, a known amount of a dewatering/flotation aid was added to the flotation cell and the slurry was agitated (or conditioned) for 2 minutes before introducing air to the slurry to initiate flotation. When the froth product was to be used for dewatering tests, the flotation tests were conducted until exhaustion and the froth product was used for filtration tests in the same manner as described above. In this procedure, the flotation cell was used effectively as a conditioner. In this series of tests, the flotation products were not analyzed for ash to determine the recovery.

e) Laboratory Test Results (Filter Feed – Standard Metallurgical Coal)

Table 2.34 gives the results obtained on the filter feed sample using RW and RV as dewatering aids. The tests were conducted at 22-24 mm cake thickness by varying the reagent dosages. The reagents were used as 1:2 blends with diesel, and the dosages given refer to the

Table 2.34 Effect of reagent addition on dewatering of Elkview's filter feed sample (standard metallurgical coal)

Reagent Dosage (lb/ton)	Moisture (%)	
	RW	RV
0	20.17	20.17
0.1	17.92	17.72
0.5	15.82	16.06
1	14.73	15.43
2	13.11	13.88
3	12.73	13.84

Table 2.35. Effect of using RW, RU and RV on the dewatering of Elkview's filter feed (STD Met Coal)

Reagent Dosage (lb/ton)	Moisture (%)		
	RW	RU	RV
0	25.56	25.56	25.56
0.1	18.04	17.69	17.99
0.5	15.83	15.45	15.49
1	14.39	13.73	15.36
2	11.93	12.97	13.82
3	11.93	12.32	12.92

active ingredients only. As shown, the cake moistures decreased from 20.17% to 12-14% range.

Table 2.35 gives the results of the laboratory vacuum filter tests conducted on the filter feed (STD met coal) sample using RW, RV and RU as dewatering aids. The tests were conducted at approximately 15 mm cake thicknesses. The moisture was reduced from 25.56% to the 12-13% range, which represents approximately 50% moisture reduction. The higher moisture obtained at the control test was probably due to the fact that the tests were conducted a few days after receiving the sample. The results showed that RW was most effective, followed closely by RU and RV.

f) Laboratory Test Results (Filter Feed – Medium Volatile Metallurgical Coal)

Table 2.36 gives the laboratory test results obtained on the Mid-Vol filter feed using RW, RV and RU as dewatering aids. The tests were conducted at approximately 15 mm cake thickness by varying the reagent dosages. Cake moistures were reduced from 23.34% to 13-14% range, representing approximately 43% moisture reductions.

Table 2.36 Effect of using Reagents RU and RV for Elkview's filter feed (Mid-Vol Met Coal)

Reagent Dosage (lb/ton)	Moisture (%)		
	RW	RU	RV
0	23.34	23.34	23.34
0.1	19.07	19.81	19.60
0.5	15.89	17.52	17.58
1	13.98	15.51	15.80
2	13.96	14.32	15.23
3	13.18	13.95	14.36

Table 2.37 shows the laboratory test results obtained on the filter feed using RW, RV and RU as dewatering aids. The tests were conducted at about 22-24 mm cake thicknesses. The moistures were reduced from 23.37% to the 15-16% range, which corresponded to approximately 35% moisture reductions.

Table 2.37. Effect of using RW, RU and RV on Elkview's filter feed sample (Mid-Vol Met Coal)

Reagent Dosage (lb/ton)	Moisture (%)		
	RW	RU	RV
0	23.37	23.37	23.37
0.1	20.11	17.99	19.86
0.5	17.50	17.69	17.86
1	15.70	16.09	16.46
2	15.42	15.69	15.92
3	15.42	15.00	16.44

g) Laboratory Test Results (Flotation Feed)

The two flotation feed samples (Std-Met and Mid-Vol-Met) were subjected to a series of dewatering tests. As shown in the previous sections of this report, cake moistures can be reduced to the 12-14% range by weight using 2 to 3 lb/ton (active ingredient) of the novel dewatering aids. This was achieved by adding the dewatering aid to a conditioning tank so that it is readily dispersed in the slurry. It was found in our previous work that moisture reduction improves with increasing energy input during conditioning. Figure 2.13 shows a relationship between moisture reduction and energy input. The results presented in this figure have been obtained on a clean bituminous coal sample (from Moss 3 preparation plant, Virginia) that has been pulverized before dewatering tests.

In the present work, it was decided that dewatering tests be conducted without using a stand-alone conditioning tank. Instead, the coal samples were conditioned during flotation with varying reagent dosages. The energy dissipation imparted by a flotation cell is substantially

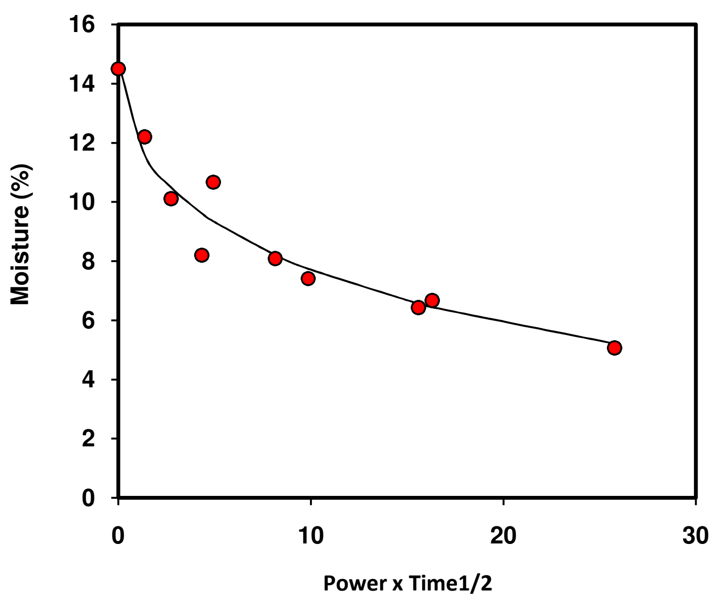


Figure 2.13 Effect of conditioning on moisture (Middlefork coal)

lower than that of a conditioner. Therefore, the results would not be as good as the case of using a conditioner, but the moisture reduction may suffice the needs at Elkview. One concern we had with this approach was the possibility that we would have to use a higher dose of frother when using a higher dose of collector/dewatering aids.

Figure 2.14 shows the results obtained with the standard metallurgical coal and the Mid-vol coal using RV. The reagent dosages given in the figure include both active ingredient and solvent. The ratio of the active ingredient and solvent ratio was 1:2. Cake thicknesses were approximately 20 mm and two minutes of drying cycle time was employed. In the control test conducted with kerosene as collector, the cake formation time was one minute, which was reduced to 25-35 seconds when using RV. The tests were conducted on the flotation products without conditioning. The flotation tests were conducted using various amounts of collectors. In all flotation tests, 0.33 lb/t of MIBC was used as frother. With this coal, it was not necessary to increase frother dosage at higher collector dosages. As expected, moisture reductions improved substantially with increasing collector dosage. The reagent dosages given are inclusive of the solvent, which comprised 66% of the total reagent addition (the results are plotted in metric units). At 1,500 g/t (3.3 lb/t), which was the highest dosage employed in these series of tests, the cake moistures were 15.9% for the Mid-vol coal and 16.6% for the standard metallurgical coal. These values are comparable to those obtained with the filter feeds. At the 1,500 g/t (3.3 lb/t) dosage, which is equivalent to 500 g/t (or 1 lb/ton) active ingredient, the dewatering tests conducted on the filter feeds after stand-alone conditioning gave moistures of 15.56% for the standard metallurgical coal and 16.46% for the Mid-Vol coal. Note that the moistures of the floated products are comparable to those of the filter feeds despite the facts that the separate conditioning step was omitted and that particle size was finer. Recall that the filter feed was 0.6

mm x 0 while the flotation product was 0.3 mm x 0. It appears, therefore, that the conditioning step can be omitted for the Elkview coal, which is a significant advantage.

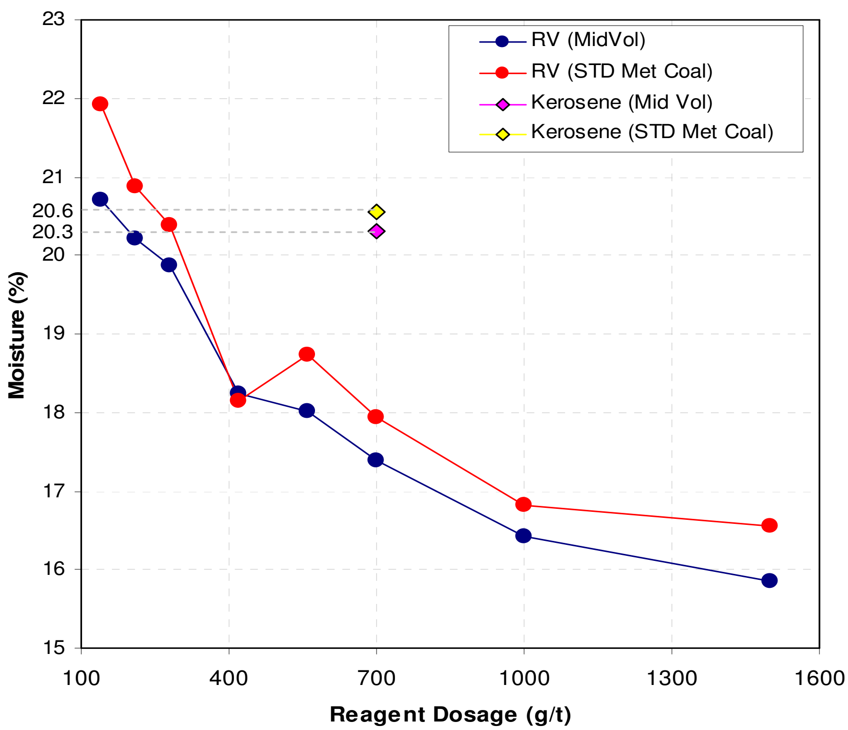


Figure 2.14 Results of the low-pressure pressure filter tests conducted on the Elkview coal sample (filter feed).

It would be of interest to compare the results obtained with RV with those obtained with kerosene. The froth products obtained using 700 g/t (1.4 lb/ton) of kerosene gave moistures of 20.6 and 20.3% for the standard metallurgical coal and medium volatile coals, respectively. At present, Elkview is using 600 g/t (1.3lb/t) kerosene as collector and 60 g/t of MIBC as frother. At 700 g/t RV and 120 g/t MIBC, we obtained 17.9 and 17.4% moistures for the standard metallurgical (1.4lb/t) coal and Mid-vol coal, respectively. Thus, the use of RV can reduce moistures by 2.7 to 2.9 percentage points absolute over the case of using kerosene as collector at dosage levels that are currently employed at Elkview. Figure 2.14 shows that further reductions in moisture can be achieved at higher reagent dosages.

2.3.7. Smith Branch Site

a) Site Description

One of the most promising coal samples evaluated in this project was obtained from the Smith Branch impoundment located near the Pinnacle Mine Complex. The complex, which is owned by Cleveland Cliffs Mining, consists of an underground mining operation, a surface wash plant, and the waste coal impoundment. The Pinnacle site contains approximately 100 million tons of unmined coal reserves of which 3.3-4.0 million tons are processed annually by the wash plant. The waste coal from the plant is diverted to the Smith Branch Impoundment, which is believed to contain 2.85 million tons of potentially recoverable fine coal.

b) Coal Samples

A number of coal samples were used to conduct dewatering tests during the evaluation and scale-up tests. Samples were taken from the PinnOak Company's Pinnacle Plant site and Smith Branch Impoundment near Pineville, WV. The samples consisted of a Vibracore composite sample taken from the Smith Branch Impoundment, a grab sample of current thickener underflow (taken in 2002), and a slip-stream sample of current thickener feed. All of these samples were tested in both the laboratory and pilot scale using newly-developed dewatering aids. Shown below is an overview of the samples used in dewatering tests.

i) Laboratory Tests

Smith Branch Vibracore Composite Sample (68% solid, 30% ash)
Plant's Thickener Underflow Sample (12% solid, 33% ash)
Plant's Thickener Feed Sample (1.6% solid, 46% ash)

ii) Pilot Scale Tests

Smith Branch Vibracore Composite Sample (68% solid, 30% ash)
Plant's Thickener Feed Sample (1.6% solid, 46% ash)

Vibracoring is a technology used to extract core samples of underwater sediments and wetland soils. The vibrating mechanism of a vibracorer, sometimes called the "vibrahead," operates on hydraulic, pneumatic, mechanical, or electrical power from an external source. The attached core tube is drilled into sediment by gravitational force and boosted by vibrational energy. When the drilling is completed, the vibracorer is turned off, and the tube is pulled out with the aid of hoist equipment. Extracting core samples via the Vibracore sampling method assessed the quality of fine coal contained in the impoundment. The 2.5 inch diameter cores were extracted down to depths of 25 to 30 ft. and subjected to size and ash analyses.

Figure 2.15 shows a typical set of size-by-size analyses that were obtained from one set of core samples. Error bars are provided to illustrate the high, average, and low values obtained for each size class. This particular set of data shows that the minus 270 mesh fraction contains about 60% of the coal tonnage, with an average ash content of about 33-35%. The raw quality within the impoundment was found to vary greatly, dependent on the distance from the discharge point into the impoundment. In general, coal extracted near the discharge point was found to be

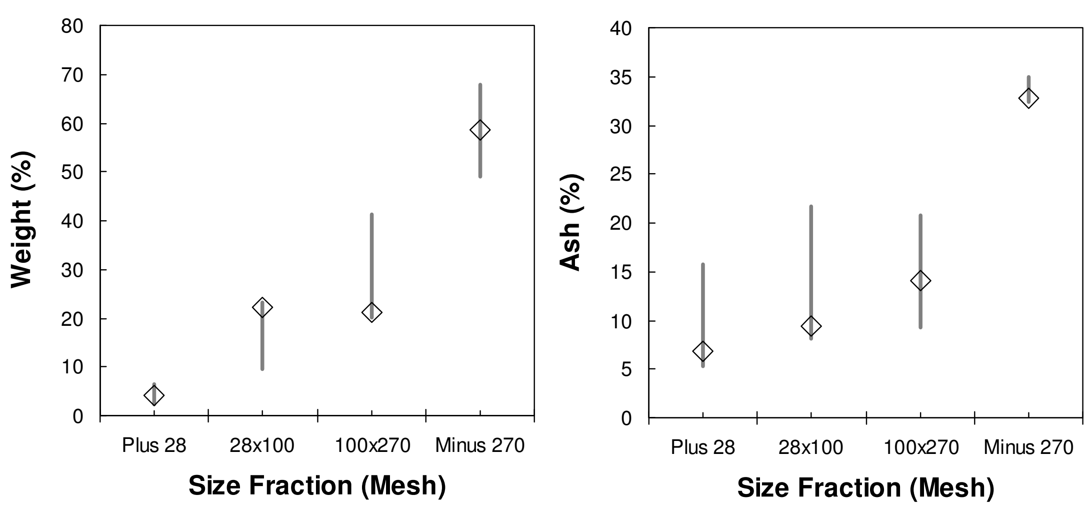


Figure 2.15 Weight and ash distributions of slurry from the Smith Branch impoundment

coarser and higher in ash, while the coal extracted farther away from the discharge point was finer and lower in ash. Also, coal fines extracted from a greater depth were found to be of better quality than coal taken from more shallow locations. This was expected because the coal fines deposited earlier in the life of the mine were discarded before several improvements were made to the fine coal circuits in the existing preparation plant. The average ash content of the remaining 40% had an ash content of less than 15%.

c) *Laboratory Procedures*

The laboratory flotation and dewatering equipment consists of a Denver laboratory flotation cell, a vacuum pump, a mechanical shaker, a stand-alone mixer – in some cases – and a Buchner Funnel with a fitted filter medium. All the vacuum filtration tests were conducted using a 2.5 inch diameter Buchner Funnel at 20-25 inch Hg (68 kPa) with a 40X60 wire screen mesh filter medium.

The dewatering tests were conducted to determine the best reagents and dosages for different samples and to investigate under what conditions a given dewatering aid can give the best performance. All the coal samples were tested within three days of receipt in order to minimize any artificial negative effects of aging or surface property changes on dewatering. Because the samples were taken from the waste pond, thickener feed discharge, and thickener underflow discharge, they possessed high impurities. For this reason, a cleaning step was employed before the filter tests. Depending on the solid content, the samples were prepared first by either diluting or decanting to 16-17% solids and floated in a laboratory Denver flotation cell approximately at 1000 rpm using 0.88 lb/t kerosene and 0.33 lb/t MIBC to remove ash. A limited number of flotation feed and product samples were analyzed for ash. Then, the samples were collected in a separate container to be used in filtration tests. The dewatering aids RW, RU, and RV were tested and diluted to 33.3% in solvent. When using the dewatering aids, conditioning is critical, and increased energy input in conditioning yields improved moisture reductions. For this reason, prior to each filter test, a known volume of slurry was conditioned with the dewatering aids in the mechanical shaker and, in some cases, the stand-alone mixer. When the mechanical shaker was used, the conditioning time was kept at 5 minutes, and when the stand-alone mixer was used, the conditioning time was kept at 2 minutes. Then, the conditioned sample was poured into the Buchner funnel for filtration. During the tests, the cake thickness was determined by the amount of the slurry sample used. Some of the conditions were kept constant during the tests. For example, a 7-10 mm cake thickness was determined for 2 minutes of drying time.

d) Pilot-Scale Procedures

The continuous dewatering test rig used at the Smith Branch site incorporated three main components, a column module, conditioner module, and a disc filter module, along with some other ancillary components. The details about the modules were described previously. For the pilot-scale tests, feed slurry was supplied from either the existing preparation pond reclaim facilities or in barrels and directed into the circuit feed sump. During the tests, the slurry was fed at a constant rate into a 12 inch diameter column by means of a peristaltic pump and flexible piping. The flotation column was used to produce a filter feed and reject ultra-fine hydrophilic clay that negatively impacts the effectiveness of some of the dewatering aids. Then, the clean coal froth was routed to the multi-stage conditioner module by gravity while the column reject slurry flowed, also by gravity, to a refuse sump. After adding appropriate dewatering reagents, the conditioned slurry was pumped at a constant rate into the filter test module. Next, the conditioned slurry was dewatered in the filter module. The filter cake was discharged into a container and immediately transferred to an oven for moisture analysis while the filter effluent was pumped to the reject sump. In addition, if any excess feed problems occurred, the column feed sump and conditioner tanks were allowed to overflow in a controlled manner.

e) Laboratory Results (Vibracore Samples)

The objective of the test series was to conduct preliminary investigations that evaluate the performance of dewatering aids. The laboratory tests were conducted on the Pinnacle Smith Branch Vibracore Composite Sample. Prior to the dewatering tests, the sample was cleaned using a Laboratory Denver Flotation Cell using 0.88 lb/t kerosene and 0.33 lb/t MIBC at 16.7 % solid content. Of the samples, the vibracore sample was very important because the data from the laboratory studies was also used to provide practical justification for the pilot scale test program which would soon to be carried out using Peterson Disc Filter.

A series of initial batch laboratory filter dewatering tests was conducted to determine whether the floated vibracore coal would respond well to the addition of the novel dewatering aids. A 62.5 mm diameter Buchner filter was used for these tests. The solids content of the sample was 16.0%. and a vacuum setup point of 20 inches Hg was utilized. The tests typically provided a cake thickness of 7-8 mm without reagent and 8-10 mm with reagent. A conditioning time of 5 minutes and a drying cycle time of 2 minutes was used. The volume of the slurry was 100 ml. The impoundment sample was floated at 16.7% solids using 0.88 lb/t kerosene and 0.33 lb/t MIBC before the filter tests. Table 2.38 gives the laboratory test results when RW was used as the dewatering aid. As shown, the moisture content in the filter cake was reduced when reagent addition was increased. At 1 and 5 lb/t RW additions, the moisture contents of the cake were reduced from 28.7% to 24.6% and 14.3%, respectively. The lower value corresponds to a 50% moisture reduction in the filter product.

Table 2.38 Effect of RW addition on the Pinnacle pond sample

RW Dosage (lb/t)	Moisture Content (%)	Moisture Reduction (%)
0	28.7	--
1	24.6	14.28
2	22.9	20.20
3	22.5	21.60
5	14.4	49.82

*62.5 mm diameter Buchner filter was used. The solid content of the sample was 16.0%. Vacuum setup point was 68 kPa (20 inches Hg). It was not changed during the experiment. Cake thickness; base: 7-8 mm, with reagent: 8-10 mm. Conditioning time: 5 minutes; drying cycle time: 2 minutes. The volume of the slurry was 100 ml. The impoundment sample was floated at 16.7% solids using 400 g/t kerosene and 150 g/t MIBC before the filter tests.

f) Laboratory Results (Thickener Underflow Samples)

At the preparation plant after cleaning the coal, the residue was sent to a thickener and then pumped to the slurry impoundment. Because it was an active impoundment, the feed sample, thickener underflow, was also investigated, and similar tests were conducted on this sample using RW and RU. Results for the thickener underflow sample are given in Table 2.39, and the moisture content of the filter product again decreases with increasing RW and RU additions from 1 to 5 lb/t. Here, the RW addition rate of 5 lb/t reduced cake moisture contents from 31.4% to 22.1%, giving a percentage moisture reduction of about 30%. This is much less than what was observed in the case of the impoundment sample and can attributed to the different mean particle sizes of these two samples. Mean particle size, as determined by Microtrac particle size analyzer in Beard Technologies' laboratory, was approximately 32 micron for the impoundment sample and 15 micron for the thickener underflow sample. The reasons for the relatively poor behavior of RU are unclear but may be related to the presence of flocculant in the sample.

Table 2.39 Effect of reagent addition on dewatering of Pinnacle thickener underflow sample

Reagent Dosage (lb/t)	Moisture Content (%)		Moisture Reduction (%)	
	RW	RU	RW	RU
0	31.4	31.4	--	--
1	30.3	28.8	3.5	8.28
2	28.0	28.2	10.82	10.19
3	22.7	--	27.71	--
5	22.1	27.8	29.61	11.5

*A 62.5 mm diameter Buchner funnel was used. The solid content of the sample was 16.0%. Vacuum setup point was 68 kPa (20 inches Hg). It changed to 61-68 kPa during the experiment. Cake thickness: base: 7.0 mm, with reagent: 8-8.5 mm. Conditioning time: 5 minutes; drying cycle time: 2 minutes. The volume of the slurry was 100 ml. The underflow sample was first floated using 400 g/t kerosene and 150 g/t MIBC before the filter tests.

Another set of tests was conducted on a deslimed thickener underflow sample to investigate the effect of particle size distribution. Before the dewatering tests 50% of minus 45 micron material was removed by wet screening. Table 2.40 shows the effect of the desliming on dewatering of this sample. The final cake moisture was reduced down to 22.0% moisture 5 lb/t dosage of RW. When RU was used as dewatering aid, the moisture could be reduced down to 21.4% moisture at the same dosage of 5 lb/t.

Table 2.40 Effect of desliming on dewatering of Pinnacle thickener underflow sample

Reagent Dosage (lb/t)	Moisture Content (%)		Moisture Reduction (%)	
	RW	RU	RW	RU
0	29.38	29.38	--	--
1	23.98	25.28	18.38	13.96
2	23.39	23.17	20.38	21.13
3	22.77	22.16	22.49	24.58
5	22.00	21.37	25.11	27.26

*A 62.5 mm diameter Buchner funnel was used. The solid content of the sample was 16.0%. Vacuum setup point was 68 kPa (20 inches Hg). It changed to 61-68 kPa during the experiment. Cake thickness: base: 7.0 mm, with reagent: 8-8.5 mm. Conditioning time: 5 minutes; drying cycle time: 2 minutes. The volume of the slurry was 100 ml. The underflow sample was first floated using 400 g/t kerosene and 150 g/t MIBC before the filter tests.

Table 2.41 Effect of reagent addition on dewatering of Pinnacle thickener feed sample

Reagent Dosage (lb/t)	Moisture Content (%)		Moisture Reduction (%)	
	RW	RU	RW	
0	29.5	29.5	--	--
1	26.5	27.3	10.17	7.46
2	24.3	25.2	17.63	14.58
3	22.2	23.1	24.75	21.70
5	21.1	21.8	28.48	26.10

*62.5 cm diameter Buchner filter was used. The solid content of the sample was 11.0%. Vacuum setup point was 68 kPa (20 inches Hg). It changed to 61-68 kPa during the experiment. Cake thickness; base: 7.0 mm, with reagent: 8-8.5 mm. Conditioning time: 5 minutes; drying cycle time: 2 minutes. The volume of the slurry was 100 ml. The thickener feed sample was floated using 400 g/t kerosene and 150 g/t MIBC before the filter tests.

g) Laboratory Results (Thickener Feed Samples)

Table 2.41 gives the results for the thickener feed sample. In this case, RW and RU reduced the cake moistures from 29.5% to 21.1% and 21.8%, respectively, and percentage moisture reductions of about 28% and 26% at addition rates of 5 lb/t. Again, these reductions are much less than those that occurred with the impoundment sample. Due to the preferential removal of fine clay during flotation, the mean size of the test sample was probably also around 15 microns.

h) Pilot-Scale Results (Vibracore Samples)

RW was used as the dewatering aid in these sets of pilot scale dewatering tests. Table 2.42 gives the pilot scale dewatering results obtained on the Smith Branch Impoundment sample. These results show that the moisture contents of the filter cakes are substantially decreased in the presence of dewatering aid. For example, at reagent dosage rates of 2-5 lb/t of RW, filter cake moisture content was reduced from 28.4% to 17.7%-16.3%. Cake thicknesses were as high as 16 mm when using RW as the dewatering aid versus 3-6 mm without a reagent. Even at this cake thickness, moisture reductions were significant. Additionally, filter effluents were much cleaner when dewatering aids were used, indicating that filter recoveries increased substantially in the presence of the dewatering aid.

Table 2.42 Effect of RW addition on the pilot scale dewatering of the Pinnacle-Smith Branch Impoundment sample using the mobile units

Reagent RW Dosage (lb/t)	Moisture Content (%)	Moisture Reduction (%)
0	28.4	--
1	19.6	30.99
2	17.7	37.68
3	17.2	39.43
5	16.3	42.61

* 10 sector single disc Peterson filter was used in the experiments. The solid content of the filter feed was 17.0%. The vacuum setup point was 81.27-84.66 kPa (24-25 inches Hg). It dropped to 67.73-77.89 kPa (20-23 inches Hg) during the experiment. Cake thickness; base: 6-8 mm, with reagent: 12-16 mm..

i) Pilot-Scale Results (Thickener Feed Samples)

Table 2.43 gives the pilot scale dewatering results obtained on the Pinnacle coal thickener feed sample tests. In this series of tests, RW was used as the dewatering aid. As shown below, the moisture content of the filter cake is reduced from 29.4% to 21.5% by the addition of 3 lb/t of RW. Likewise, the test results given in Table 2.44 show that the moisture content of the filter cakes are significantly decreased when RW and RU are used as dewatering aids. For example,

Table 2.43 Effect of RW addition on the pilot scale dewatering of the Pinnacle thickener feed sample using the mobile test units

Reagent RW Dosage (lb/t)	Moisture Content (%)	Moisture Reduction (%)
0	29.4	--
1	--	--
2	--	--
3	21.5	27.2
5	--	--

* 10 sector single disc Peterson filter was used in the experiments. The solid content of the filter feed was 12.3%. Vacuum setup point was 81.27-84.66 kPa (24-25 inches Hg). It dropped to 71.11-74.50 kPa (21-22 inches Hg) during the experiment. Cake thickness; base: 6-8 mm, with reagent: 12-16 mm.

Table 2.44 Effect of reagent addition on dewatering of Pinnacle thickener feed sample

Reagent Type	Reagent Dosage (lb/t)	Moisture Content (%)	Moisture Reduction (%)
RW	0	28.0	--
	0.95	23.4	16.43
	3.7	20.3	27.5
	4.45	21.8	22.14
RU	0	28.0	--
	3.1	20.6	26.43
	5.5	21.0	25

* 10 sector single disc Peterson filter was used in the experiments. The solid content of the filter feed was 8-9%. Vacuum setup point was 71.11-88.05 kPa (21-26 inches Hg). It dropped to 47.41-77.89 kPa (14-23 inches Hg) during the experiment. Cake thickness; base: 5-6 mm, with reagent: 10-20 mm.

filter cake moisture contents were reduced from 28.0% to 23.4% at 2.09 lb/t and 20.3% at a 3.7 lb/t reagent dosage. RU performs as well as RW in terms of moisture reduction. The moisture content in the filter cake was reduced from 28% to 20.6% in the presence of 3.1 lb/t of RU. These results are in good agreement with those obtained in the previous laboratory tests.

2.3.8. Moatize Site

a) Site Specific Information

A new reserve is being developed by Companhia Vale do Rio Doce (CVRD). The coal sample was from the Section 2A of the reserve. CVRD is in the process of designing a coal preparation plant with a throughput capacity of 4,000 metric tonnes per hour (t/h). The coal sample received was a fine coal (0.25 mm x 0), which will be fed to a single-stage flotation circuit. Columns are chosen over mechanically agitated cells, because the former can produce cleaner froth products. The throughput capacity of the flotation circuit will be around 440 to 560 t/h.

The CVRD is looking for the possibility of reducing the moisture of the column flotation product to less than 15% by weight using horizontal belt filters (HBF). It has been a challenge to achieve such a low level of moisture with a by-zero froth product using a vacuum filter. It is possible, however, to achieve the objective, if the HBF is used in conjunction with the novel dewatering aids developed at Virginia Tech. For this reason a series of laboratory dewatering tests have been conducted on a coal sample from the Moatize region in Mozambique.

b) Experimental Design

Flotation and Dewatering Aids

During this investigation, dewatering tests were performed with varying amounts of two types of dewatering aids, RU, and RV. Since these dewatering aids are insoluble in water, they were dissolved in diesel which was used as a solvent. The ratio of reagents-to-solvent was optimized in previous studies by varying the individual dosages (0.5 to 3lb/t), while maintaining the total blend dosage constant. For this test program, the optimum combination for a given dewatering aid and solvent is one to two (1:2) dewatering aid-diesel ratio. When flotation tests were conducted, RV, Nalco-9855 and kerosene were used as collectors while Nalco-9840 was used as frother.

Samples and Procedures

Coal Samples: The coal sample was received as wet cake in a bucket, which had been obtained by decantation. Prior to each series of dewatering tests, a portion of the wet cake was removed from the bucket, mixed with a volume of tap water, and agitated for a minimum of 3 hours. After the agitation, part of the slurry was transferred to a Denver D-12 laboratory flotation cell and subjected to flotation. The froth product was then used as a feed to a series of laboratory vacuum filtration tests.

Procedure: The froth product from a single-stage laboratory flotation tests was placed in container, and agitated to obtain a homogenous suspension of clean coal. A volume of the suspension was removed by means of a cup of known volume. The suspension was then subjected to vacuum filtration tests using a 2-inch diameter Buchner filter using a fabric filter medium. The thickness of the filter cake formed on the filter cloth was varied by controlling the volume of the suspension filtered. When the slurry volume was varied from 100, 150 to 200 ml, the cake thicknesses varied from approximately 10, 15, and 20-22 mm. When the slurry volume was varied from 100, 150 to 200 ml, the cake thicknesses varied from approximately 10, 15, and 20-22 mm. At a given filtration experiment, the initial vacuum pressure was set at 20-inch Hg, which was reduced to 15- to 17-inch Hg at the end of the 2 minutes of filtration time employed in the present work.

Initially, the flotation tests were conducted at 5 to 6%. In later experiments, the solid concentration was increased to 10% to obtain froth products with a higher solid content. In some experiments, the feeds to filtration experiments were conditioned in a standalone conditioner. In others, the flotation cell was used as a conditioner.

c) *Results and Discussions*

Using Flotation Cell as Conditioner

The first set of dewatering tests was conducted by adding the dewatering aids directly into the flotation cell without using a stand-alone conditioning tank. In this procedure, the flotation cell was used effectively as a conditioner. This was possible because the dewatering aids being tested in the project can increase the hydrophobicity of coal, as noted previously. The froth products obtained from the flotation tests were then subjected to dewatering tests using the Buchner filter. Each test was conducted at 15 mm cake thickness and 2 minutes of filtration time, including cake formation and drying cycle times.

For comparison, a series of dewatering tests was also conducted using kerosene as collector. The use of Nalco 9855 and RV gave substantially lower cake moistures than the diesel at low reagent dosages. At 0.88 lb/t, diesel produced a cake with 22.8% moisture, while using RV resulted in 18.1% moisture reduction.

Using a Standalone Conditioner

In the second set of tests, flotation tests were conducted using 0.88 lb/t of kerosene as collector. The froth product (approximately 10% solids) was conditioned in the presence of a dewatering aid for 2 minutes in a small, rectangular-shaped conditioner prior to dewatering test. The results obtained using varying amounts of two different dewatering aids (RV and RU) are given in Table 2.45.

Table 2.45 Effect of reagent addition on the dewatering of the froth products obtained using 400 g/t diesel as collector

Reagent Dosage (lb/t)	Cake Moisture (%)	
	RV	RU
0	22.30	22.30
0.25	17.35	17.50
0.5	16.28	16.03
2	15.93	15.91
3	14.78	15.00

The control tests conducted without any dewatering aid gave 22.30 % moisture. At increased reagent dosages, the cake moisture was reduced to the 14-15% range, which represented approximately 33% reduction in moisture. The two different dewatering aids tested gave no significant differences. All tests were conducted at 15 mm cake thicknesses.

Effect of Conditioning Time

The 2 minutes of conditioning time employed in the test work as described in the foregoing sections of this test work were based on the previous experiences with other coal samples. In order to determine more specific conditioning times needed for the Moatize coal, a series of dewatering tests were conducted by varying the conditioning time. At each dosage level, 15, 30 and 60 seconds of conditioning times were employed. It was found that at the lower reagent dosage, a short conditioning time was sufficient. At 3lb/t, however, cake moistures varied in the range of 16.19, 15.30, and 14.84% at 15, 30 and 60 seconds of conditioning times, respectively.

Another set of dewatering tests were conducted at 2 lb/t RV In this series, the conditioning times were varied from 15 to 120 seconds. The results are given in Table 2. As shown, the conditioning time makes no significant difference beyond 15 seconds. This finding suggests that the conditioner can be much smaller than originally anticipated. The data represented in Table 2.46 suggest that at reagent dosages of up to 2 lb/t, only a short conditioning time was necessary. At higher reagent dosages, longer conditioning times were helpful.

Table 2.46 Effect of conditioning time on the dewatering of flotation product using 2 lb/t RV

Conditioning Time (seconds)	Cake Moisture (%)
0	22.83
15	16.55
30	16.21
45	16.10
60	16.26
90	16.05
120	16.08

Effect of Cake Thickness

Another important parameter affecting filtration is the cake thickness. Therefore, a series of dewatering tests were conducted by varying cake thicknesses in the range of 10 to 22 mm in the presence of 3 lb/t RV. In each test, a 1-minute conditioning time and 2 minute filtration time were employed. The results showed that that cake thickness is critical in controlling cake moistures. As shown in Table 2.47, the targeted 15% moisture can be readily obtained by decreasing cake thickness.

Table 2.47 Effect of cake thickness on cake moisture in the presence of 3 lb/t RV

Cake Thickness (mm)	Cake Moisture (% wt)
20-22	19.60
15	14.83
10	12.33

2.4. SUMMARY AND CONCLUSIONS

Laboratory- and pilot-scale dewatering tests were conducted on various fine coal samples received from several coal preparation plants. The objectives of the dewatering tests were to maximize the effects of novel dewatering aids in terms of final cake moisture and the dewatering kinetics in order to evaluate possible industrial applications. The evaluation tests were conducted using various types and dosages of novel dewatering aids while varying operating parameters (e.g., cake formation time, dry cycle time, vacuum and air pressure and cake thickness.) Laboratory tests were conducted using a Buchner Funnel and air pressure filters while the pilot-scale tests were conducted using vacuum disc and horizontal belt filters. During pilot-scale tests, a two-step conditioner was also used to ensure effective mixing of the aids. In some of the dewatering tests, a column flotation unit was utilized to produce fresh coal.

Currently, fine particle coal reports to the Mingo Logan Preparation Plant's discard fine stream due to the inability of centrifuges to capture this material. A series of laboratory- and pilot-scale dewatering tests were also conducted to study the feasibility of using the novel dewatering aids in conjunction with horizontal belt filters to recover this fine coal. The tests varied types and dosages of dewatering aids added to feeds comprised of different mixtures of relatively coarse and fine coal particles. Dewatering test results showed that a 20% moisture reduction along with almost 50% increase in kinetics could be achieved on fine coal feeds (nearly 43% particles of this product was passing 325 mesh (-44 μm) and contained 81% ash). When fine coal samples were subjected to dewatering tests after re-cleaning to reduce the amount of hydrophilic ash forming minerals, the moisture reduction was further increased to 35%. Similar results were obtained when tests were conducted on relatively coarser samples comprised of 50% fine and 50% spiral product: the moisture could be reduced by 20-30% while increasing the kinetics by 30-50%.

On-site pilot-scale tests using HBF showed that, when the operating parameters are optimized, it is possible to lower the moisture of fine coal samples by 18% and coarser samples by 30%. Results also indicated that adequate mixing time and intensity is very critical and should be optimized. Using flocculants along with the novel dewatering aids additionally increased the effectiveness by increasing kinetics. If the dewatering aids are used along with the required equipment (i.e., column and HBF), the additional annual net revenue was calculated to be $1.57 million with the payback time of 3.1 years.

Several pilot-scale dewatering tests were also conducted on Coal Clean Preparation Plant feeds (100% -325 mesh) to investigate the possible application of novel dewatering aids. The fine coal was treated by using a column flotation unit and fed to a pilot-scale disc filter. The results showed that under optimized conditions, it is possible to lower the moisture by almost 20% while increasing the dewatering kinetics. Based on the results that were obtained on coal from this site, it appears possible to increase the fine circuit's recovery while lowering the moisture.

Coal samples obtained from Concord Preparation plant were subjected to flotation and dewatering tests to investigate the recovery of the fine coal at acceptable moisture levels. Laboratory test results showed that the moisture could be reduced by 26%, and to verify these results, a number of on-site pilot-scale tests were also conducted. Pilot-scale results confirmed that dewatering aids are capable of lowering the final cake moisture by 21%. Additionally, a number of tests were conducted using the dewatering aids as collectors, wherein the dosage was only 50% of a normal diesel dosage. Under these conditions, it was observed that recovery was increased and the moisture was lowered by 7%. When the same dosage as diesel was utilized, the moisture was further reduced, giving a 14% overall reduction. The dewatering kinetics were also increased.

Several dewatering tests were conducted on Buchanan filter feed samples. The final cake moisture was found to be 18.1% for the control tests and 14.9% in the presence of a novel dewatering aid. Similar tests were conducted with the pilot-scale unit and comparable moisture reductions were obtained. However, considering the relatively coarser size of the coal, the results were not good as expected. This may be attributed to water chemistry (i.e., excessive amounts of Ca^{2+} ions present in Buchanan plant water.) Additionally, a series of tests were conducted to investigate the effect of mixing intensity and time. It was found that supplying proper mixing intensity and time may lower the final cake moisture substantially. One study showed that total moisture reduction was around 36% under proper mixing conditions as compared to 19% when the mixing was not sufficient.

A number of coal samples were received from Elkview Coal Preparation Plant, where insufficient moisture reduction causes a bottle neck in production. Laboratory dewatering test results indicated that the final cake moisture may be lowered by 35-50% depending on the coal feed type. Using dewatering aids as collectors at a 700 g/t dosage, the recovery was the same or better as compared to diesel. Without any additional dewatering aids, a moisture reduction of about 13% was observed.

An investigation was also conducted to determine if the novel dewatering aids could be utilized in industrial applications for fine coal beneficiation at the Smith Branch Reclamation Site. To this end, a series of laboratory- and pilot-scale dewatering tests were conducted on samples from this site. The results showed that the moisture content of the filter cake was reduced as reagent addition was increased. At 1 and 5 lb/t RW additions, the moisture contents of the cake were reduced from an initial 28.7% to 24.6%, and to 14.3%, respectively. The latter value corresponds to a 50% moisture reduction overall in the filter product. A number of laboratory tests were also conducted on thickener feed and thickener underflow samples, wherein the moisture was reduced by 25-27%. Pilot-scale tests produced similar results.

In a coal preparation plant that will be built in Moatize, Mozambique, Companhia Vale Rio Doce seeks to lower the moisture of it's column flotation product to less than 15% (by weight) using horizontal belt filters (HBF). Achieving this low moisture level on a by-zero froth product is very challenging using a vacuum filter. It is possible, however, to achieve this level if the HBF is used in conjunction with the novel dewatering aids. This was proven during a series of laboratory dewatering tests on a coal sample from the Moatize region. Results indicated that considerably low moistures are possible when using the novel aids as only dewatering agents. Furthermore, when using the aids as collector, it is also possible to obtain low moisture contents. Additionally, the Moatize coal can be treated using relatively shorter chemical mixing times.

Overall, test results clearly showed that when operating parameters are optimized, use of novel dewatering aids can generate substantial moisture reductions that cannot be achieved by mechanical means. Additionally, the dewatering kinetics can be increased, which in turn may greatly increase the throughput of filter operations. The dewatering aids can also be used as collectors where mixing is not possible. They not only produce similar or better results than conventional collectors, but also lowering moisture contents.

2.5. REFERENCES

1. Y.A. Liu, The Physical Cleaning of Coal: Present and Developing Methods. Energy, Power, and Environment, Vol. 17. 1982.
2. P. Somasundaran, .Advances in Mineral Processing 1986, Littleton: The Society of Mining Engineers of AIME. 756.
3. B.K. Parekah, J.D. Miller, Advances in Flotation Technology, 1999, Littleton: Society for Mining, Metallurgy, and Exploration. 463.
4. Wills, B.A., Coal Preparation Technology: D.G. Osborne Graham & Trotman Ltd, Minerals Engineering, 1988. 1(4): p. 374-375.
5. Svarovsky, Solid-Liquid Separation. 1991, London: Butterworth.
6. Akers, R.J., Solid-Liquid Separation: Edited by L. Svarovsky, published by Butterworths, London, The Chemical Engineering Journal, 1978. 15(1): p. 77-79.
7. D.J. Condie, M.H., and C.J. Veal, Modelling The Vacuum Filtration of Fine Coal. Filtration & Separation, 1996.
8. Condie, D.J. and C.J. Veal, Modelling the Vacuum Filtration of Fine Coal: Part 2. Filtration & Separation, 1997. 34(9): p. 957-963.

CHAPTER 3 SIMULATION AND SCALE-UP

3.1. INTRODUCTION

There are certain difficulties in coal processing, especially with fine coal cleaning and dewatering. When coal is cleaned by flotation – the most dominant fine coal cleaning method – the product is in slurry form and has to be dewatered to acceptable moisture levels before sale.[1-3] As stated in the previous chapter, the final cake moisture and the production rate of a filter unit are important factors which must be optimized for an effective operation.[2] The Mingo Logan Preparation Plant is a particular site which has suffered from inefficiencies in flotation and especially dewatering. The plant has been losing valuable coal because its current operation design does not allow recovering the entire fine coal fraction from centrifuges. Typically, centrifuges are very effective at coarse sizes, but the fine coal recovery is relatively low; and when the fine size fraction is increased, the moisture is increased to levels that are unacceptable.

Taking these facts into account, a detailed dewatering study was undertaken to investigate the feasibility of a vacuum filtration operation, a vacuum disc filter or a horizontal belt filter (HBF). Based on the promising results obtained using novel dewatering aids, it was proposed that vacuum filtration may be used to decrease the loss of fines fraction and effectively lower the moisture.

In addition to the feasibility study, a number of parallel dewatering tests were conducted to develop empirical models and simulations for a vacuum disc filter or HBF. The results from these dewatering tests, including the plots, tables and the general information were used to create the empirical model by correlating and combining the filtration parameters. Development of an effective computer simulation will be valuable in handling the complexities involved in data analysis and general performance evaluation.

Data from the dewatering tests require moderately straightforward analysis in order to make them useful for the simulation development. As a starting point, some qualitative screening tests were carried out to determine what effects certain reagents and other fundamental dewatering parameters have on final cake moisture. After establishing the key parameters and their influences on cake moisture, more specific tests were undertaken to evaluate overall performance.

It should be noted that there is not a standard procedure for dewatering modeling and simulation. However, it is essential to perform the analysis to assess the variations in cake thickness, cake weight, vacuum pressure, filtration time, concentration in the slurry, particle size and distribution, filtration rate, etc.; experience has shown that these parameters have a marked effect on the dewatering performance and profound influence in designing the simulation.

3.2. EXPERIMENTAL

The filtration tests were carried out with a simple apparatus to determine the effects of the parameters on final cake moisture. For these tests, a Buchner Funnel equipped with a filter cloth is used. The general procedure to obtain the dewatering data is as follows:

1. the solid concentration in the slurry is determined,
2. a well mixed-known amount of slurry sample is poured into the funnel containing a filter cloth,
3. the vacuum pressure is adjusted to a pre-decided level and the vacuum valve was opened,
4. the filtration characteristic were observed and recorded (i.e., pressure drop, cake formation time, dry cycle time, cake weight and thickness),
5. chemical pre-treatment was done using dewatering aids,
6. the final cake moisture is determined in each test.

Dewatering aids are used to enhance the solid-liquid separation efficiency by aggregation and hydrophobization mechanisms. The role of novel dewatering aids is mainly to lower the surface tension and increase the coal hydrophobicity and thus enhance the ease of the water removal.

3.3. DATA COLLECTION AND RESULTS

The dewatering process is controlled by several factors, including reagent type and dosage, particle size, filtration time, cake thickness and weight, and vacuum pressure during filtration. These parameters are required for preliminary filtration calculations and empirical modeling. The effects of these parameters were investigated and a limited number of test results are represented as examples below.

The tests were conducted on two samples obtained from the Mingo Logan Preparation Plant. One sample was a 'mixture' and the other was a flotation product. The mixture sample was a blend of flotation product and spiral product at 50:50 blend ratios. The second sample was 100% flotation product. The -325 size fraction was determined to be about 30-35% for the mixture sample and 40-45% for the flotation product sample.

The first series of evaluation tests were conducted to determine the effects of the filter cloth and particle size on dewatering performance. As expected, the kinetics and moisture values were more acceptable with the coarser material (i.e., the mixture sample) compared to flotation product. Results also showed that among the filter cloths that were tested, 60X40 mesh screen produced the best results. After establishing the optimum filter cloth type and how the two samples responded to filtration, another set of tests was conducted to investigate the effects of reagent type and dosage, filtration time, vacuum pressure cake weight and thickness. The following tables represent the general trend of the effects of these parameters on the mixture sample. Identical tests were conducted for the flotation product sample as well, but only the mixture sample results are given as examples.

Example 1: Effect of Reagent Type and Dosage

A series of tests were conducted to investigate the effect of reagent type and dosage. The results showed that RU and RV performances were encouraging at 1 to 3lb/t. To increase the filtration rate Nalco 9806 was also used. Table 3.1 shows the effect of reagent type and dosage.

Table 3.1 Effect of reagent type and dosage on dewatering of mixture sample

Reagent Dosage	Moisture Content (%wt)		
(lb/ton)	RW	RU	RV
0	19.00	19.00	19.00
1	15.66	14.4	14.86
3	15.7	13.81	14.91
5	15.12	13.70	14.35

Reagent conditioning time: 5 minutes, Set up Vacuum: 68 kPa (20 inches Hg), Actual Vacuum: 16-18.5 inches Hg, Drying cycle time: 2 minutes, Cake thickness: 9-12mm, Solid Content: 25.0%, Dewatering Reagents: RW, RU, RV in Diesel (1:2)

Example 2: Effect of Filtration Time (CFT + DCT)

After establishing the optimum reagent type and dosage, a series of dewatering tests were performed to investigate the effects of the dry cycle time. An example of simplified and summarized series of tests is shown in Table 3.2. Tests were conducted at various intervals in the presence and absence of the dewatering reagents. As seen from Table 3.2, the dry cycle time has an important effect on final cake moisture. It is also an important parameter when designing the HBF's. The dry cycle time is directly related to the belt speed (i.e., the longer the dry cycle time is, the slower the belt speed or vice versa.) Generally, in real applications, if the feed to the filter is constant, slower belt speed allows a thicker cake formation but higher cake moistures. Higher belt speed results in greater solids production rates by forming thinner cake and lower cake moisture.

During the tests vacuum set-up pressure was adjusted to 20 in Hg and 100 ml of slurry was used to obtain approximately 1/2in filter cake. It was observed that there is a good correlation between drying time and the cake moisture. It was also found that the in

Table 3.2 Effect of filtration time on dewatering of mixture sample (0.5 in. cake)

Reagent Dosage lb/t	Filtration Time (sec)		Moisture (%)	Moisture Reduction (%)
	CFT	DCT		
Baseline (No Reagent)	59	2	30.32	-
	56	30	22.62	25.40
	51	60	20.55	32.22
	66	120	19.67	35.13
	77	300	18.88	37.73
RU (3 lbs/t)	40	2	24.31	-
	37	30	19.77	18.68
	45	60	18.85	22.46
	48	120	16.90	30.48
	47	300	14.84	38.96
RV (3 lbs/t)	29	2	31.91	-
	31	30	21.72	31.93
	25	60	17.76	44.34
	25	120	18.29	42.68
	27	300	15.53	51.33

* slurry tested: 100 ml, 34% solid; conditioning time: 5 min.; filter media: wire mesh screen

the presence of RU and RV the formation times were much shorter compared to baseline tests. In the presence of the dewatering aids, the moisture content is also lowered by 3 - 5%.

Example 3: Effect of Vacuum Pressure

During the filtration, the vacuum pressure is unquestionably related to the moisture content of the final cake. The higher the pressure, the lower the final cake moisture content and the better the cake consolidation. Table 3.3 shows the effects of the vacuum pressure on final cake moisture at various dry cycle times. The results show that the cake moisture is very sensitive to pressure changes. For instance, the cake moisture drops from 23.29% to 20.55% when the vacuum pressure is increased from 15 inHg to 20 inHg at 60 sec dry cycle time. The same trend can be observed at other dry cycle times as well.

Table 3.3 Effect of vacuum pressure on dewatering of mixture sample (0.5 in cake)

Reagent Dosage (lb/t)	Filtration Time (sec)		Cake Thickness (mm)	Vacuum inHG	Moisture (%)
	CFT	DCT			
Control (no Chemical)	95	2	11-14	15 inHG	30.37
	75	15			27.43
	87	30			25.67
	83	60			23.20
	72	120			21.11
Baseline (No Reagent)	59	2	13~14	20	30.32
	54	15			25.49
	56	30			22.62
	51	60			20.55
	66	120			19.67

Example 4: Effect of cake weight and thickness

The effect of cake weight and thickness on final cake moisture at various drying times is represented in Table 3.4. In this set of tests, the cake thickness was controlled by changing the slurry volume. RU was used as dewatering aid at 3lb/t. As seen under the same operating conditions, increasing the cake thickness increased the final cake moisture. A cake thickness of 5-7 mm resulted in approximately 14% moisture, and a thickness of 12-15 mm resulted in 17.3% moisture. When the thickness was further increased, the moisture was increased to 21.27%.

Table 3.4 Effect of cake weight and thickness on dewatering of mixture sample

Slurry Volume (ml)	Filtration Time (sec)		Cake Thickness/Weight (mm/gr)	Moisture (%)
	CFT	DCT		
50	2	15	(5~7)/(15)	16.78
	2	30		15.02
	3	60		14.04
100	18	15	(12~15)/(30)	18.90
	18	30		19.39
	19	60		17.32
150	61	15	(21~24)/(50)	23.63
	69	30		22.29
	68	60		20.01
200	91	15	(21~24)/(65)	26.59
	83	30		24.16
	88	60		21.27

*slurry tested: 30% solid; conditioning time: 5 min.; filter media: white filter cloth

After establishing the general effects of operating parameters, a series of tests were also conducted to be used for the modeling and simulation. Dewatering tests were conducted on the same two samples that were previously used for parameter evaluation tests. Approximately 60 dewatering tests were conducted varying the slurry volume and Dry Cycle Time (DCT). All parameters such as cake formation time (CFT), pressure during cake formation (P_F) and pressure during drying cycle (P_D) were recorded. The specific cake weight (W_S), normalization factor (NF) and the percent moisture were calculated from the data. In this modeling study, instead of cake thickness, the specific cake weight is used; this is because, in the presence of the dewatering aids which increase porosity, the cake thickness was artificially increased and created unreliable results. It was found that the cake weight gave more consistent and reliable results. Table 3.5 represents an example of the data collection sheet.

Table 3.5 Sample data collection sheet

Sample ID	Mixture			Flotation Product		
Reagent Type & Dosage (lb/t)	Baseline 0	RV 3	RU 3	Baseline 0	RV 3	RU 3
P_F (in Hg)	20	20	20	20	20	20
P_D (in Hg)	19	20	20	20	20	20
CFT (sec)	94	7	15	52	42	65
DCT (sec)	20	80	52	120	88	64
W_S (lb/ft^2)	3.70	3.38	4.73	2.42	3.49	5.00
Moisture (%)	34.08	14.27	21.35	27.86	26.18	28.71
P_F x CFT	31.33	2.33	5.00	17.33	14.00	21.66
NF	79.78	50.74	109.24	16.48	8.38	4.25

P_F (in Hg): Vacuum Pressure during Cake Forming, P_D (in Hg): Vacuum Pressure during Cake Drying, CFT (sec): Cake Formation Time, DCT (sec): Dry Cycle Time, W_S (lb/ft^2): Specific Cake Weight, NF: Normalization Factor $(1/W_S)(P_D.DCT/\mu)$, μ: Viscosity of the Liquid

3.4. EMPIRICAL MODEL

The empirical model and simulation were developed to assist in predicting the general behavior of cake filtration, especially for a vacuum disc or horizontal belt filter. The model includes the behavior of filtration in the absence and presence of novel dewatering aids (i.e., either RU or RV). It model consists of two segments:

i) Predicting (W_S) as a function of P_F (inHg) and CFT (sec):

$$W_S = f(CFT, P_F) \quad (1)$$

ii) Predicting the final cake moisture as a function of specific cake weight – obtained from i) – DCT (sec) and P_D (inHg):

$$M = f(W_S, DCT, P_D, \mu) \quad (2)$$

For disc filter application, the cake weight is determined by the length of time in which the filter is submerged and vacuum pressure is applied. At a given pressure, greater submersion time allows for greater accumulation on the filter surface (i.e., greater cake weight.) Likewise, for a given submersion time, increased pressured will also increase the accumulation of coal on the filter medium. To determine the relationship between time, pressure and cake weight, various amounts of coal slurry (30-35 % solids) were filtered at a pre-adjusted pressure. The cake formation times and pressure changes were recorded. As shown in Figure 3.1 and Figure 3.2**,** the specific cake weight (W_S) is plotted as a function of CFT x P_F for mixture and flotation product samples. When these parameters were plotted for the baseline condition, for RV dosage of 3lb/t, and for RU dosage of 3lb/t, power equations are obtained. Using these power equations, it is then possible to predict the amount of cake pickup or amount of cake generation during submersion as a function of CFT and P_F. From Figure 3.1, for example, W_S for the mixture sample in the presence of RV can be calculated according to the model if P_F and CFT are known:

$$W_S = 1.8961 \times (CFT \times P_F)^{0.6876} \tag{3}$$

Using the above and the other obtained power equations, W_S can be solved for both mixture and flotation samples in the absence and the presence of the dewatering aids.

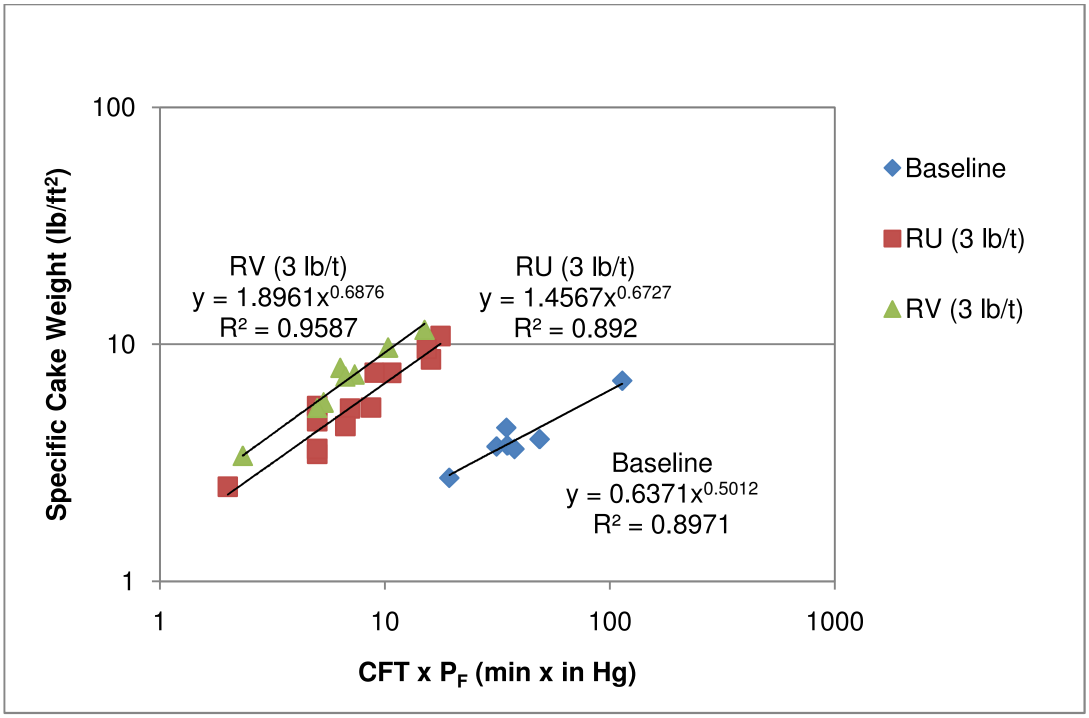


Figure 3.1 WS vs. CFTxPF for mixture sample

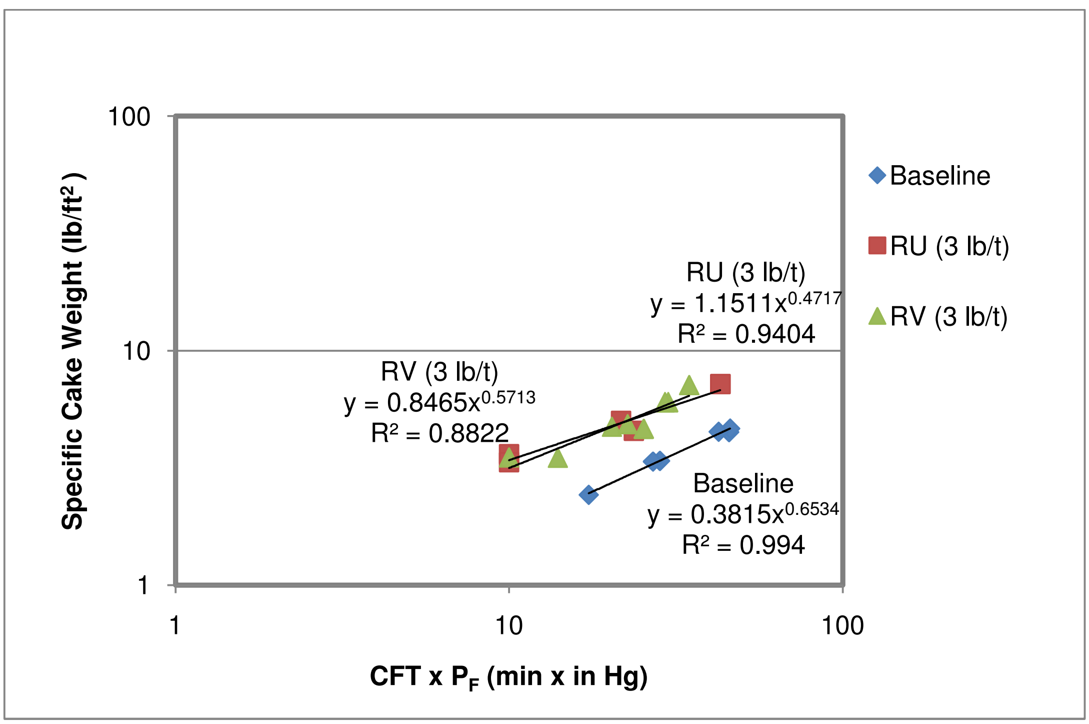


Figure 3.2 WS vs. CFTxPF for flotation sample

In filtration, cake formation is followed by cake drying which results in a product with a particular final moisture. Final cake moisture is controlled by the pressure during the drying cycle time (P_D) and the length of the drying cycle time (DCT), the weight of the cake (W_S) and the viscosity (μ) of the liquid. W_S is obtained from the Figure 3.1 and Figure 3.2. Empirically, a normalization factor is also determined as a function of P_D, DCT and μ:

$$NF = \frac{DCT \times P_D}{\mu} \tag{4}$$

Using the normalization factor and calculated specific cake weight, the best data fit is obtained when the final cake moisture is plotted as a function of the normalization factor and the specific cake weight:

$$M \propto \left(\frac{1}{W_S}\right)\left(\frac{DCT \times P_D}{\mu}\right) \tag{5}$$

When the above relationship was plotted, the equations in Figure 3.4 and Figure 3.3 were obtained. The plots include both the mixture and the flotation product. The equations then allow for calculation of the final cake moisture, which accounts for parameters during cake formation and cake drying cycles, and the specific cake weight. For instance, to predict the final cake moisture of the mixture sample in the presence of RV (at 3lb/t dosage), the following model equation can be used:

$$M = -4.779\ln(X) + 25.898 \tag{6}$$

Substituting X with (5), the equation becomes

$$M = -4.779\ln\left(\left(\frac{1}{W_S}\right)\left(\frac{DCT \times P_D}{\mu}\right)\right) + 25.898 \tag{7}$$

As demonstrated, the developed model can effectively simulate cake filtration for a vacuum filter application, and the final cake moisture can be predicted for a given type of coal.

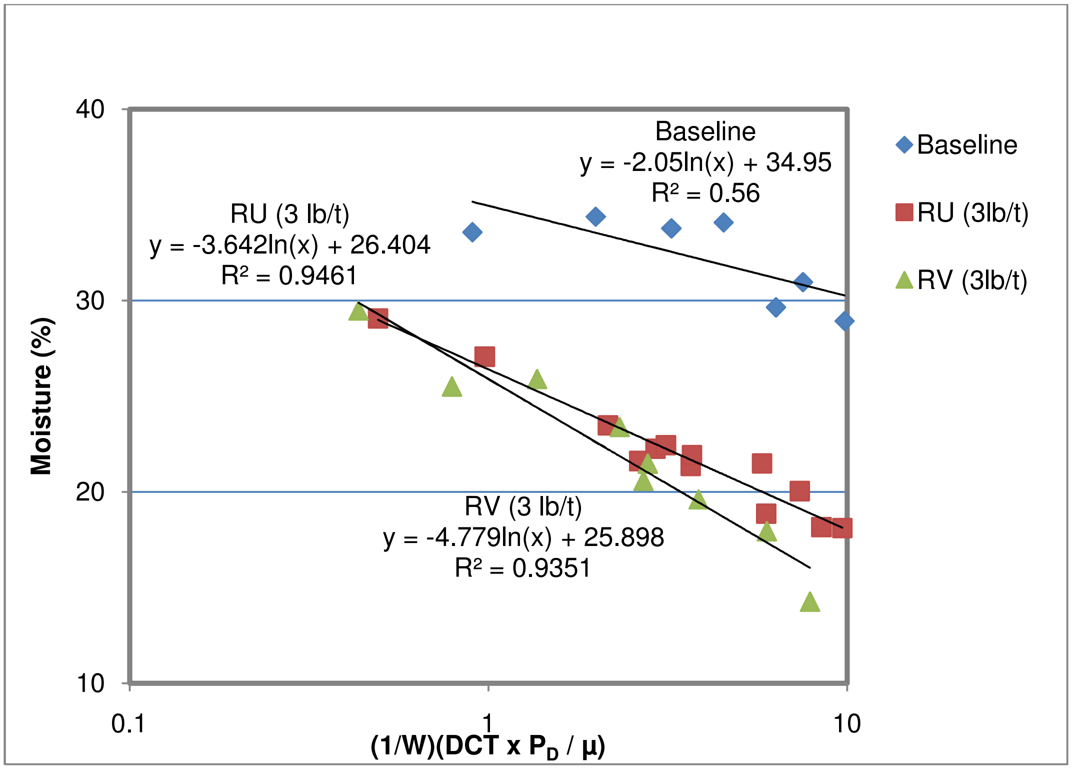


Figure 3.3 Moisture(%) vs. $(1/W)(DCTxP_D/\mu)$ for mixture sample

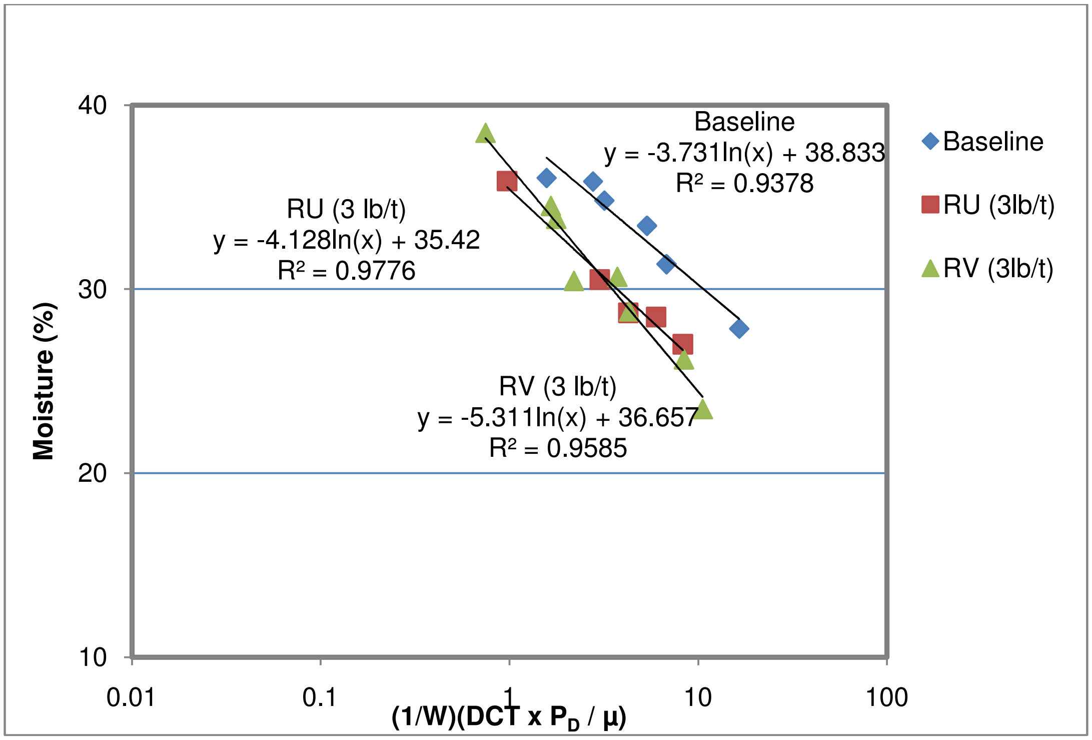


Figure 3.4 Moisture(%) vs. (1/W)(DCTxP_D/μ) for flotation product sample

Unlike the disc filters, for HBF application, the cake formation rate is not dictated by the pressure and CFT. Instead, the feed rate (t/hr) can be mechanically controlled by simply changing the feed pump speed to the filter, which in turn determines the throughput of the filter, and thus W_S. For this reason, if the pressure is kept constant, the HBF's moisture is mainly controlled by the cake formation time and dry cycle time. The filtration time, sum of the cake formation time and dry cycle time, can be controlled by changing the belt speed. At a set pressure and feed rate, slower belt speed may generate thicker cakes with relatively high cake moistures; whereas higher belt speed may result in forming thinner cake with low cake moistures. To determine the final cake moisture for RV-treated mixture sample at a given feed rate and pressure, the dry cycle time should be determined. This time the difference between the cake formation time and the total filtration time (TFT). The cake formation time can be predicted from the equations that were obtained from Figure 1:

$$CFT = \left(\frac{1}{P_F}\right) \times \left(\frac{W_S}{1.8961}\right)^{\frac{1}{0.6876}} \tag{8}$$

and

$$TFT = CFT + DCT \tag{9}$$

Substituting (8) into (9) and rearranging:

$$DCT = TFT - \left(\left(\frac{1}{P_F}\right) \times \left(\frac{W_S}{1.8961}\right)^{\frac{1}{0.6876}}\right) \tag{10}$$

So, once the dry cycle time is predicted from (10), the final cake moisture for a horizontal belt filter can also be predicted using (7).

Table 3.6 and Table 3.7 show examples for specific cake weight, dry cycle time and final cake moistures for mixture and flotation product samples. Both the disc and the horizontal belt filters are modeled.

Table 3.6 Simulation results on Mingo Logan's mixture sample

	Vacuum Disc Filter			Vacuum Horizontal Belt Filter		
	P_F	20	InHg	P_D	20	InHg
	P_D	20	InHg	P_F	20	InHg
	CFT + DCT	2	min	CFT + DCT	2	min
	CFT (%)	0.33		CFT	0.054	min
	DCT (%)	0.67		W	2	lb/ft2
RV (3 lb/t)	**Weight**	**11.26**	**lb/ft^2**	**DCT**	**1.95**	**min**
	Moisture	**24.50**	**(%)**	**Moisture**	**17.15**	**(%)**
	P_F	20	InHg	P_D	20	InHg
	P_D	20	InHg	P_F	20	InHg
	CFT + DCT	2	min	CFT + DCT	2	min
	CFT (%)	0.33		CFT	0.585	min
	DCT (%)	0.67		W	2	lb/ft2
Baseline	**Weight**	**2.33**	**lb/ft^2**	**DCT**	**1.42**	**min**
	Moisture	**32.07**	**(%)**	**Moisture**	**31.52**	**(%)**
	P_F	20	InHg	P_D	20	InHg
	P_D	20	InHg	P_F	20	InHg
	CFT + DCT	2	min	CFT + DCT	2	min
	CFT (%)	0.33		CFT	0.080	min
	DCT (%)	0.67		W	2	lb/ft2
RU (3 lb/t)	**Weight**	**8.32**	**lb/ft^2**	**DCT**	**1.92**	**min**
	Moisture	**24.97**	**(%)**	**Moisture**	**19.76**	**(%)**

Table 3.7 Simulation results on Mingo Logan's flotation sample product

	Vacuum Disc Filter			Vacuum Horizontal Belt Filter		
RV (3 lb/t)	P_F	20	InHg	P_D	20	InHg
	P_D	20	InHg	P_F	20	InHg
	CFT + DCT	2	min	CFT + DCT	2	min
	CFT (%)	0.33		CFT	0.225	min
	DCT (%)	0.67		W	2	lb/ft2
	Weight	**3.72**	**lb/ft^2**	**DCT**	**1.77**	**min**
	Moisture	**31.97**	**(%)**	**Moisture**	**27.18**	**(%)**
Baseline	P_F	20	InHg	P_D	20	InHg
	P_D	20	InHg	P_F	20	InHg
	CFT + DCT	2	min	CFT + DCT	2	min
	CFT (%)	0.33		CFT	0.631	min
	DCT (%)	0.67		W	2	lb/ft2
	Weight	**2.07**	**lb/ft^2**	**DCT**	**1.37**	**min**
	Moisture	**32.92**	**(%)**	**Moisture**	**32.66**	**(%)**
RU (3 lb/t)	P_F	20	InHg	P_D	20	InHg
	P_D	20	InHg	P_F	20	InHg
	CFT + DCT	2	min	CFT + DCT	2	min
	CFT (%)	0.33		CFT	0.161	min
	DCT (%)	0.67		W	2	lb/ft2
	Weight	**3.91**	**lb/ft^2**	**DCT**	**1.84**	**min**
	Moisture	**31.95**	**(%)**	**Moisture**	**27.98**	**(%)**

3.5. SUMMARY AND CONCLUSIONS

The complexities associated with data and general performance analysis of coal dewatering processes have typically required an abundance of tests to predict final moistures under a given set of conditions. As such, an empirical model was developed for vacuum filtration applications based on data obtained from two comprehensive tests. The tests were conducted on two samples with different size distributions by varying parameters such as dry cycle time, vacuum pressure during cake formation, and cake drying time, formation time, cake weight and thickness. Additionally, different types and dosages of dewatering aids were utilized. Based on the test data, models were established and a scale-up simulation was created for vacuum disc filter and horizontal belt filter.

Simulation results indicated that dewatering aids are most effective when used in horizontal belt filtration due to increased dewatering kinetics, which allow the filter to employ a longer cake drying time. This also allows more flexibility in controlling the cake moisture and throughput capacity when the horizontal belt filter is utilized. On the contrary, when using disc filters, the total filtration time is almost equally shared (1:1 ratio) between cake formation and drying times. Thus, the disc filter is not as effective at dewatering. It was also found that cake thickness may sometimes be deceiving because of increased cake porosity caused by dewatering aid addition. A more reliable evaluation can be made, however, by using specific cake weight instead of cake thickness in the simulation.

3.6. REFERENCES

1. Smith, M.R., Solid-Liquid Separation Process and Technology: By L. Svarovsky. Handbook of Powder Technology, vol. 5, 1985, 147 pp., Chemical Engineering Science, 1987. 42(11): p. 2802-2803.
2. Holdich, R.G., Solid-Liquid Separation Equipment Selection and Modelling. Minerals Engineering, 2003. 16(2): p. 75-83.
3. Akers, R.J., Solid-Liquid Separation : Edited by L. Svarovsky, 1978. 15(1): p. 77-79.

CHAPTER 4 INDUSTRIAL DEMONSTRATION OF DEWATERING AIDS

4.1. INTRODUCTION

4.1.1. General Background

The United States possesses approximately 25.1 % of the world's coal reserves with 400 coalfields and small deposits. Coal alone is the largest source of fuel for domestic energy production, and 90% of that energy source is consumed in power plants. Almost 52% of the domestic electric power generation is supplied by coal, which accounts for about 33% of the total energy production. Coal is also used directly by the manufacturing industries that produce chemicals, cement, paper, ceramics and various metal products. [1, 2]

Annually, more than 1.1 billion tons of coal is mined in the United States. About 600-650 million tons are processed yearly, and, typically, 350-400 million tons are handled in wet processing. Most of the coal preparation plants only process fine coal from the size fraction greater than 100 mesh (150 micrometer). For large size material, 16 X 100 mesh (1.0 X 150 micrometer), water-only cyclones and spirals are utilized. Depending on the quantity of the material present, material smaller than 100 mesh in the total plant feed may either be treated or discarded. If it is not treated, the fine material – material smaller than 100 mesh – is sent to a thickener. Then, it is pumped to a slurry impoundment. If the fine material is to be treated, classifying cyclones are utilized to remove the coal particles finer than 325 mesh (45 micrometer), and the 100 X 325 mesh size material is treated by utilizing flotation; however, processing and associated costs are significantly increased when dealing with the finer material. This fact has been a necessary limiting factor for coal processors, sometimes leaving no economical choice but to discard the fines to a slurry impoundment. Increased mechanization in the mining industry has also decreased the selectivity and increased the volume of refuse. As a result, 70-90 million tons of fine coal is discarded yearly to refuse impoundments. As of 2001, approximately 500-800 million tons of coal had been discarded in 713 active waste impoundments. In the United States, the majority of the coal waste impoundments are found in the eastern states – mainly in West Virginia, Pennsylvania, Kentucky, and Virginia.[1-6]

The coal waste impoundments have usually been considered permanent disposal sites; however, it can also be viewed as an unexploited energy resource, representing a loss of profit. In the past, recovery of fine coal was not as efficient, resulting in many older slurry impoundments containing significant amounts of coal refuse larger than 28 mesh (600 micrometer). Today, processing technologies have improved, and efforts have been made to reduce the coal loss by increasing the quality control during mining, optimizing the processing systems by improving fine coal recovery, minimizing the mass of solids for disposal, and utilizing dewatering effectively. As a result, the fraction of material smaller than 100 mesh in refuse slurries has increased, and the amount of coal being deposited in refuse slurries has decreased; however, the coal present in impoundments still represents a massive recoverable energy value. [2, 4-7]

Typically, if an impoundment contains at least 1 million tons of in-situ slurry, a recovery rate of at least 30% of a marketable, fine coal product from the slurry can prove to be a profitable venture. For this reason, re-mining impoundments can be considered as a promising method to recover coal and reduce the slurry volume. Many successful and unsuccessful efforts have been made over the last twenty to fifty years to re-mine the numerous waste sites. There are a number of impoundments that are already being mined, and recently, increases in coal prices have started to improve the overall economics of waste coal recovery.[4-6, 8]

4.1.2. Smith Branch Impoundment

The Smith Branch impoundment is located in Wyoming Country, West Virginia. At the time of this project, Beard Technologies Inc. owned the permit to re-mine and recover coal from this site. The impoundment currently serves the Pinnacle mining complex, which is now owned by Cleveland Cliffs Mining. The waste coal impoundment is estimated to contain 2.85 million

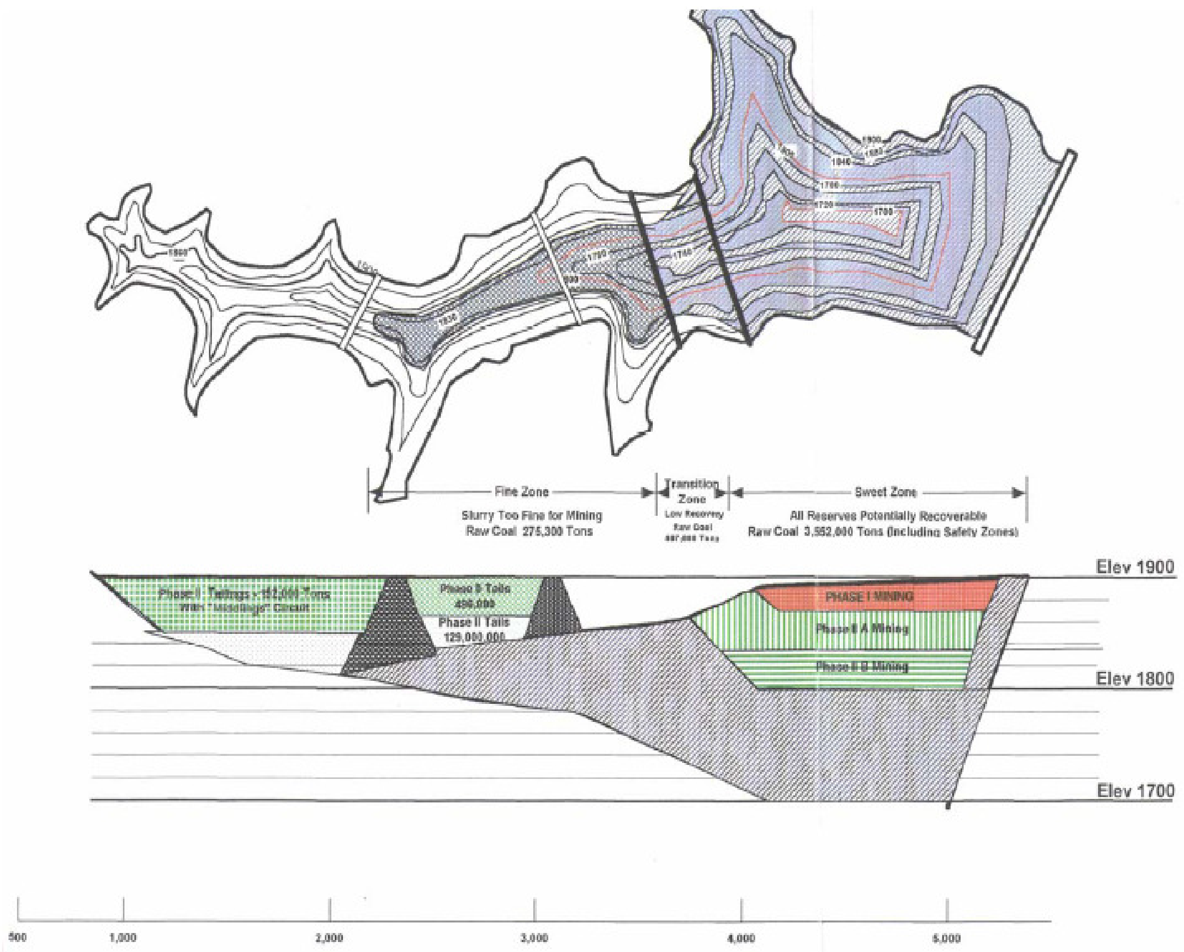


Figure 4.1. Pinnacle waste coal impoundment cross-section from vibracore analysis

tons of potentially recoverable fine coal (see Figure 4.1). The impoundment is an active site and continues to receive coal slurry refuse from the existing Pinnacle preparation plant as thickener underflow. Approximately 200,000 tons of additional fine coal is discharged into the impoundment annually by the existing Pinnacle preparation plant.

This site was ideally suited for a demonstration project since the dewatering aids made it possible to convert the waste coal impoundment at this site from environmental liability into a profitable resource.

4.1.3. Preliminary Analysis

Considering the amount of the fines that are going to be fed from the waste pond to the preparation plant, the success of this project was strongly depended on an effective cleaning and dewatering operation. Thus, Virginia Tech personnel conducted preliminary assessments on equipment specification, preliminary circuit design and preliminary cost analysis. A conceptual circuit design was also developed by personnel at Virginia Tech using standard process design and cost estimation procedures (Figure 4.2).

The circuit was developed to recover the waste coal fines therefore it included an advanced flotation processes together with the dewatering technologies developed. The flowsheet was designed based on average feed rate (dry basis) of 103 tph for the pond reclaim and was assumed to operate for two 8-hour shifts per day and 250 days per year with 90% availability. The projections indicated that the POC circuit installed for pond reclamation would produce clean coal around 68.7 tph and required two 8-disc filters to achieve the target capacity of 34.0 tph. The raw plant feed will be screened at 6.3 mm and the oversize discarded. The

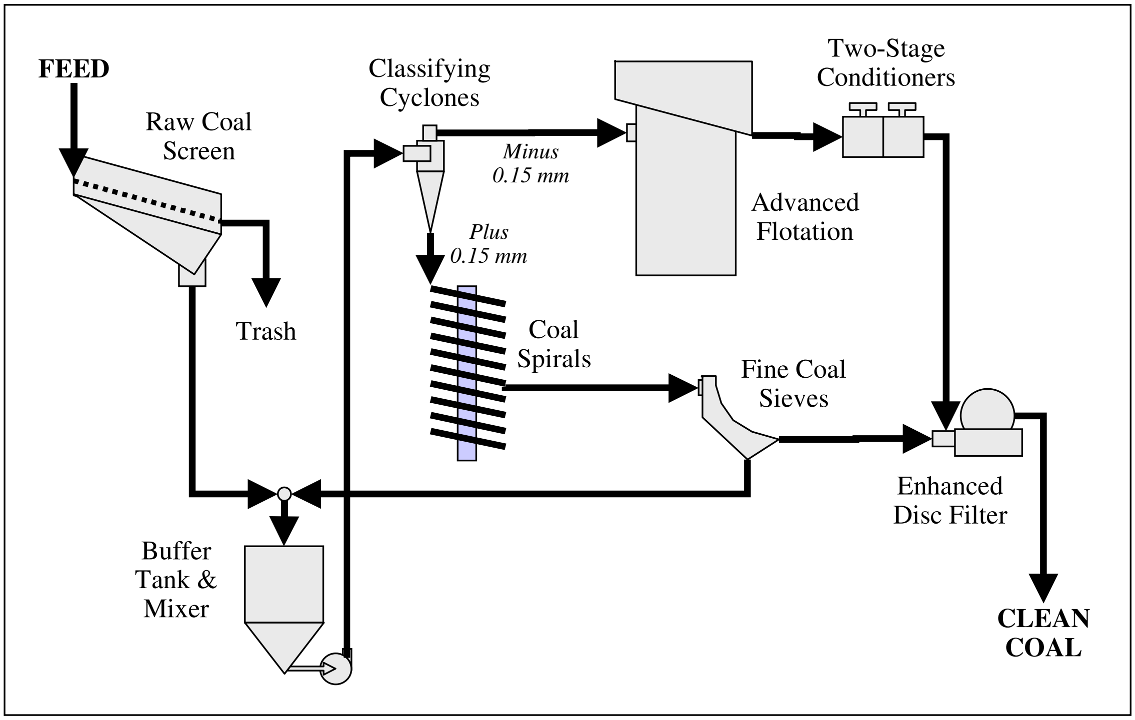


Figure 4.2. Generic POC flowsheet for treating pond reclaim material

minus 6.3 mm fraction will be directed to a buffer tank that feeds a bank of 14-inch classifying cyclones.

The coarse (6.3 x 0.25 mm) underflow from the cyclones will be treated by coal spirals, while the fine (minus 0.25) overflow will be passed to the advanced flotation circuit. The spiral and flotation circuits have been designed to handle 90 tph and 98 tph, respectively. The clean coal froth will be conditioned with appropriate dewatering reagents and passed along with the clean coal spiral product to the enhanced disc filter. Analyses conducted in this project indicate that the disc filter will produce approximately 68 tph of clean coal with 22% total moisture content.

The results of the cost-benefit analyses (done by Virginia Tech personnel) for the POC circuit showed that a payback time of slightly more than 6 years were estimated

4.2. PROOF-OF-CONCEPT DEMONSTRATION

Based on the feasibility studies described above and Chapter 2, a decision was made to demonstrate the capabilities of the novel dewatering aids at the Smith Branch impoundment site by constructing a proof-of-concept (POC) facility.

4.2.1. POC Engineering and Circuit Design

The data compiled by Beard Technologies Inc. was used to quantify the amount of potentially recoverable coal present within the impoundment and to formulate the best possible processing strategies for sizing, cleaning and dewatering of the coal fines. Based on these analyses, a detailed flowsheet was prepared for the POC facility by E.T. Kilbourne and Associates of Kingsport, Tennessee. The flowsheet was prepared in close cooperation with personnel from

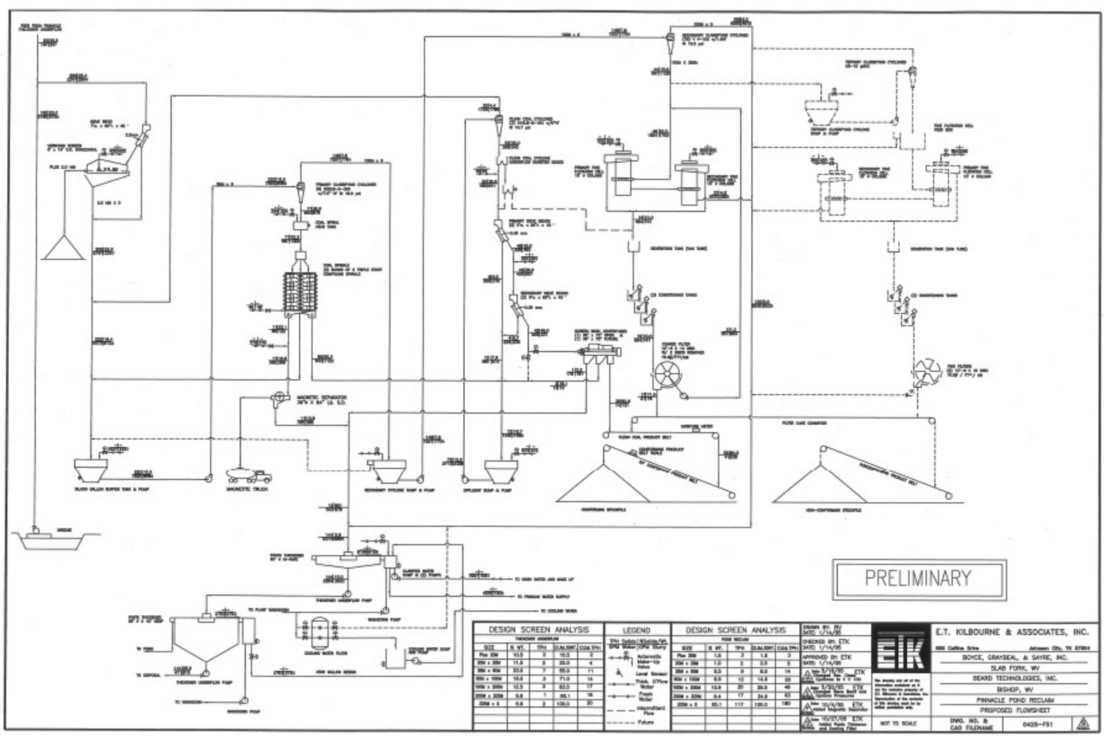


Figure 4.3 Engineering flowsheet for the POC-scale pond reclaim facility

Beard Technologies and Virginia Tech. The detailed flowsheet included mass and flow balances, equipment specifications, and size distributions for the various process streams. A proof copy of the POC flowsheet is provided in Figure 4.3 as an AutoCAD drawing.

Once the flowsheet was completed, detailed listings of required unit operations, such as equipment type, unit size, throughput capacity, reagent/chemical requirements, power requirements, air/water requirements, operating limitations, vendor cut-sheets, were produced by technical personnel at Beard Technologies. Detailed plant layout diagrams were prepared by Boyce, Graybeal and Sayre (BGS), Inc. of Slab Fork, West Virginia. The layout diagrams specified the physical arrangement of all primary operations, ancillary processing units, connecting streams, location of electrical wiring, arrangement of piping and plumbing, and other pertinent electrical/mechanical requirements. The engineering drawings and specifications were of sufficient detail to permit mechanical/electrical subcontractors to fabricate, construct, and assemble the proposed POC circuitry.

4.2.2. POC Fabrication and Installation

Bid packages were prepared for soliciting bids for major purchases of equipment, materials, fabricated components, and services necessary to complete the installation of the POC circuitry. Upon receipt, the bid packages were reviewed, and appropriate vendors were selected based on cost, availability, and suitability. This work included (i) fabrication of all required components associated with the various POC circuits, (ii) shipping of POC modules, ancillary equipment and construction materials to the POC site, (iii) inspection of all purchased POC modules, ancillary equipment and materials to ensure that they are of suitable workmanship and are structurally, mechanically and/or electrically operational, and (iv) preparation of operation, maintenance, and safety manuals for each unit operation.

After developing the flowsheet, the installation of all unit operations, piping, electrical wiring, and instrumentation were undertaken. The on-site construction work was contracted and managed by Boyce, Graybeal and Sayre (BGS), Inc. of Slab Fork, West Virginia. The fabrication and on-site construction activities required approximately one year to complete. The plant incorporates some of the most advanced technologies available to the coal preparation industry as POC circuits. The most significant of these included a two-stage advanced column flotation circuit for fine coal separation, a three-stage bank of agitated mixing tanks to condition the dewatering aids and a paste thickener to convert the fine high-ash wastes into a high-solids product for disposal with minimal environmental impact.

A simplified process flow diagram of the as-built plant incorporating the POC dewatering circuitry is provided in Figure 4.4. A photograph of the nearly completed plant is shown in Figure 4.5. The plant was designed with a raw feed nameplate capacity of approximately 200 tph of dry solids. The plant sizes/classifies the feed from the dredge into four nominal size fractions, plus28 mesh, 28 x 100 mesh, 100 x 325 mesh, and minus 325 mesh. The initial separation occurs on a sieve bend and single deck vibrating screen. The plus28 mesh material (screen oversize) is discharged to a stockpile outside the plant, and the minus 28 mesh reports to a 30,000 gallon surge tank. Material from the surge tank is pumped into 20 inch diameter classifying cyclones, which nominally sizes the feed at about 100 mesh.

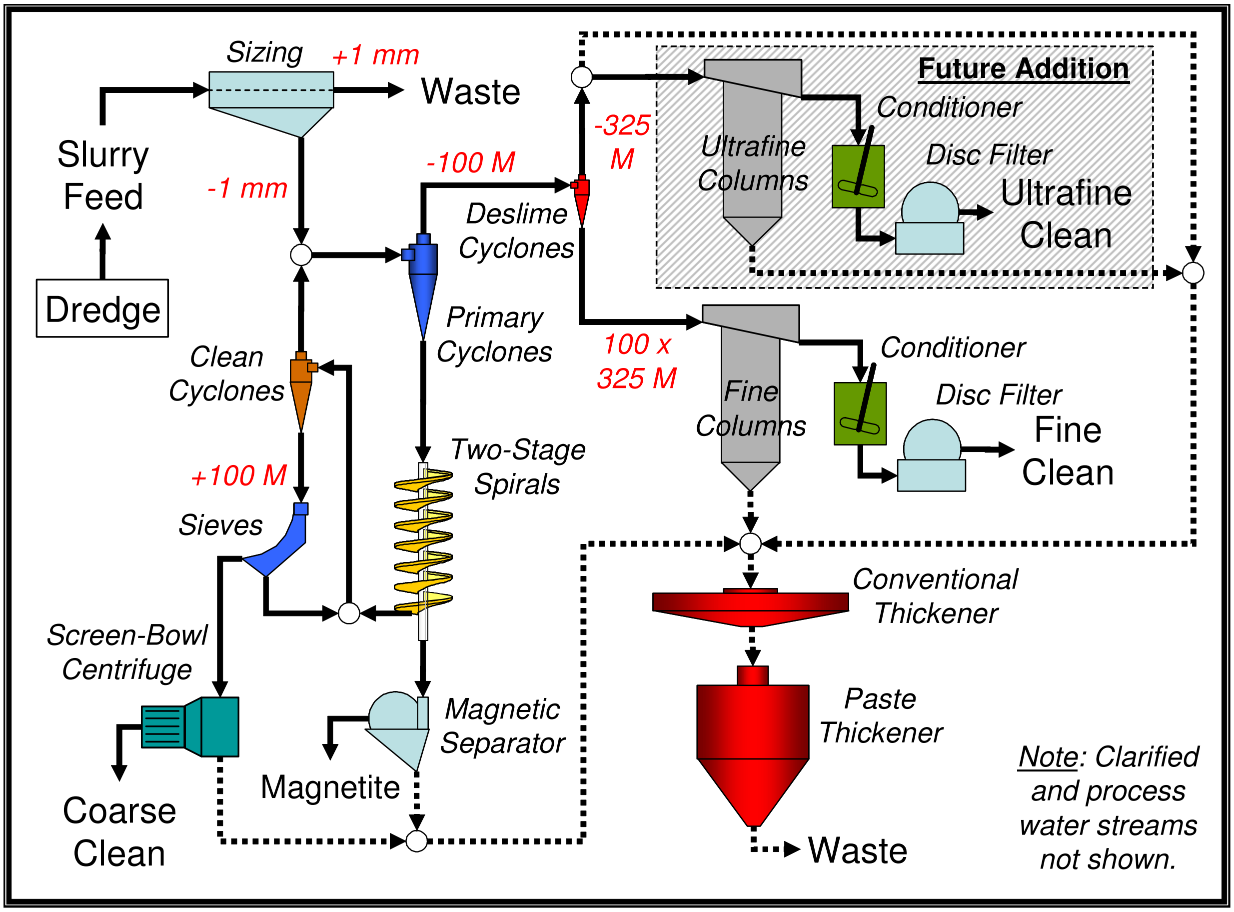


Figure 4.4 Simplified process flow diagram for the pond reclaim facility

The plus 100 mesh underflow from the classifying cyclones (Figure 4.6) reports to two banks of compound spirals. The spiral product is discharged to a clean coal sump and pumped to a bank of 15-inch diameter clean coal classifying cyclones. The underflow from the clean coal cyclones is passed across a two-stage rapped sieve bend system to remove high-ash minus 100 mesh solids from the spiral product. The clean coal product from the spirals is dewatered using two 36 x 72 inch triple-lead screen-bowl centrifuges (Figure 4.7). The dewatered product is dropped to a clean coal collection belt and transported to a radial stacker stockpile. The spiral

Figure 4.5 Nearly completed plant incorporating the POC circuitry

Figure 4.6 Classifying cyclones used to provide a 100 mesh cutsize

reject passes through a magnetic separator to recover any magnetite that may have been present in the slurry before being discarded as waste.

The minus 100 mesh overflow from the classifying cyclones is pumped through two banks of 4 inch diameter classifying cyclones (Figure 4.8). The "deslime" cyclones make a nominal size separation at 325 mesh. The deslimed underflow reports to a two-stage advanced

Figure 4.7 Screen-bowl centrifuge used to dewater 100x235 mesh product

Figure 4.8 Deslime cyclones used to perform a nominal 325 mesh cutsize

column flotation bank. The deslime overflow is currently discharged as waste, but provisions are included in the flowsheet design to incorporate a secondary advanced flotation bank, dewatering aid conditioners, and disc filter to recover the ultrafine coal that is now lost in this stream. The cleaned froth product from the column units passes into a de-aeration tank to promote breakdown of any residual froth.

The froth product is then passed through three agitated mixing tanks (Figure 4.9) where dewatering aids are added. After conditioning for several minutes, the treated slurry is fed to a bank of disc filters (Figure 4.10) for final dewatering. Previous test work conducted in this project has shown that adequate mixing, both in terms of time and intensity, is critical to the performance of the dewatering aids. The dewatered froth product is discharged to a reversible product collection belt. The coal product can be directed to either a clean coal collection belt or a noncompliant conveyor system, if the moisture of the filter product is higher than the specifications allow. The total product moisture is continuously monitored using an on-line

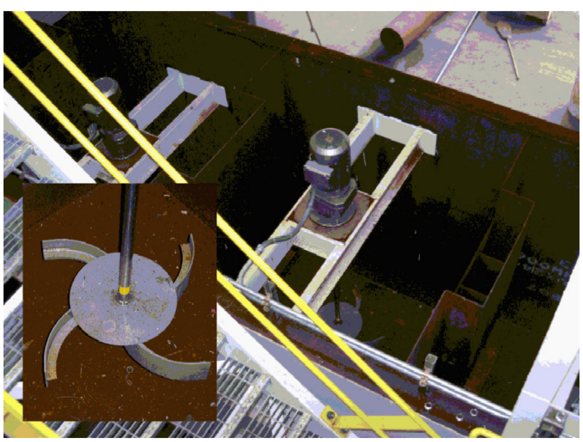


Figure 4.9 Three-stage conditions used for conditioning the dewatering aid

Figure 4.10 Bank of disc filters used to dewater the fine coal froth product

Figure 4.11 Static thickener used to thicken solids and clarify process water

Figure 4.12 Paste thickener used to further thicken wastes for disposal

moisture analyzer located on the clean coal collection belt.

The refuse from the various plant circuits is treated using a two-stage thickener system. The first stage consists of a 90 ft. diameter high-rate static thickener (Figure 4.11) into which coagulant and flocculant is added to promote aggregation and rapid settling of the fine solid waste. The overflow from this unit is taken back into the plant as clarified process water. The underflow is pumped to a 60 ft. diameter paste thickener (Figure 4.12) for secondary densification. The bed depth is maintained between 12 and 25 ft and has a retention time between 11 to 15 hrs.

4.2.3. POC Circuit Testing

a) Shakedown Testing

At the completion of the POC design and construction, preliminary shakedown tests were conducted to resolve operational problems that arose during start-up of the POC plant. Initial test runs were conducted to ensure that pumping capacities, pipe sizes, electrical supplies, control systems, and instrumentation were adequate. After completing start-up activities, exploratory tests were conducted to validate the design capacities of the various unit operations used in the POC circuits. Data obtained from these tests were used to identity key operating parameters that should be investigated in detailed testing. This work was followed by detailed testing of the plant circuitry and, in particular, evaluation of the fine coal vacuum filters where the dewatering aides were utilized.

In general, the in-plant testing of the POC circuitry showed that the sizing, cleaning, dewatering and disposal circuits performed as expected. During these tests, the plant was fed the design capacity of 200 ton/hr of raw feed and produced an average of 58 ton/hr of fine clean coal. Of this tonnage, 39 ton/hr was plus 100 mesh and 17 ton/hr was nominal 100 x 325 mesh. The final product (nominal plus 325 mesh fraction) generally met the product quality specifications of 5.5% ash, 0.8% sulfur, and 17% moisture; however, some fluctuations in product qualities were observed from time to time due to the widely varying characteristics of the feed material extracted from the impoundment. Size-by-size summaries of the performance data for the shakedown tests are provided in Figure 4.13 and Figure 4.14 for the coarse and fine plant circuits, respectively.

The test data indicate that the 20-inch diameter classifying cyclones provided a nominal cut size of 100 mesh. The underflow contained about 75.05% of the plus 100 mesh solids at an ash content of 7.70%. 24.95% of the plus 100 mesh solids containing 36.36% ash was misplaced material. The overflow contained 2.87% of the plus 100 mesh solids at 2.50% ash, which reported to the de-slime cyclone circuit. The compound spirals generated a coal product containing 80.15% plus 100 mesh at 2.98% ash, with 19.85% minus 100 mesh at 21.49% ash reporting with the product. The spiral middlings product was found to be of a sufficient quality

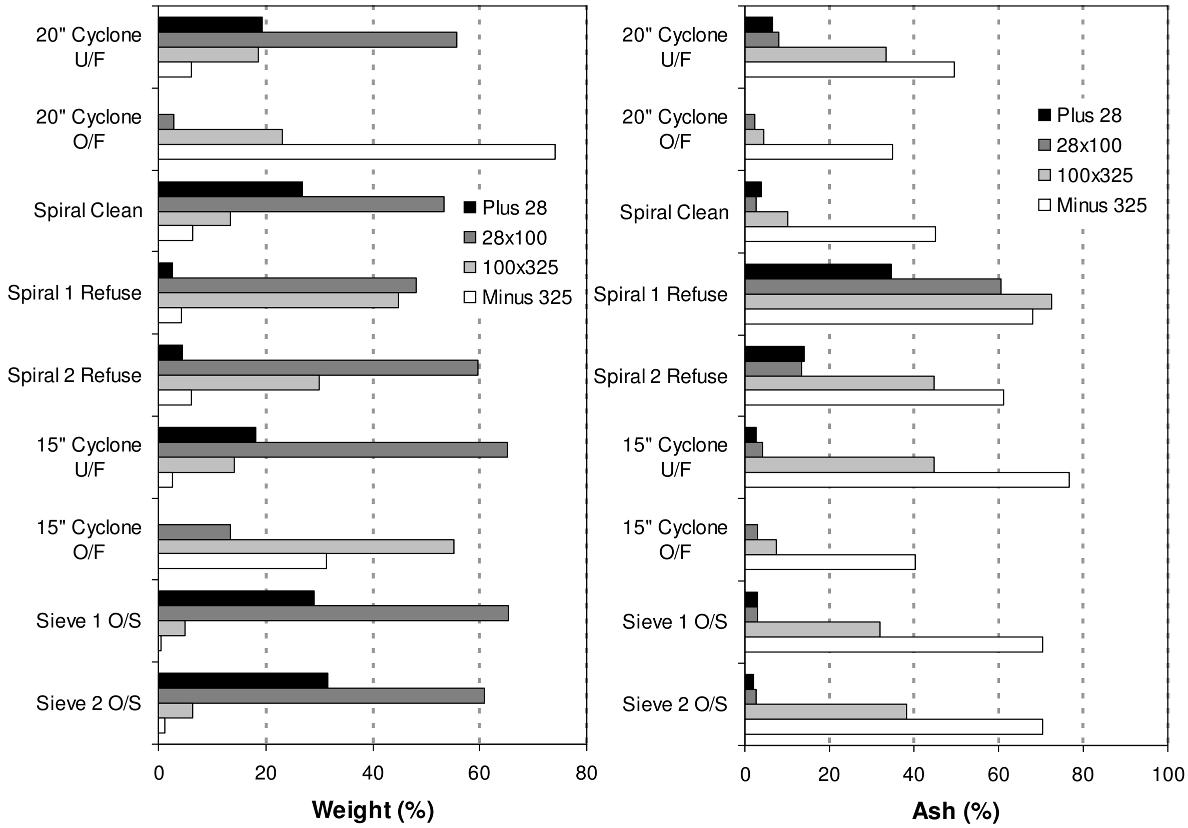


Figure 4.13 Weight and ash distributions for samples collected from the shakedown tests conducted on the coarse coal treatment circuits

to be recovered without additional treatment. So, the splitter was closed to direct this stream into the product. The primary and secondary spiral reject streams contained 50.73% and 64.11% plus 100 mesh solids at 59.08% and 13.42% ash, respectively, and 49.27% and 39.89% minus 100 mesh solids at 72.09% and 42.72% ash, respectively.

The spiral product was pumped into the clean coal cyclone circuit to provide desliming of the plus l00 mesh product. The circuit incorporates 15-inch diameter classifying cyclones and a two-stage rapped sieve bend system. The cyclone underflow contained 83.25% of plus 100 mesh solids at 4.10% ash. A total of 16.75% of minus 100 mesh solids at 49.73% ash were contained in this stream. The overflow, which reports to the de-slime cyclone circuit, contained 13.51% of plus 100 mesh at a 2.84% ash. The two-stage rapped sieve was used to remove minus 100 mesh contamination and associated ash from the coarse circuit product. The higher quality of the oversize product produced by the primary and secondary rapped sieve bends reduced the amount of minus 100 mesh in the product from 16.75% to 7.46% with a corresponding ash reduction from 11.74% to 5.47%. The undersize solids from the rapped sieve bend circuit were circulated back to the clean coal sump.

The product from the cyclone/spiral circuit was dewatered through two 36 x 72 inch screen bowl centrifuges. The coal product from the screen-bowl was 5.18% ash, which

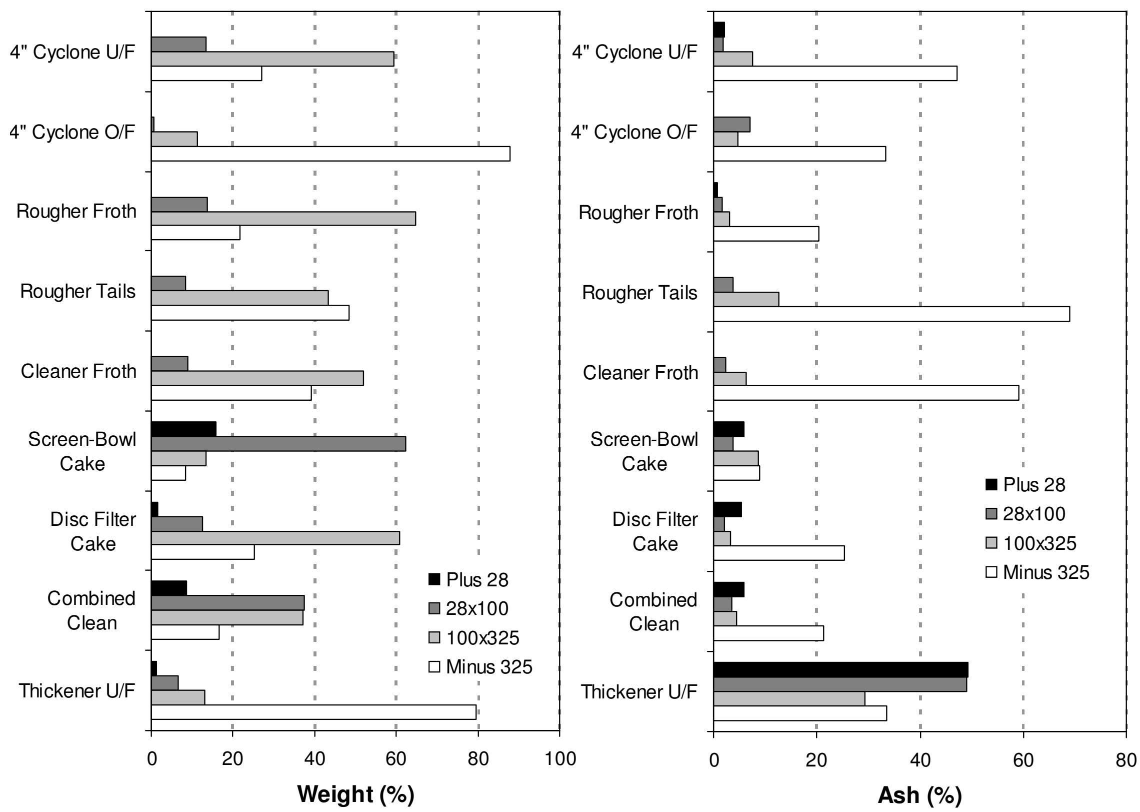


Figure 4.14 Weight and ash distributions for samples collected from the shakedown tests conducted on the fine coal treatment circuits

contained 78.29% plus 100 mesh at 4.17% ash and 21.71% minus 100 mesh at 8.82% ash.

The shakedown test data showed that the 4-inch diameter classifying cyclones provided a nominal cut size of 325 mesh. The deslimed underflow contained 73.06% plus 325 mesh (6.52% ash) and 26.94% minus 325 mesh (47.12% ash). The deslime overflow, which was discarded to the static thickener as waste, contained 12.05% plus 325 mesh (4.77% ash). The loss of low-ash material in this stream was expected because the ultrafine cleaning circuit had not yet been added to the plant circuitry. The clean froth product from the entire column flotation circuit was found to contain 6.66% ash. This product contained 78.34% plus 325 mesh at 2.84% ash and 21.66% minus 325 mesh at 20.45% ash. The rougher column reject (i.e., feed to the cleaner column) was found to contain 39.12% ash. Of the rougher column reject, 51.72% was plus 325 mesh (11.17% ash) and 48.28% was minus 325 mesh (69.05% ash). The ash content of the cleaner column froth was 26.69%, and of that percentage, 60.81% at a 5.72% ash is plus 325 mesh and 39.19% at 59.22% ash is minus 325 mesh. The high ash content of the minus 325 mesh fraction can be attributed to problems associated with the entrainment of fine clay in the column froth. As such, the froth quality is expected to improve after modifications are made to the wash water system. So, more minus 325 mesh clay is eliminated from the froth products.

As indicated previously, the product from the deslime cyclone/column flotation circuit is dewatered by using a twelve-disc 12.5 ft diameter vacuum disc filter. In the shakedown tests, the filter product was found to contain 8.76% ash (74.88% plus 325 mesh at 3.21% ash and 25.12% minus 325 mesh at 25.30% ash). The higher ash in the minus 325 mesh size fraction was found to have a major impact on both the final ash quality of the filter product and on the overall product moisture. On average, the moisture of the filter cake was found to be approximately 26% in the absence of dewatering aids. The addition of dewatering aids reduced the filter moisture to below 18%.

The 90 ft diameter, high-rate thickener required modifications to allow optimum operation during the shakedown tests. The required modifications were to raise the flume above the liquid level in the thickener unit to avoid surface froth entering into the clarified water tank, flatten the slope and increase the width of the flume for capacity to slow the velocity and minimize the agitation of the slurry material, and divert the slurry flow down into the center well at the end of the flume to reduce short-circuiting of slurry out of the center well into the main body of the thickener. The underflow from the static thickener provided the feed for the paste thickener. The paste thickener, which is still being subjected to shakedown testing, appears to be capable of handling the varying feed rates, size distributions, and qualities of waste slurry from the processing plant.

b) Detailed Testing

After completing the shakedown tests, several series of detailed tests were performed for the POC dewatering circuit. For comparison, a set of laboratory dewatering tests were conducted during the detailed testing. Both the laboratory and POC-scale tests were completed using RV. The laboratory filtration tests were conducted using a 2.5- inch diameter Buchner funnel. After establishing the optimum operating conditions and reagent blends using the laboratory filter, the dewatering aid was added to the feed of the three-stage conditioners that were located just ahead of the vacuum disc filter. As indicated previously, adequate mixing is critical because previous studies showed that the moisture reduction improves with increasing energy input during conditioning. For this reason, the full three-stages of conditioners were used in all of the POC-scale dewatering tests.

It was also found that the quality and size distribution of the coal within the impoundment varied greatly at different locations. These variations affect the cleaning and dewatering processes. For that reason, some parallel dewatering tests were also conducted during the plant testing. In this case, the sample was collected directly from the plant's cyclone overflow stream and floated at the lab using the Denver Flotation cell to produce the filter feed sample to be used in laboratory filtration tests. Table 4.1 shows the results of dewatering test using RW and RV as dewatering aids. The moisture reduction corresponds to an approximate 40% decrease; however, as shown below, the results are seven to eight points higher compared to previous results. This change in the moisture can be attributed to the particle size distribution. The flotation feed sample collected from the cyclone overflow contained approximately 70 % minus 325 mesh particles.

Another set of tests was conducted with another sample collected a different day. The results are shown in Table 4.2. The moisture reduction was again around 40%; however, the minus 325 percent material in the feed was too high. The reagents were greatly effective, but the sample was not representative of their effectiveness. To simulate the average plant filtration

Table 4.1 Effect of RV and RW (dissolved in diesel at 1:2 ratio) at various dosages

Reagent Dosage (lb/ton)	Moisture Content (%)		Moisture Reduction (%)	
	RW	RV	RW	RV
0	34.03	34.03	--	--
0.5	33.05	24.58	2.9	27.77
1.0	26.03	22.91	23.50	32.68
3	23.97	20.76	29.56	38.99
5	23.33	20.15	31.44	40.79

Table 4.2 Effect of RV and RW (dissolved in diesel at 1:2 ratio) at various dosages on dewatering of floated sample

Reagent Dosage (lb/ton)	Moisture Content (%)		Moisture Reduction (%)	
	RW	RV	RW	RV
0	32.92	32.92	--	--
3	20.59	19.21	37.45	41.67
5	19.27	19.14	41.46	41.86

operation the sample was deslimed by removing either all (totally deslimed) or just two-thirds (partly deslimed) of the minus 325 mesh solids from the test sample. Table 4.3 shows that the deslimed samples gave moisture contents below the target value of 17% moisture.

Table 4.3. Effect of RV (dissolved in diesel at 1:2 ratio) at various dosages on dewatering of deslimed samples

Reagent Dosage (lb/ton)	Moisture Content (%)		Moisture Reduction (%)	
	Total Deslimed	Partly Deslimed	Totally Deslimed	Partly Deslimed
0	20.90	26.41	--	--
3	13.22	17.01	36.75	35.56
5	13.08	16.99	37.42	35.67

4.2.4. Scale-Up Assessment

A comparison of the laboratory and POC-scale filtration test results is given in Figure 4.15. Because the baseline moisture was different in each case (i.e., 26% vs. 24%), the data have been plotted again in Figure 4.16 as percentage moisture reduction for each reagent dosage. The experimental data clearly demonstrate that the addition of dewatering aid substantially reduced the moisture contents of the filter products. In the POC-scale tests, the total moisture content

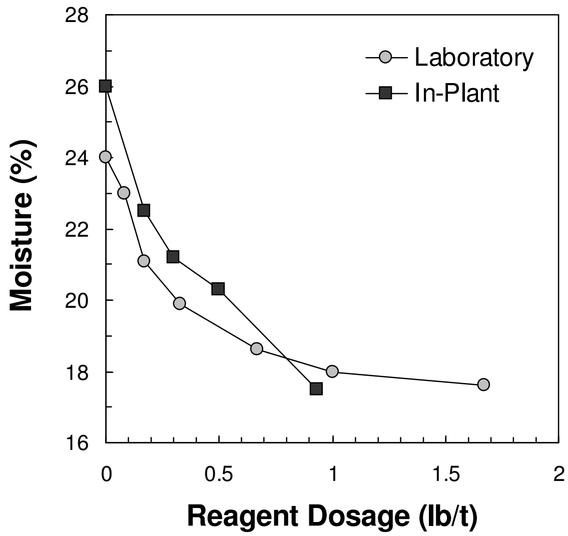


Figure 4.15 Moisture content versus dewatering aid dosage (RV)

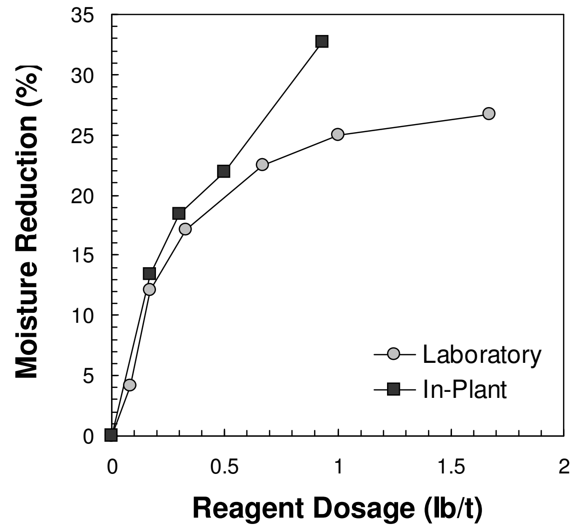


Figure 4.16 Moisture reduction versus dewatering aid dosage (RV)

was reduced from 26% to 20% at a dosage of approximately 0.5 lb/ton and farther down to 17.5% at about 1 lb/ton. The 17.5% moisture represents a moisture reduction of nearly one-third compared to the baseline moisture of 26%. For example, the cake thicknesses observed in some of the POC-scale tests were as large as 3 inches. These POC-scale results compare favorably with the laboratory data. The apparent gap between the laboratory and POC-scale results obtained at higher dosages of dewatering aid may be explained by differences in particle size, cake thickness, drying time, etc., used in the two test programs.

Another important observation made during the detailed test program was that the power draw for the disc filter vacuum pumps dropped dramatically upon addition of the dewatering aid. An example of this behavior is shown in Figure 4.17 for one of the test runs performed at the plant. The power draw dropped from a normal baseline value of about 160 to 115 amps after the

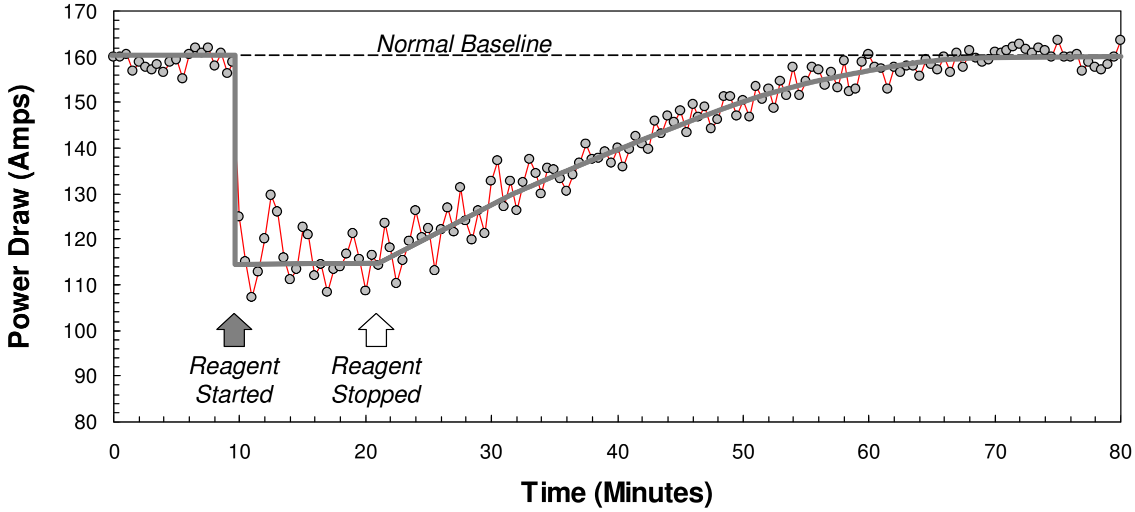


Figure 4.17 Effect of dewatering aid addition on vacuum filter pump power demand

addition of the dewatering aid. This represents a power savings of nearly one-third for the vacuum filter system.

4.3. SUMMARY AND CONCLUSIONS

In the present work, capabilities of novel dewatering aids were successfully tested on the fine coal that is processed at the Smith Branch impoundment site. In United Stated, annually, approximately 1.1 billion tons of coal is mined and 600-650 million tons of the production is wet processed while remainder is utilized as is. Typically, wet processing methods are responsible for processing 350-400 million tons of coal. However, especially for fine materials smaller than 100 mesh, cleaning and dewatering costs have significantly been high. This fact has been a limiting factor for the coal processing industry. As a result, depending on the quality and the quantity of the fine size fraction, coal has been either treated and sold or discarded to waste slurry impoundments.

As of 2001, there are 713 active waste slurry impoundments in which approximately 500-800 million tons of coal lies. Considering today's economical conditions however, the coal in the waste slurry impoundments may now be an unexploited energy source. Consequently, re-mining the impoundments may be considered a promising method to recover this coal which previously represented a loss of profit.

The Smith Branch impoundment, located in West Virginia, is estimated to contain 2.85 million tons of potentially recoverable fine coal. At the time of this work, Beard Technologies Inc. owned the permit to re-mine and recover coal from this site. Considering the amount of fine size fraction, the success in recovering the coal from this impoundment was strongly dependent on successful cleaning and a very effective dewatering operation. Thus, this site was ideally suited for a demonstration project since the novel dewatering aids made it possible to convert the waste coal impoundment at this site from environmental liability into a profitable resource.

Based on the feasibility studies, Beard Technologies constructed a POC circuit. The plant was designed with a raw feed capacity of approximately 200 tph of dry solids. The plant sizes/classifies the feed from the dredge into four nominal size fractions, plus28 mesh, 28 x 100 mesh, 100 x 325 mesh, and minus 325 mesh. The circuit incorporates the novel dewatering technology together with advanced flotation process to ensure maximum moisture reduction and coal recovery for the 100 x 325 mesh material.

Cleaning is done by a two-stage advanced column flotation bank. The cleaned froth product from the column units passes into a de-aeration tank to promote breakdown of any residual froth. The froth product is then passed through three agitated mixing tanks where dewatering aids are added. Previous test work has shown that adequate mixing, both in terms of time and intensity, is critical to the performance of the dewatering aids. After conditioning for several minutes, the treated slurry is fed to a bank of a twelve-disc, 12.5ft diameter vacuum disc filter for final dewatering.

In the POC-scale tests, the total moisture content was reduced from 26% to 20% at a dosage of approximately 0.5 lb/ton and further to 17.5% at approximately 1 lb/ton of novel dewatering aid; this represents a total moisture reduction of nearly 33%. As well, dewatering kinetics were also increased i.e., the cake thicknesses observed in some of the POC-scale tests were as much as 3 inches. These POC-scale results also compare favorably with the laboratory data.

Another important observation made during the detailed test program was that the current required for the disc filter vacuum pumps dropped dramatically upon addition of the dewatering aid. The normal baseline current value was about 160 amps, but fell to 115 amps after the addition of the dewatering aid. This represents a power savings of nearly 30% for the vacuum filter system.

In conclusion, testing of the POC-scale facility demonstrated that the novel dewatering aids made it possible to achieve low moisture contents and also improved the handling characteristics of fine coal products produced via re-mining of a waste coal impoundment.

4.4. REFERENCES

1. Administration, Energy .Information. Official Energy Statistics from the U.S Government. 2009
Available from:http://www.eia.doe.gov/.
2. Energy, U.S.DOE. 2009
Available from:http://www.energy.gov/energysources/coal.htm.
3. Svarovsky, Solid-Liquid Separation. 1991, London: Butterworth.
4. Clean-Energy.US., About Coal. 2009
Available from:http://www.clean-energy.us/facts/coal.htm.
5. World Coal Institute, Coal Facts 2007. 2009
Available from:http://www.worldcoal.org/.
6. Coal Impoundment Location & Information System,. List Coal Impoundments. 2009
Available from:http://www.coalimpoundment.org/.
7. J. Gregory, Solid-liquid separation, ed. 1984,
8. Dickenson, T.C., Filters and Filtration Handbook. 4 ed. 1997:

CHAPTER 5 DEWATERING OTHER MINERALS

5.1. INTRODUCTION

In the previous chapters, the feasibility of using novel dewatering aids was investigated in detail. These dewatering aids adsorb on the surface of the hydrophobic particles such as coal, and increase the hydrophobicity. The results obtained from laboratory, pilot, and full scale tests that were conducted on several coal samples confirmed that the dewatering aids are capable of decreasing the final cake moisture by 20-50 % and increasing the filtration kinetics to a point that cannot be achieved with existing mechanical capabilities. In order to investigate and confirm that the novel dewatering aids can also be effective on other minerals and hydrophilic particles, a number of dewatering tests were conducted on minerals such as copper and hydrophilic materials such as kaolin clay and fly ash.

5.2. KAOLIN CLAY

5.2.1. Background Information

Kaolin clay, a hydrated aluminum silicate ($Al_2Si_2O_5(OH)_4$), is a white, soft mineral and is mainly composed of fine-grained, plate-like particles. It is commonly referred to as "China Clay," which refers to its discovery in Kao-Lin, China. Kaolin clay deposits are classified as either *primary* or *secondary*. *Primary* kaolin results from residual weathering or hydrothermal alteration of feldspar and muscovite. *Secondary* kaolin is sedimentary in origin.[4, 1, 2]

Kaolin is a distinctive industrial mineral due to its unique physical, physiochemical, and chemical properties. For example, it remains chemically inert over a relatively wide pH range, it is non-abrasive, and it has low thermal and electrical conductivities. As a result, it has broad applications in many industries including (by percent): paper and filling (45%), refractory (16%), ceramics (15%), fiberglass (6%) cement (6%), rubber and plastic (5%) and paint catalyst and others (7%). In the paper industry, high-brightness kaolin is used for coating, and low-brightness kaolin is used as filler.[4, 5, 6]

The total global kaolin production is estimated (2007) to be 37.8 million tons per year, and the US has approximately 20% of the clay production share in the world markets. The other major kaolin producers are Brazil, Czech Republic, Germany, Republic of Korea and United Kingdom. In the US, a stream of kaolin-containing rocks reaches from Eufaula, Alabama approximately 250 miles to Aiken, South Carolina. This stream contains 7 to 10 billion metric tons of sedimentary kaolin. 90% of US kaolin production is concentrated in a 150-mile-long segment of this stream that extends from Macon, Georgia to Aiken, South Carolina. Georgia is responsible for producing 92% of the tonnage and 96% of the dollar value, while South Carolina produces 4% of the tonnage and 2% of the dollar value. Kaolin from this region of the US is of high quality due to its high brightness and relatively low viscosity at high solids concentration (70%).[2-4, 7]

By and large, all kaolin is mined using open pit methods utilizing shovels, draglines, and backhoes. Two processing methods – one dry, the other wet – are used in the production of kaolin. The dry process is rather simple and inexpensive, while the wet process employs sophisticated separation techniques to improve the color brightness and size consistency.[1, 5]

In the wet process, kaolin clay is treated with magnetic separation, flotation, selective flocculation, leaching, or combinations of multiple methods, and usually contains more than 80% moisture content. The most difficult phase of processing is dewatering of this moisture content, due to the naturally ultra-fine particle size of the clay. Fine particles require more complex and intense chemical and mechanical treatment due to the larger surface area to be dewatered and smaller capillaries. This, along with the kaolin clay's natural hydrophilic characteristics, gives higher capillary pressures. In industry, kaolin clay is dewatered using vacuum drum filters and various types of pressure filters, which can lower the moisture to 40% to 60%. In kaolin clay dewatering, a unique function of the filtration process is also to remove chemicals from the clay. Depending on the market requirements the filtration is followed by spray drying, which, although costly, is the only method capable of producing a very low moisture product (approximately 10% to 30%).[4, 8-10]

In this section, the effects of novel dewatering aids were investigated using two new methods, namely Two Step Hydrophobization and Foam Assisted Dewatering Method.

5.2.2. Two Step Hydrophobization Method

a) *Experimental*

A number of kaolin clay samples with various properties were obtained from different companies for use in dewatering tests. The tests utilized a two-step hydrophobization technique, described below. The samples are:

i) Thiele Kaolin Company's Kaolin Clay Sample (25% solid, pH 7.2)

ii) J.M.H Corporation's PSD Filter Vat Clay Sample (30% solid, pH 4)

iii) J.M.H Corporation's PSD Filter pre-leached Clay Sample (30% solid, pH 9.5)

b) *Results and Discussion*

1) Thieles' Sample (30% solid at pH 7)

In the first set of laboratory dewatering tests, East Georgia kaolin clay sample received from Thiele Clay Company was subjected to dewatering tests using the two-step hydrophobization method. The as-received sample was first subjected to some simple analyses, such as pH value and solid content determination. The measured pH value for the clay slurry sample was 7.14 and had 25% solids with a size that was finer than 2μm. In this set of tests, a laboratory pressure filter unit was used for all tests at 60psi pressure. The as-received sample was first agitated for a given period of time to keep the clay particles suspended and well-dispersed in the slurry, followed by dilution using Blacksburg tap water from 25% to 8%. In the first dewatering test, a known volume of the diluted sample was taken and tested at 60psi pressure without any surface treatment. During this control test, the dewatering kinetics were noticeably low, giving approximately 25 minutes cake formation time, where the adjusted drying cycle time was 5 minutes. The cake thickness was 1.5 to 2mm, having an average of 39.14% final moisture. The following tests were done in the presence of various dosages of dodecylamine chloride (DAC) to create a slightly hydrophobic surface as the first step of two-step hydrophobization. After DAC was added, the slurry was agitated for 5 minutes with the stand-alone 3cm, three-bladed, propeller-type mixer, transferred to the pressure filter, and subjected to the same tests. During these tests, it was observed that the slurry was becoming more viscous by further addition of DAC after about 10 lb/ton. This may be attributed to the spontaneous flocculation facilitated by amine during mixing. As shown in Table 5.1, in the presence of DAC, the moisture contents were reduced to 32.14% and 31.90% at 5lb/ton and 10lb/ton reagent addition, respectively. The cake formation time, which is an indicator of dewatering kinetics, was also cut down to 4-8 minutes, even though the cake thickness was increased to 2mm to 4mm. The increased cake thickness may be an indication to an increased diameter of capillaries, which in turn, lower the capillary pressure and ease the water removal from the cake.

Table 5.1 Effect of amine (DAC) addition as the first step on dewatering of Georgia clay

Reagent Dosage (lb/t)	Moisture Content (%)
0	39.14
5	32.14
10	31.90
15	33.23
20	34.23

*The clay sample was diluted from 25% to 8%.The measured pH was 7.14. Conditioning time was 5 minutes. Applied pressure of the filter was 60 Psi. The cake thickness for baseline was 1.5- 2 mm, with reagent 2-4 mm. Cake formation time for baseline was 25 minutes, with reagent 4-8 minutes. Drying time was 5 minutes.

As mentioned above, this is a two-step hydrophobization process, and the addition of various dosages of RW was investigated as the second step on the amine-treated sample.

Table 5.2 gives the laboratory test results obtained on the clay sample using amine (DAC) and RW under the given conditions. For this series of tests amine dosage was kept constant at 10lb/ton. The mixing time for RW was 2 minutes. The addition of RW as the second hydrophobization step increased the kinetics. The cake formation time was further decreased to 3-5 minutes from 4-8 minutes depending on the RW dosage. The final cake moisture for 3 lb/ton RW addition was reduced to 28.3%, giving 27.6% overall moisture reduction. These results clearly indicate that the two-step hydrophobization process is a very effective way of lowering the final cake moisture and increasing the dewatering kinetics and thus throughput of a filter unit. Fundamentally, it may be also concluded that the reagents that are used, either alone or in combination with each other, enhance the clay dewatering by controlling contact angle, surface tension, and capillary radius. These quantities are related by the Laplace equation, and are discussed in detail in the preceding literature review.

Table 5.2 Effect of RW addition as the second step (10 lb/t DAC & various dosages of RW) on dewatering of Georgia clay

Reagent Dosage (lb/t)	Moisture Content (%)
	RW
0	31.90
3	28.33
5	28.60
10	29.30

*Slurry was diluted from 25% to 8%.The measured pH was 7.14. Conditioning time was 5minutes for amine, 2 minutes for RW. Applied pressure was 60 Psi. The cake thickness for baseline was 1.5- 2 mm, with reagent 2-4 mm. Cake formation time for baseline was 25 minutes, with reagents 3-5 minutes. Drying time was 5 minutes.

2) J.M.H Corporation's PSD Filter Vat Kaolin Clay Sample (30% solid, pH 4)

In this set of dewatering tests, East Georgia kaolin clay samples received from J.M. Huber Corporation were subjected to dewatering tests to investigate the effect of using the two-step hydrophobization method. The same test procedure was followed, except that the 2.5inch-diameter, 6inch-height, Buchner funnel vacuum filter at 25in. Hg vacuum pressure was used instead of a pressure filter. The filter cloth was supplied from the same company. The first sample tested was PSD filter Vat Kaolin Clay sample that had 30% solid content, and a pH of about 4.

Tests results obtained on the as-received clay sample show that varying amine addition did not make any significant difference in final cake moisture where the baseline moisture was approximately 40-41%. However, dewatering kinetics was observed to increase with increasing amine addition. When RW was added to the amine-treated sample as the second step of hydrophobization, the moisture reduction was not significant, i.e., 1-2 %. This might be explained by the fact that the magnitude of zeta potential has an important influence on the aggregation of particles in water. When the magnitude of zeta potential is low, the repulsive force between the particles is reduced so that the particles come closer and form agglomerates, which occur at acidic pH. Agglomeration increases the settling rate of particles, resulting in fast dewatering kinetics. On the contrary, it may induce the trapping of water in the agglomerate structures, which directly increases the moisture content of the final cake. The solid concentration of the slurry is of interest, too, because dewatering kinetics and final cake moisture are affected by the settling rate. As the suspended solid increases, a lower settling rate will occur up to a certain concentration level, which will, in turn, affect moisture content. High solid content also makes reagent conditioning difficult because of the high viscosity. This may be another reason that the as-received sample of 30% solid content did not yield desirable moisture reduction, even after two-step hydrophobization.

Taking these facts into account, another series of laboratory dewatering tests was conducted at a reduced solid content using the same sample. For these tests, the as-received sample was first diluted to 15% solid content using Blacksburg tap water prior to dewatering to exclusively inspect the effect of solid concentration on dewatering of fine clay samples. Accordingly, the pH was also increased to 4.5-5. Table 5.3 shows the results obtained to investigate the effect of two-step hydrophobization on the dewatering of the diluted clay slurry sample. For filtration tests, again, a 2.5-inch diameter, 6-inch tall Buchner funnel vacuum filter was used at 25in. Hg vacuum pressure. During the tests, amine dosage was kept constant at a pre-determined dosage i.e.,10lb/ton, while RW dosages varied from 1lb/ton to 20lb/ton. For RW addition of 10lb/ton, the results indicate that lowering the solid content decreases the final cake moisture significantly, from 40-41% to 34.58%, even when the slurry is at acidic pH.

The same tests were repeated on the diluted sample with pH further adjusted to about 7 by adding sodium carbonate (Na_2CO_3). Table 5.6 shows the effect of dilution and pH adjustment on dewatering of the clay sample. Test results show that the cake moisture was significantly

Table 5.3 Effect of Using 10 lbs/t amine and RW on the dewatering of PSD Filter Vat kaolin clay sample after dilution to 15% solids at pH 4.5-5.0

Reagent Dosage (lb/t)	Moisture Content (%) pH 4.5-5
	Amine+RW
0	40-41
5	40.61
10	42.73
15	34.58
20	35.22

Table 5.4 Effect of using 10 lbs/t amine and RW on the dewatering of PSD Filter Vat kaolin clay sample after dilution to 15% solids at pH 7.0

Reagent Dosage (lb/t)	Moisture Content (%) pH 7.0
	Amine+RW
0	40-41
5	38.85
10	32.00
15	31.85
20	28.57

reduced from 41% to 28.57% when RW was added at pH 7 and 15% solid content on amine treated sample. This is a 30% overall moisture reduction. The results also indicate that filterability is better at lower solid concentrations. When the solid content of the slurry is increased, it is more difficult to filter kaolin clay. It is also evident that pH has a considerable influence on the filterability and final cake moisture of clay.

3) J.M.H Corporation's PSD Filter pre-leached Kaolin Clay Sample (30% solid, pH 9.5)

To investigate the effect of two step-hydrophobization dewatering technique the sample was first tested as it is received i.e. 30% solid at pH 9.5. It was observed that at this higher value of pH, the particles were so well dispersed in the slurry that there was not any significant cake formation. Also, the filter cloth had an increased tendency to blind during the control tests. As a result of filter cloth blinding, no cake formation was observed. Test results obtained on this sample indicated that amine addition tended to help cake formation in the expense of increasing the final cake moisture (Table 5.5). At low amine dosage (1-5lbs/ton), the cake formation was very poor, and the cake was extremely thin, rendering the results not reproducible. In contrast, an amine addition of 10lb/ton or above provided decent cake thicknesses and produced more

Table 5.5 Effect of using amine only on the dewatering of PSD Filter pre-leached kaolin clay sample (pH 9.5& 30% solids)

Reagent Dosage (lb/t)	Moisture Content (%) pH 9.5
0	-
1	29.30
3	28.86
5	32.80
10	**37.66**
15	40.00
20	42.24

reliable moisture data. The best slurry characteristics for dewatering tests were obtained at 10 lb/t of amine dosage.

In the previous tests, promising results were obtained at pH 7 and for that reason some parallel tests were conducted on the same sample by lowering the pH to 7 using sodium carbonate (Na_2CO_3). Table 5.6 shows the test results obtained on the same sample at 30% solids and pH 7. At this pH, the particles were still so well dispersed in the slurry that there was not any significant cake formation for the control tests. Observations during dewatering tests on this sample indicate that at low amine dosage (1-5 lbs/ton), the cake formation was again poor and

Table 5.6 Effect of Using Amine Only on the Dewatering of the Coarse Pre-leach kaolin Sample from Huber (pH 7 & 30% Solids)

Reagent Dosage (lb/t)	Moisture Content (%) pH 7
0	-
1	24.79
3	30.14
5	30.67
10	**37.01**
15	41.92
20	42.98

the cake was thin. As in the previous test, the amine addition of 10 lb/ton, or above, improved cake thicknesses and produced more reliable moisture data. Further addition of amine was observed to make the slurry more viscous and, therefore, slurry conditioning with reagents became difficult. As a result, the final cake moisture actually increased. For this reason, a 10lb/ton amine dosage addition was chosen for the first hydrophobization for the tests that were done at pH 7 and 9.5.

Table 5.7 gives the overall laboratory test results using amine and RW under the given conditions, 30% solid content and pH 7 and 9.5. As mentioned above, this is a two-step hydrophobization process, where the addition of RW constitutes the second hydrophobization step.

In this series of tests, amine dosages were kept constant at 10lb/ton, while RW dosage was varied from 1 to 10 lb/t. At pH 9.5, with 10 lb/ton amine and 10 lbs/t RW the final cake moisture was significantly reduced from around 37.66% to 28.3%; at pH 7, the moisture was reduced from 37.01% to 29.11%. These results correspond to 24% and 21% overall moisture reduction, respectively. The addition of RW in conjunction with amine also increased the kinetics. A shorter cake formation time and lower final cake moisture indicated that the two-step

Table 5.7 Effect of using amine (10lb/t) and RW on the dewatering of PSD Filter pre-leached kaolin clay sample at 30% solids at pH 9.5 and pH 7.0

Reagent Dosage (lb/t)	Moisture Content (%)	
	pH 9.5	pH 7.0
0	37.66	37.01
1	36.87	36.02
3	33.60	32.02
5	33.92	31.03
10	28.31	29.11

hydrophobization technique greatly enhances the clay dewatering by controlling the contact angle, surface tension, and capillary radius.

Alum is also used in industry to coagulate the clay particles and facilitate dewatering at different pH and solid concentrations. In brief tests with alum, no significant changes were observed in final cake moisture, despite observed increases in kinetics. In some cases, moisture levels actually increased substantially.

5.2.3. Foam Aided Dewatering Method

There are no previously published studies on foam dewatering and the mechanisms which render this method effective are not completely understood at this time. However, tests conducted at Virginia tech suggest the following: first, the foam increases the void spaces (porosity) within the cake structure and, second, foam lowers the surface tension of particles such that in a clay slurry system, for example, the foam may actually increase the dewatering efficiency. This idea is in agreement with Laplace's theory, i.e., lowering surface tension, increasing capillary radius and contact angle. To investigate this new concept, some tests were conducted using foam.

a) *Experimental*

For this study, the Thiele Kaolin Company sample was used. It is a flotation product and contains approximately 25% solids. In some of the tests, the clay slurry is diluted with tap water down to 5~8% solid for dewatering testing. A set of tests was conducted at natural pH of about 7.2.

Two procedures were tested to apply foam to clay dewatering. These methods are:

i) Method 1: A foam-generating agent (Tergitol) was added to the slurry, and then agitated strongly with air blown into the slurry until it was filled with foam. The foamy slurry is then subjected to the pressure filtration tests.

ii) Method 2: The foam was generated separately by agitating the foam-generating agent containing water in presence of blowing air. Then the foam that was generated was added onto the top of clay cake inside the pressure filter chamber after the cake formation time.

In foam dewatering tests Tergitol (with different NP numbers) were used as foaming agent. One of the advantages for using Tergitol as foaming agent is that it does not contaminate the clay surface because it does not adsorb onto the clay surface.

b) Results and Discussion

The preliminary test results showed that the vacuum filtration was not effective with the foam dewatering of clay. With consideration of the natural difficulty in clay dewatering, and unsatisfactory moisture reduction obtained from vacuum filtration, all the tests were conducted using a laboratory-scale pressure filter. In each dewatering test, a desired volume of clay slurry was prepared using the methods that was described, and then transferred to the pressure filter. During the filtration, cake formation time and drying cycle time were recorded. The air pressure for filtration was set at 60 psi for all the tests. #44 filter papers manufactured by Whatman Company were chosen as filtration media. The filtrate obtained with #44 filter paper was always very clear.

<u>Foam Aided Clay Dewatering Tests at pH 7.2</u>

<u>Method 1</u>: To investigate the effect of foam on clay dewatering, clay sample was conditioned with various dosages of Tergitol NP-7 and Tergitol NP-9 (surfactants that are capable of generating foam in a solution) and aerated to create a foamy environment. In this set of test, after adjusting the slurry ph to 7.14, Tergitol, foam generating agent, was added to the slurry and agitated strongly while injecting air. 5 minutes was given for this process. The pressure for filtration was adjusted to 60 Psi. Table 5.8 shows that Tergitol NP-7 or Tergitol NP-9 alone could reduce the cake moisture down to as low as 25%. This moisture reduction correspond to approximately 35 % overall reduction. The results also showed that, Tergitol was very effective at low dosage i.e., 3 lb/t and higher dosages did not make a lot difference.

Table 5.9 Effect of foam on dewatering of Georgia clay at pH 7.0 when Tergitol NP-7 and NP-9 is used as foam generating agents.

Reagent Dosage (lb/t)	Moisture Content (%) pH 7.14	
	Tergitol (NP-7)	Tergitol (NP-9)
0	39.14	38.04
3	26.69	27.01
5	-	26.52
10	26.35	26.11
30	25.52	25.33

Slurry was diluted from 25% to 8%.The measured pH was 7.14. Conditioning time was 5minutes. Applied pressure was 60 Psi. The cake thickness for baseline was 1.5- 2 mm, with reagent 2-4 mm. Cake formation time for baseline was 25 minutes, with reagent 20-25 minutes. Drying time was 5 minutes

The cake formation time at pH 7.14 was 25 minutes for baseline test and 20 minutes in the presence of different types of Tergitol. The drying time was 5 minutes. During the tests, it was observed that, when Tergitol added into the slurry along with air and agitated, the cake thickness increased from 1- 1.5 mm to 2-4 mm.

Another set of tests were conducted to investigate the effect of foam on a thicker cake. The tests discussed in the above section were conducted at 2~4 mm thick cake by using 50 ml slurry. Table 5.9 shows the test results obtained with approximately 5-6 mm thick cake when 100 ml slurry was used for filtration tests in the presence of Tergitol NP-7. As expected, the increased cake thickness gave higher cake moisture, but the moisture reduction from 43% to 29% is still impressive, giving 33% overall moisture reduction.

Table 5.8 Effect of Tergitol with thick cake (5-6mm) on dewatering of Georgia clay

Reagent Dosage (lb/t)	Moisture (%)
0	43.45
1	29.19
5	29.94
10	30.23

Method 2: Another way of using foam is adding foam that is generated separately on the top of the cake. Once the cake is formed, it is possible to add foam onto the top of the cake. The test results showed that creating foam inside the slurry or adding foam on the top of the cake for the dewatering tests gave approximately same moisture reductions. When Tergitol added on top of the cake, the moisture again drops from baseline moisture of 38-39 % down to 25-26 %. This moisture reduction can be attributed to the decrease in the in the surface tension of the liquid that has to be dewatered. As seen in Figure 5.1 Tergitol is very effective in reducing the surface tension even at very low dosages.

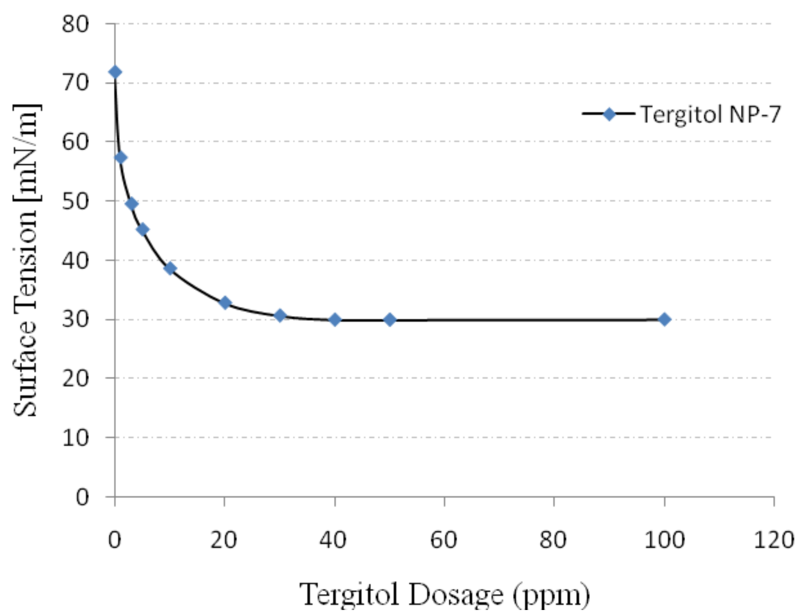


Figure 5.1 Tergitol NP-7

As explained in the Laplace equation, the cake moisture can be lowered by lowering the surface tension, increasing capillary radius and contact angle. Tergitol is probably controlling the surface tension. Addition to this, applying Tergitol in foam method enlarges the capillary radius, thus lowering the moisture. This phenomenon may also explain the moisture values level out at very low dosages. As presented in Table 5.8 and Table 5.9 the moisture reduction is almost same after the first dosages. Increasing the dosage does not change the final cake moisture.

It should also be noted that, when Tergitol is used as a foaming agent in water, dosage added on top of the pre-formed cake cannot determined. Both of the techniques of introducing foam gave significantly low moistures. However, one undesirable point about the method 2 is, when the foam is added on top, an extra time is given for the foam to travel within the cake. This extra time increased the total filtration time.

Clay Dewatering Tests at Acidic pH (pH 3-3.5)

As introduced above, clay is usually further treated by chemical bleaching after separation in order to achieve the required brightness. In industry, the leaching process takes place in the presence of sodium hydrosulfite at an acidic pH (pH 3) to keep the dissolved iron in a soluble ferrous state and prevent the formation of ferric hydroxide.

In this series of test, the effect of Tergitol as foaming agent was investigated after the clay sample was bleached in the similar manner as in industry. The pH of slurry was adjusted to 3 using sulfuric acid, and then sodium hydrosulfite was used as a reducing agent. Alum is also added to the clay slurry to help coagulation and increase the dewatering kinetics. Table 5.10 shows the dewatering results at pH 3.

Table 5.10 Effect of foam on dewatering of Georgia clay at pH 3 when Tergitol NP-7 is used as foam generating agent

Reagent Dosage (lb/t)	Moisture Content (%) pH 3
	Tergitol (NP-7)
0	38.04
3	32.87
5	32.63
10	32.14
30	30.84

*Solid Content was 5-6 %. Tergitol was added to the slurry (mixed and aerated to produce foam). Applied pressure 60 Psi., Cake thickness was 1.5-2 mm. with 50 ml slurry.

The cake moisture was increased after the clay sample was bleached at acidic pH. However, the filtration kinetics was improved at acidic pH because of the coagulation effect. The cake formation time was lowered down to 5 minutes when the medium is acidic. The cake moisture can still be lowered to 30% when foam was generated in the slurry using Tergitol as foaming agent.

A series of tests were conducted to improve dewatering kinetics further by using Alum as coagulating agent. Alum addition at 5 lb/ton increased the kinetics dramatically (Table 5.11); the

Table 5.11 Effect of alum addition on dewatering of Georgia clay when Tergitol was used as foaming agent

Reagent Dosage (lb/t)	Moisture Content (%) pH 3
	Tergitol (NP-7) + 5lb/t Alum
0	41.10
1	37.01
3	35.48
5	35.26

cake formation time drops down to 50 seconds from 5 minutes, but it also caused slight increase in cake moisture.

All the cakes were also visually inspected and it was observed that when the foam was generated in the clay slurry and dewatered, the bubbles left numerous small pockets creating a flaky structure which is an indication of a drastic increase in porosity.

5.3. FLY ASH

5.3.1. Background Information

When coal is burned in a power generation facility or in an industrial boiler, it produces noncombustible mineral matter called ash. This residue is partitioned into bottom ash (or slag), and fly ash. Bottom ash is coarse in nature and falls to the bottom of the combustion chamber, where it is easy to collect during routine cleaning. The properties of bottom ash make it a good road base and construction material, so it can be readily given away or sold. On the other hand, fly ash is lighter and very fine in nature, and remains suspended in the flue gas which is not so easily disposed of. Most fly ash is captured by particulate emission control devices, such as electrostatic precipitators or filter fabric collectors – commonly referred to as baghouses – before release to the atmosphere. There are also two other byproducts of coal combustion air control technologies, namely flue-gas desulfurization (FGD) and fluidized-bed combustion (FBC) wastes. Collectively, all the residue materials are named as coal combustion products (CCP).[11, 12, 13]

In 2004, approximately 64 million metric tons of fly ash was produced, 25.5 million metric tons of which were used in a variety of engineering applications. Approximately 61% of the useable fly ash finds its application as an addition to cement in concrete production. Depending on the intended final use, quality requirements for fly ash vary. For concrete applications, there are four important characteristics: loss of ignition (LOI), fineness, chemical composition and uniformity. LOI is especially critical for concrete application. It is a measurement of unburned coal (carbon) that remains in the fly ash. Basically, carbon content in fly ash may result in significant air-entrainment problems in fresh concrete, and it may also unfavorably affect the durability of concrete.[12, 13, 14] At high LOI, carbon can be either reclaimed or discarded utilizing flotation which is widely demonstrated. Clean fly ash that is hydrophilic in nature is dewatered to generate a dry product for end use.[12, 14-16]

5.3.2. Two Step Hydrophobization Technique

a) Experimental

Several fly ash samples with various carbon contents were provided for flotation and dewatering tests. The samples were received from stockpiles at two power plants, one located in Kentucky and the other in Mexico.

i) Kentucky Fly Ash Samples

- 65 % Ash-35 % Carbon (LOI); (91 % -325 mesh material)
- 100% Fly Ash; (100% -325 material)

Only laboratory dewatering tests were conducted on the above samples, utilizing the two-step hydrophobization technique previously described.

ii) Mexico Fly Ash Sample

The carbon content (LOI) for this sample ranged from 2.59% to 5.85% with an average of 4.03% at 10-12 % moisture. The amount of the -100 mesh material was approximately 76%, the +1 mm was 8%, and the -1mm x 100 mesh was 16%. On this sample a number of flotation and dewatering tests were conducted to produce clean and dry fly ash product. The tests included both laboratory and pilot scale tests, described below.

A number of laboratory flotation tests were conducted to lower the LOI using a procedure that was suggested by the company which supplied the fly ash samples. The samples were prepared at 25 % solids, and conditioned with the collector (2.67 lb/t) for 8 minutes at 2000 rpm in Denver cell. This step is designed as a conditioning step. After conditioning, a known amount of water was added to lower the slurry solid content down to 20%. After frother addition at about 0.18 lb/t, the flotation was initiated at 950 rpm. After 12 minutes of flotation and sample collection, another 0.67 lb/t of collector was added as the second step and conditioned for 4 more minutes. Right after the mixing the flotation was continued for another 4 minutes and bulk samples were collected. Subsequent to sample generation, dewatering tests were conducted using various types and dosages of chemicals.

Pilot scale tests were conducted on the same sample and tests were run to produce clean fly ash samples for dewatering tests. The general layout of the pilot scale circuitry includes

- A 12” wide fine sieve to classify the raw feed at nominally 100 mesh. The screen feed was delivered by a pump and buckets from the barrels
- A conical bottom slurry pump, approximately 250 gallons capacity, received the underflow from the sieve screen. A mixer was used in the sump to aid in keeping the solids in suspension
- A centrifugal pump to circulate the slurry from the sump to the upper floor level and back into the sump using 3” diameter PVC piping.
- A bank of 4, 3 cu ft conventional flow-through type flotation cells utilized as primary cells. Feed to these cells was supplied by a slip stream taken from the 3” diameter pump discharge pipe.
- A bank of 4, 3 cu ft conventional cell to cell type flotation cells utilizing the first cell as conditioner (air valve closed) and the remaining cells as secondary cells.
- Barrels were used to collect the floated product and tails product. The tails product was flocculated in barrels and the water decanted to thicken the product.
- A 2 sq ft, single disc vacuum filter to dewater the thickened product.

Following the screening process, the slurry was circulated/blended in the sump using the pump and the mixer for several minutes. The collector was added to the slurry and was conditioned for 20 minutes (using the pump and the mixer). After conditioning period, water was added to the sump to dilute the slurry to 20% solids (by weight). The slurry was again circulated/blended in the sump for several minutes.

The supplier company of the fly ash sample established 16 minutes of retention time in the laboratory. It was suggested that the scale up to the pilot-scale cell would require 2-2.5 times the lab cell retention time. Considering the total capacity of the seven pilot scale cells is 20.1 cu ft (active) or about 150 gallons the slurry feed rate was set to 4 gpm to accommodate a 32-40 minutes retention time.

5.3.3. Results and Discussion

a) Kentucky Fly Ash Samples

65 % Ash-35 % Carbon Sample

Typically fly ash contains low amounts of carbon. However, the sample received from the Kentucky location contained 35% carbon matter. Considering this, a limited number of dewatering tests were conducted directly using dewatering aids such as RW, RU and RV, without first using dosing amine. However, the cake moisture did not change significantly with

Table 5.12 Effect of reagent addition at (1:2) ratio on dewatering of Kentucky coal sample at 17-19 inches Hg vacuum

Reagent Dosage (lb/ton)	Moisture (%)		
	RW	RU	RV
0	41.81	41.81	41.81
1	41.45	41.97	41.41
2	41.48	41.77	41.44
3	39.99	41.30	40.88

*dissolved in diesel at 1:2 ratio

addition of reagents. Table 5.12 shows the dewatering test results with the addition of various dewatering aids. RW addition lowered the moisture by only 2%. For filtration tests, a 2.5-inch diameter, 6-inch tall Buchner funnel vacuum filter was used at 25 in Hg vacuum pressure

This result was not surprising because, despite the presence of carbon, this sample was primarily comprised of fly ash. Since ash is hydrophilic, it has no affinity for the dewatering reagents tested. (RW, RU and RV were developed for coal dewatering and are low-HLB surfactants, designed to adsorb on hydrophobic surfaces, like coal particles.) From these results, it was clear that the first step of the two-step hydrophobization process would be necessary to dewater this sample. Amine is a cationic surfactant, and is known to interact with hydrophilic particles, such as clay.

The second set of tests was conducted using amine. Table 5.13 shows the effect of amine addition as the first step of the two-step hydrophobization technique. The cake moisture was reduced from 41.81% to 29% at the addition of 20 lbs/t amine, representing a 30% moisture reduction. Even though moisture reduction was considerably high, the viscosity was increased when the amine dosage was more than 10 lb/t and caused handleability problems.

Table 5.13 Effect of amine addition at (1:2) ratio on dewatering of Kentucky coal sample at 17-19 inches Hg vacuum

Reagent Dosage (lb/t)	Moisture Content (%)
0	41.81
0.5	41.7
1	38.3
2	41.0
3	38.1
5	35.4
10	33.4
20	29.0
30	28.9

As mentioned above, this is a two-step hydrophobization process and addition of various dosages of RW was investigated as the second step on the amine treated sample. Table 5.14 gives the laboratory test results obtained on this sample using amine and RW. For this series of tests, amine dosage was kept constant at 10lb/t. The mixing time for RW was 2 minutes. The addition of RW as second hydrophobization step increased the kinetics, i.e., the cake formation time was decreased almost by half depending on the RW dosage. The final cake moisture for 3lb/t RW addition was further reduced to 28.5%, indicating that the two-step hydrophobization process is superior to the one-step process for both moisture reduction and filtration kinetics.

Table 5.14 Effect of amine (10 lb/t) and RW addition at (1:2) ratio on dewatering of Kentucky coal sample at 17-19 inHg vacuum

Reagent Dosage (lb/t)	Moisture Content (%) Amine+ RW
0	33.4
1	33.0
2	30.5
3	28.5
5	28.0

100% Fly Ash Sample

Another set of dewatering tests was conducted on the 100 % fly ash sample. During the tests, it was observed that the viscosity of the fly ash slurry was gradually increased with respect to increased amine dosage. When amine dosages of 5lb/t or greater of were used, the viscosity was increased to the point that the conditioning of the fly ash slurry became almost impossible. Also, very high viscosity yielded poor dewatering performance. With respect to handleability, kinetics and viscosity, the best performance was obtained with 3lb/t of amine addition; however,

Table 5.15 Effect of amine addition as the first step on dewatering of fly ash sample at 20 in Hg Vacuum pressure

Reagent Dosage (lb/t)	Moisture Content (%)
0	22.6
1	22.13
3	22.44
5	19.41

varying amine dosage did not significantly change the moisture reduction. Table 5.15 shows the results obtained using different amounts of amine as the first step of hydrophobization.

Subsequent to treating the sample with amine, addition of various dosages of RW, RU and RV was investigated as the second step. In this series of tests, amine dosage was kept constant at 3lb/ton. As seen in Table 5.16 the addition of the dewatering aids lowered the moisture content from 22% down to approximately 17%, giving 22% overall moisture reduction. RV gave the least moisture reduction, but the kinetics was considerably fast.

Another set of tests were conducted using a 2.5 inch pressure filter. The pressure was adjusted to 30 psi and the rest of the test parameters were kept similar to the vacuum filtration tests. As expected, there was not any significant moisture reduction with the amine addition

Table 5.16 Effect RW, RU and RV addition at (1:2) ratio on dewatering of amine pre-treated (3 lb/t of amine) Kentucky fly ash sample at 20 in Hg vacuum

Reagent Dosage (lb/ton)	Moisture (%)		
	Reagent RW	Reagent RU	Reagent RV
0	22.44	22.44	22.44
1	22.63	-	23.38
3	18.41	19.17	21.51
5	17.07	17.26	20.60

*dissolved in diesel at 1:2 ratio

Table 5.17 Effect of amine addition as the first step on dewatering of fly ash sample at 30psi pressure

Reagent Dosage (lb/t)	Moisture Content (%)
0	13.11
1	12.19
3	12.50
5	12.16

(Table 5.17), but increased kinetics was observed. The amine dosage was varied from 0 to 5lb/t and best kinetics and viscosity were observed at 3lb/t of dosage. Baseline moisture was around 13.11% which was considerably lower compared to vacuum filtration moisture values. This is directly related to the difference in the pressures applied.

As the second step, dewatering aids, RW, RU and RV were added at various dosages from 1 to 5lb/t (Table 5.18). The best moisture reduction was achieved using RW which resulted in a total moisture reduction of 22% moisture reduction followed by RU with a 16.6% reduction; RV was the least effective in moisture reduction.

Table 5.18 Effect of amine (3 lb/t) and RW addition at (1:2) ratio on dewatering of Kentucky coal sample at 30psi pressure

Reagent Dosage (lb/ton)	Moisture (%)		
	Reagent RW	Reagent RU	Reagent RV
0	12.50	12.50	12.50
1	12.52	11.92	11.72
3	11.07	11.13	11.88
5	9.86	10.48	11.74

*dissolved in diesel at 1:2 ratio

b) Mexico Fly Ash Samples

The sample from Mexico contained average carbon content (LOI) of 4.03%. To lower the carbon content of the fly ash and produce bulk samples for dewatering tests, a number of laboratory scale flotation tests were conducted. RV was used as collector and Nalco DVS4U013 was used as frother and added as previously described. In this way, the carbon content was lowered from to an average of 0.74 %. The RV dosage was 2.67 lb/t for the first and 0.67 lb/t for the second step of the flotation. After enough samples were produced, dewatering tests were conducted. Amine was used as the first hydrophobizing step at a dosage of 1lb/t. The baseline moisture values did not change; however, the kinetics was increased in the presence of the amine. As seen in Table 5.19, the moisture was decreased from 18.2% to 13.2 %, corresponding to a 27 % overall moisture reduction.

Table 5.19 Effect of amine (1lb/t) and RV addition as the first step on dewatering of fly ash sample at 20 in Hg vacuum pressure

Reagent Dosage (lb/t)	Moisture Content (%)
0	18.20
1	16.10
3	14.70
5	13.20

Similar flotation and dewatering tests were conducted using the pilot scale units. The flotation tests were conducted using collector dosages of 2lb/t and 0.5lb/t, and frother (DVS4U013) dosages of 0.75lb/t and 0.25lb/t in the first and second steps of flotation, respectively. Flotation yielded tailings of 92.7% and the LOI value was 0.68%. After producing a sufficient amount fly ash sample, a number of dewatering tests were conducted. The first set of

Table 5.20 Effect of using amine (1lb/t) and RV on the dewatering of the fly ash sample at 30% solids

Reagent Dosage (lb/t)	Moisture Content (%)	
	Test 1	Test 2
0	17.90	18.50
3	15.20	15.60
3	15.60	15.40

tests was conducted using 1lb/t of DAC and 3lb/t of RV. The amine was mixed for approximately 4 to 5 minutes, and RV was mixed for 10 minutes at 25% solids. The slurry was then diluted to 20% solids and subjected to dewatering tests. I n this set of tests, unlike the laboratory dewatering tests, an adequate particle pick up was not possible for a decent cake formation.

To be able to increase the cake formation and thickness, after treating the sample with amine and RV, two sets of tests were conducted in the presence of flocculant (Test 1) and a combination of flocculant and coagulant (Test 2). The results are presented in Table 5.20. The filter cake without the dewatering aid had a moisture content of 17.90% when only flocculant was used. When treated with the dewatering aid at 3lb/t, the moisture was decreased to 15.20%. In the second test, wherein a flocculant and a coagulant were used along with the dewatering aid, a moisture content of 15.40% was obtained. As shown, the addition of coagulant increased the baseline moisture to 18.50%; however, the filtration kinetics and cake thickness were also increased. The pilot scale results clearly showed that it is possible to lower the final cake moisture by approximately 15%.

5.3.4. Foam aided dewatering method

a) Experimental

As previously mentioned foam can be applied in two ways, by adding the foam-generating agent into the slurry or generating externally and adding on top of the cake. In this set of test the method 1 employed. The tests were conducted on samples that were obtained from a stockpile in Kentucky.

b) Results and Discussion

The first set of tests was conducted on the sample which contains 65 % fly ash-35% carbon matter. These tests were carried out using PPG at various dosages. The foam is generated in situ and dewatering tests were conducted 20 in Hg vacuum pressure. The results are presented in Table 5.21. The results showed that in the presence of foam the cake moisture was dropped down to 32.04 % from 42.03% of baseline moisture %. This decrease corresponds to 24% overall moisture reduction.

It was also observed that at 50 g/t of PPG dosage, the moisture was leveled out and increasing PPG dosage did not have any impact in final cake moisture. This can be explained by the reduction in surface tension (Figure 5.2) and drastically increased porosity in the cake structure.

Table 5.21 Effect of PPG addition on dewatering of Kentucky fly ash sample at 17-19 in Hg vacuum

Reagent Dosage (lb/t)	Moisture Content (%)
	PPG-400
0	42.03
50	33.06
200	32.66
150	32.04
200	32.22

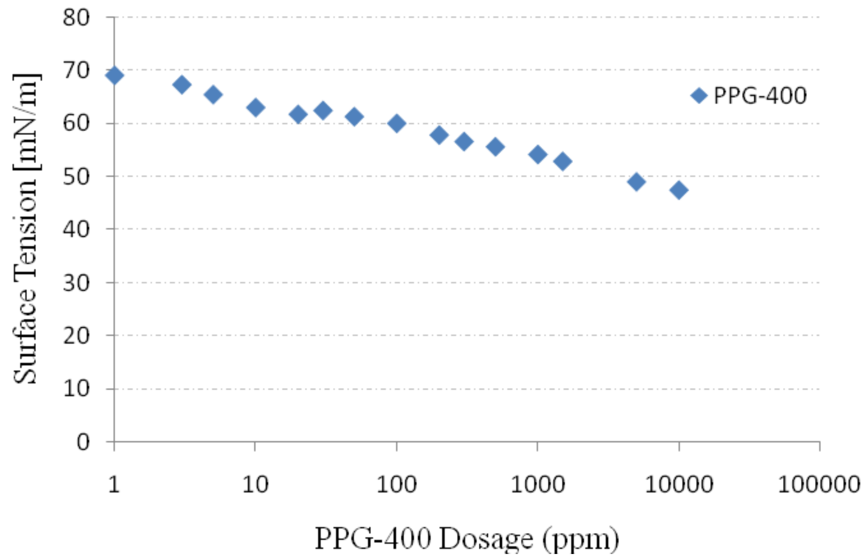


Figure 5.2 PPG-400

Another set of foam aided dewatering tests conducted on 100% fly ash sample obtained from Kentucky. As observed in the previous tests, the moisture was again lowered to its final level at very low dosages The results showed that the Tergitol addition at 1 lb/t lowered the moisture down to 16.42 % from a baseline of 22.44 %, which corresponds to 27 % overall moisture reduction. The results presented in Table 5.22.

Table 5.22 Effect of Tergitol addition on dewatering of Kentucky coal sample at 17-19 in Hg vacuum

Reagent Dosage (lb/t)	Moisture Content (%) Tergitol NP-7
0	22.44
1	16.42
3	16.56
5	16.97

5.4. COPPER

5.4.1. Background Information

Copper use has evolved over millenniums, often with the aspects of civilization that define the times. Used in ancient weapons, as currency, in ornamental fixtures of the Renaissance, and now as an essential material in building infrastructure and electronics, the metal has been an important commodity since its discovery. In addition to being malleable and ductile, copper forms favorable alloys, because it is corrosion resistant, biostatic and easily cast, and is an exceptional electrical and thermal conductor. Today, copper is nearly exclusively exploited for its conductivity and resistance to corrosion, and the rapid progression of construction and technology maintains a high demand for the metal. The copper worldwide usages in various sectors are: building construction (37 %), electrical components (27%), industrial machinery (15%), transportation (11%), and consumer products (11 %). [17-19]

Although there are variations on processing methods for copper (e.g. bioleaching, flash smelting), the most widely used, by far, are the conventional concentrate-smelting-refining (pyrometallurgy) and leaching-solvent extraction-electrowinning (hydrometallurgy) methods. Conventional concentrating includes crushing, grinding, screening, flotation, thickening and dewatering. [17, 20]

5.4.2. Experimental

Copper mineral samples from Sweden were used for dewatering tests in the presence and absence of dewatering aids. The samples were flotation products and received from preparation plants in dry form. All tests were conducted at 2 minutes of total dewatering time. This time included the cake formation and dry cycle times.

5.4.3. Results and Discussion

It was determined in previous studies that RW is an effective dewatering aid for particles with hydrophobic surfaces. To investigate its suitability for use on flotation products from copper processing, a series of dewatering tests were performed on the clean copper samples. These tests included both pressure and vacuum filtration methods. Table 5.23 shows the copper dewatering results obtained with RW at varying dosages for tests using vacuum filtration. At 1 lb/ton RW, cake moisture was reduced from 14.98% (baseline) to 12.61% for medium cake thickness (8mm). At 3lb/ton RW, the moisture was further reduced to 8.08%. When a thicker cake (14 mm) was used, the baseline moisture was 17.01% and an addition of 3 lb/t of RW reduced the moisture to 12.39 %.

Table 5.23 Effect of using RW on the dewatering of the copper sample at 20 in Hg vacuum pressure

Reagent Dosage (lb/t)	Moisture Content (%)	
	8 mm	14 mm
0	14.98	17.01
1	12.61	15.91
3	8.08	12.39

A second set of tests was conducted using a pressure filter and the results obtained are presented in Table 5.24. The baseline moisture levels were much lower in these tests. For example, with an RW dosage of 3lb/t and a cake thickness of 8 mm, the moisture was reduced from 8.48 % to 5.07%. At the same RW dosage and a cake thickness of 14mm cake, the moisture was reduced from 9.66 % to 6.38 %.

Table 5.24 Effect of using RW on the dewatering of the copper sample at 500 Kpa pressure

Reagent Dosage (lb/t)	Moisture Content (%)	
	8 mm	14 mm
0	8.48	9.66
1	7.11	8.12
3	5.07	6.38

5.5. SUMMARY AND CONCLUSIONS

Dewatering tests were conducted to effectively reduce moisture content in kaolin clay, copper and fly ash samples. The experimental work included a variety of parameters (e.g., vacuum and air pressure, cake thickness, pH, and LOI) and two new dewatering techniques were utilized: a two-step hydrophobization method using novel dewatering aids, and a foam-aided dewatering method.

Kaolin clay is one of the most difficult minerals to dewater due to hydrophilic nature and ultrafine particle size. It has been shown that kaolin clay samples can be dewatered using both of the new methods. Two-step hydrophobization was tested on samples with various pH values. During pressure filtration (60psi) tests on Thiele's clay samples (pH 7), the moisture was lowered from 39.14% to 31.90% and 28.33% when using amine and Reagent W, respectively. This corresponds to approximately 28% overall moisture reduction. Using amine and RW together, the kinetics were also increased such that the cake formation time was lowered from 25 minutes to 3-5 minutes, for a cake thickness of 2-4 mm at 60psi. The results showed that the slurry pH and the solid content are two very important parameters when using two step hydrophobization methods.

It was confirmed that the two-step hydrophobization method can also be very effective in vacuum filtration when the slurry is diluted. For PSD Filter Vat kaolin clay sample diluted to 15% solids, it is possible to lower the final cake moisture by 16% at pH 4.5 and 31% at pH 7.

Foam-aided dewatering may be applied using two different methods. The foam can either be generated in the slurry or added on top of the filter cake separately. Tergitol was used as the foam generating agent. The results confirmed that, when the final cake moistures are considered, both methods of foam application gave the same moisture reductions. However, considering the filtration time constraints, it was found to be more favorable to generate foam in the slurry. The foam-aided dewatering test results also showed that very low dosages of Tergitol addition lowered the final cake moisture by 19% at pH 3 and 35% at pH 7. Also, Alum can be used to increase the filtration kinetics, even though it may cause a slight increase in final cake moisture. For example, in one test, cake formation time was lowered from 5 minutes to 50 seconds.

In addition to utilizing the above methods in clay dewatering, similar tests were conducted on various fly ash samples. These test results have shown that depending on the fly ash properties, the two-step hydrophobization method is significantly effective in reducing the final cake moisture and increasing the dewatering kinetics. For samples obtained from Kentucky, it was possible to lower the overall cake moisture by 24-32 % in vacuum filtration and 11-24 % in pressure filtration.

Dewatering tests utilizing two-step hydrophobization conducted on another fly ash sample from Mexico confirmed similar results. This sample was treated with 1 lb/t of amine and 3lb/t of RV. The tests were conducted using a 2.5-inch diameter, 6-inch tall Buchner funnel vacuum filter, operated at vacuum pressure of 25 inches of Hg. The moisture was decreased from 18.2% to 13.2%, corresponding to a 27% overall moisture reduction. A brief study was also conducted using a pilot-scale vacuum disc filter. The results showed that a 15-17% moisture reduction was obtained when the two-step hydrophobization method was used in the presence of dewatering aids.

The foam-aided dewatering method was tested on various fly ash samples. PPG-400 and Tergitol were used to produce foam. At low dosages of these foaming agents, the final cake moisture was lowered by 23-27% when the foam was generated in-situ. This confirms that high moisture reductions are obtainable with relatively low dosages of foaming agents.

Novel dewatering aids were also tested on (hydrophobic) copper samples in both vacuum and pressure filtration tests. The vacuum filtration tests results showed that 45% and 30% overall moisture reduction can be obtained at 8mm and 14mm cake thicknesses, respectively. The pressure filtration tests showed that the moisture can be lowered to 5% and 6.3% at 8mm and 14 mm cake thicknesses, respectively.

Overall, both the two-step hydrophobization and foam-aided dewatering methods are capable of reducing the moisture of several mineral types to levels which cannot be achieved with current mechanical techniques. As well, the experimental results showed that the kinetics are also increased drastically when these methods are employed, which in turn increases the throughput of a filter unit.

5.6. REFERENCES

1. Elzea Kegel, Industrial Minerals and Rocks, Commodities, Markets and Uses, SME 2006, 7^{th} edition
2. USGS-Mineral Commodity Summaries – Clays 2008
   Available from: http://minerals.usgs.gov/minerals/pubs/commodity/clays/mcs-2008-clays.pdf)
3. Michael L. Zientek and Greta J. Orris, USGS-Geology and Nonfuel Mineral Deposits of the United States,Open-File Report 2005-1294A
   Available from: http://pubs.usgs.gov/of/2005/1294/a/of2005-1294a.pdf)
4. Yildirim, I., "Surface Free Energy Characterization of Powders", Ph.D. dissertation, Virginia Tech, April 2001.
5. Roe-Hoan Yoon and Joseph Shi, Processing of Kaolin Clay, 1986
6. Professor Haydn Murray, Industrial Clays Case Study, March 2002 No64 MMSD, Indiana University
7. Robert L. Virta, Clay and Shale, USGS-2006 Minerals Yearbook, 2008
8. Rushton, A., Ward, A.S. and Holdich, R.G., Solid-Liquid Filtration and Separation Technology",VCH, New York, 2000.
9. San, O. and Hosten, C., Filtration Behavior of Flocculated Kaolin Slurries", Proceedings of the 6*th* International Mineral Processing Symposium, Turkey, 655-660, 1996.
10. Basilio, C.I., Recent Advances in the Processing of Kaolin", Thiele Kaolin Company,
11. American Coal Council & American Coal Ash Association, Coal Ash Fact Sheet 2005, Available from: www.acaa-usa.org
12. American Coal Ash Association, Fly Ash Facts for highway Engineers, Technical report (FHWA-IF-03-019), 2003
13. W. Lee Daniels, Barry Stewart, Kathryn Haering, and Carl Zipper, The Potential for Beneficial Reuse of Coal Fly Ash in Southwest Virginia Mining Environments, Virginia Cooperative Extension Publication (460-134) 2002
14. Kelly, T.D., and Matos, G.R., Sullivan D.E., Historical Statistics for Mineral and Material Commodities in the United States: U.S. Geological Survey Data Series 140, Available from: http://pubs.usgs.gov/ds/2005/140/.

15. M.Niewiadomski, J.Hupka, J.D.Miller, Flotation of Fine Coal Particles From Fly Ash , Fizyochemiczne Problemy Mineralurgii, 31 (1997), 221-228
16. Paul H. Zandhuis, Michael V. Ciocco, Richard P. Killmeyer, McMahan L. Gray, Yee Soong, Technique for Determining the Potential Recovery of Carbon from Fly ash, NETL 2001 Conference on Unburned carbon (UBC) on Utility Fly Ash
17. Copper Development Association, Inc., *Copper Supply and Consumption*, *Annual Data 2004*
18. Lugaski, Thomas P., *Copper: The Red Metal*, University of Nevada, Reno, NV, 1997.
19. Phelps Dodge Corporation, *2004 Annual Report*, Phoenix, AZ, 2004.
20. U.S. Geological Survey, *Mineral Commodity Summaries*, USGS, Reston, VA, January 2005.

CHAPTER 6 CONCLUSIONS

In the preceding chapters, the results of the dewatering test work have been discussed to determine the effectiveness of using novel dewatering aids for reducing the moisture of the fine coal and other mineral products. Overall, the aids can provide an economically feasible means of dewatering fine particles, often producing more desirable results than can be achieved by mechanical means alone. Additionally, a completely new dewatering method – foam-aided dewatering – was developed and tested, also with favorable results. This chapter summarizes the general conclusions from these investigations.

Laboratory- and pilot-scale dewatering tests were conducted on various fine coal samples to do engineering evaluations of the novel dewatering aids. The aids were capable of increasing the contact angle and lowering the surface tension of the water, while increasing the capillary radius, all of which are responsible lowering the final cake moistures. The evaluation tests were conducted using different types and dosages of the novel dewatering aids while varying operating parameters. A Buchner Funnel vacuum filter and an air pressure filters were used for laboratory tests while a vacuum disc filter (VDF) and a horizontal belt filter (HBF) were utilized for pilot-scale tests. Laboratory test results showed that the use of dewatering aids can substantially decrease the final cake moisture by as much as 50%, while increasing the dewatering kinetics by 30-50%. Due to the improvements in moisture reduction and increased kinetics, it was estimated that the Mingo Logan Preparation Plant can realize additional annual net revenue of $1.57 million with a payback time of 3.1 years.

Other results also indicated that adequate conditioning time and intensity are critical to obtain desired moisture reductions. Thus, mixing should be optimized. In one test, moisture reduction was around 36% after proper conditioning as compared to 19% when the conditioning was not sufficient. For this reason, for both laboratory- and pilot-scale test work, conditioning tanks were used.

A number of tests were conducted using the dewatering aids as collectors where conditioning was not possible. When the reagent dosage was only 50% of a normal diesel dosage, it was observed that coal recovery was increased and the moisture was lowered by 7%. In the pilot-scale tests, when the reagent dosage was the same as diesel, the moisture reduction was further reduced, giving a 14% overall reduction in addition to increased kinetics.

At the Buchanan coal preparation plant, VA, the use of novel dewatering aids improved both the moisture reduction and throughput, although the moisture reduction was less than anticipated. This was primarily due to water chemistry and possibly due to insufficient conditioning, which may have negatively affected the performance. The plant water had large amounts of Ca^{2+} ions, and conditioner was not equipped with a properly-sized impeller.

An empirical scale-up model was developed to predict final moistures under a given set of operating conditions i.e. the reagent type and dosage, filtration time, vacuum pressure, and cake weight. The model was for the scale-up of vacuum disc filters and horizontal belt filters. The model shows that dewatering aids can be most effective when used in conjunction with a HBF which allows control of cake thickness and drying cycle time independently.

The benefits of using the novel dewatering aids have been demonstrated in full-scale at the Smith Branch impoundment site. In the full-scale operation, the total moisture content was reduced from 26% to 20% at a reagent dosage of approximately 0.5 lb/ton, and further to 17.5% at approximately 1 lb/ton; this represents a total moisture reduction of nearly 33%. The use of the dewatering aids also reduced the power consumption by 30% for the vacuum disc filter. Furthermore, the use of the novel dewatering aids improved handlability of the coal.

Two new dewatering techniques were tested for minerals that are hydrophilic in nature: a two-step hydrophobization method using novel dewatering aids, and a foam-aided dewatering method. The results showed that the use of these new dewatering methods can effectively reduce the moisture contents in kaolin clay and fly ash samples

The two-step hydrophobization process was tested on kaolin clay samples at different pH and solids contents. A pressure filtration test conducted at 60 psi showed that substantial moisture reductions could be obtained. Using the addition of amine at the first hydrophobization step and a novel dewatering aid at the second hydrophobization step, the moisture was reduced and the kinetics were increased. It was confirmed that the two-step hydrophobization method can also be very effective in vacuum filtration when the clay slurry is diluted.

The foam-aided dewatering method can be implemented in two different ways. The foam can either be generated in the slurry or added on top of the filter cake separately. Even though both of the methods are very effective in reducing the cake moisture, when considering filtration time constraints, it was found that generating the foam in the slurry is more favorable. Test results showed that significant moisture reductions could be obtained in a wide range of slurry pH values.

Similar dewatering tests were also conducted on different fly ash samples. These test results have shown that depending on the fly ash properties, the two-step hydrophobization method is effective in reducing the final cake moisture and increasing the dewatering kinetics.

Additionally, the foam-aided dewatering method was tested on various fly ash samples. It was confirmed that high moisture reductions are obtainable with relatively low dosages of foaming agents.

Novel dewatering aids were also tested on copper samples in both vacuum and pressure filtration tests. The vacuum filtration test results showed that 30-45% overall moisture reduction can be obtained depending on the cake thicknesses. The pressure filtration tests showed that, in the presence of dewatering aids, the moisture can be lowered to 5% and 6.3% at 8 mm and 14 mm cake thicknesses, respectively.

Overall, test results clearly showed that when operating parameters are optimized, use of novel dewatering aids can generate substantial moisture reductions in mechanical dewatering. Additionally, the dewatering kinetics can be increased, which in turn should increase the throughput of filter operations. The dewatering aids can also be used as collectors where installation of a conditioner is not possible. They not only produce similar or better results than conventional collectors, but also lower moisture contents. Both the two-step hydrophobization and foam-aided dewatering methods are capable of reducing the moistures of ultrafine mineral particles that are difficult to be dewatered using the currently available mechanical dewatering devices.

FUTURE WORK

In view of the results and conclusions of the experimental work included in this dissertation, some recommendations regarding potential future work are provided below:

- Novel dewatering aids were successfully tested on fine coal and significant moisture reductions were obtained. The dewatering aids effectively increased the hydrophobicity of the coal surface. The same approach could also be applied to coarse coal, i.e., 2 inches x 2 mm size. Typically, after density separation, the coarse coal is spray-washed and fed to dewatering screens to remove excess water. At this stage the novel dewatering aids might be sprayed on the coal travelling on the screen. To increase the effectiveness of the dewatering aids for coarse coal application, an appropriate surface tension reducing agent might also be blended with dewatering aid.

- The novel dewatering aids are water insoluble and needs a certain amount of conditioning. Test results confirmed that conditioning intensity and time are two important parameters for dewatering aid dispersion and adsorption. In laboratory- and pilot-scale tests, the proper conditioning was supplied by in-house build mixing tanks. However, conditioning in this way may not be possible for full-scale applications. Alternatively, inline mixers might be investigated to properly condition the dewatering aids for real operations. It would be very beneficial to conduct dewatering tests to investigate the effect of different types of inline mixers at various solid contents and flow rates.

- A two-step hydrophobization dewatering technique was successfully tested in the laboratory on clay samples. The results confirmed that significant moisture reductions and increased kinetics can be obtained. To be able to confirm the moisture reductions that were obtained in laboratory tests, pilot-scale tests should be conducted under various operating parameters.

- In the two-step hydrophobization technique, the first step involved treatment with a cationic reagent and the second step involved use of novel dewatering aids. Dewatering aids are effective at relatively low dosages; however, in some cases, the required dosages of cationic reagent were high. This may not be acceptable for full-scale applications. Taking this fact into account, a series of laboratory tests should be conducted to investigate new types of cationic reagents which may be used at considerably lower dosages.

- Foam aided dewatering tests were conducted on clay samples. The results showed that this technique can be successfully utilized over a wide range of pH values to dewater the clay particles. Additional laboratory tests should be conducted using various solid contents and particle size distributions. Afterward, pilot-scale work would be needed to better define the effectiveness of the technique.

- While lowering the moisture content of fine particles, dewatering aids have been shown to also increase the filtration kinetics. This is especially important for coal processing plants that utilize in horizontal belt filters (HBF), because they are able to take full advantage of the increased kinetics. Plants using HBF should be selected to further evaluate the combined benefits of dewatering aids.