1.1 Background

The production of fine coal is increasing as a percentage of all coals being processed in coal cleaning plants. This is due in part to the growth of continuous mechanical extraction methods. Moreover, it is desired to ground coal finely for sulfur liberation to meet clean air standards. It is well known that, as the particle size of a coal decreases, the surface area of particles exposed to water increases. As a result, the final moisture content of the processed fine coal, which is proportional to the surface area, is higher than the coarser sizes at the end of the cleaning stage. The fine coal dewatering is regarded as the most difficult unit operation in coal preparation industry today considering its inefficiency and high cost. For the time being, many coal companies would rather discharge fine coal than clean it, resulting in a waste of valuable resources and creating environmental problems.

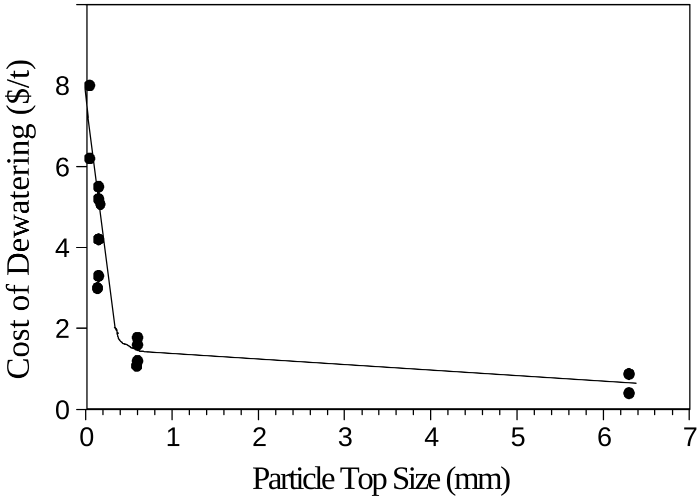


Figure 1.1. Estimated coal dewatering costs at different particles sizes (Couch, 1991).

The definition of fine coal varies from plant to plant, depending on the type of separation and the cleaning equipment available. Most of the time it refers to the material that passes through a minus 28 mesh Tyler screen (sieve opening of 0.6 mm). In some plants, it goes further down to minus 100 mesh (0.15 mm). At present, there are only two methods of removing moisture from fine coal, namely, mechanical dewatering and thermal drying. The mechanical dewatering methods are inefficient, typically producing 20-35% moisture in the product stream. The thermal drying is the only method that can reduce the moisture to satisfactory low level, but it is an expensive technique and it creates environmental pollution. Consequently, it is difficult to obtain permits for installing new thermal dryers in many states.

Figure1.1 illustrates the estimated cost of dewatering of coal at different particle sizes (Couch, 1991). It shows a drastic increase in dewatering expenses for particles smaller than 0.5 mm. In an effort to solve this long-standing problem, Virginia Tech has recently developed novel dewatering reagents. The efficiency of the mechanical dewatering was highly improved in the presence of these reagents in the system as dewatering aids. The moisture contents of various fine coal samples were successfully decreased by approximately 50%. Furthermore, they are environmentally safe to use; thus eliminating environmental concerns.

1.2. Literature Review

1.2.1. Dewatering of Coal

1.2.1.1. Water in Coal and Filter cake Structure

The Nature of Coal Surfaces

Coal is a naturally hydrophobic substance basically made of carbon. It forms in the presence of many geological, physical, chemical, and biological events. Depending on the complexity in the formation of coal, it is a non-uniform, heterogeneous material. There are many impurities in the coal structure. Also, the chemical composition and physical properties of the coal vary due to the changes in the formation conditions, which are called the macerals of the coal. As a result, the coal can be described as an aggregate of different macerals and minerals. The heterogeneity of the coal structure is also reflected in the surface of coal particles.

The characterization of the surface of fine coal is an example of the complex problem of characterizing powder surfaces. This phenomenon was surveyed in detail and the surface identification was classified into two broad classes (Parfitt and Sing, 1976). These are the physical and the chemical aspects. The physical aspects of surface characterization are concerned with particle size and shape. The chemical aspects, on the other hand, are devoted to identifying the nature of the surface functional groups, which are largely responsible for determining the types of interaction at the solid-liquid interface. The physical structure of coal is studied by surface area and pore size distribution analyses. The total surface area of the coal increases as the number of the pores in the coal structure increases.

The pores are classified as micropores, mesopores and macropores according to their sizes. The size ranges of these classes are given by International Union of Pure and Applied Chemistry (IUPAC, 1972) as follows,

1. Micropores (very small pores) - Less than ~2 nm
2. Mesopores (intermediate sized pores) - Between ~2 nm and ~50 nm
3. Macropores (large pores) - Above ~50 nm

Pore size distributions are often determined by the mercury porosimetry technique (Shaw, 1980). The volume of mercury (non-wetting to most solids) which can be forced into the pores of the solid is measured as a function of pressure. The pore size distribution is calculated based on the Laplace equation, which relates the pressure difference across a curved liquid surface to the diameter of the pore. (Laplace equation will be explained in detail later in this chapter). The scattering and gas adsorption techniques are also used to determine the pore structure and pore size distribution of the porous solids or particle beds. Another type of device was developed by Porous Materials Inc., is called the capillary flow porometer (Porous Materials Inc, 1997). This device determines the pore size distribution of a sample filled with a fluid. The pressure at which the largest pores are emptied of fluid indicates the bubble point of the sample. Then the pressure is increased until all the pores are emptied. According to the variations in required pressure to pass the liquid or gas through the sample, pre-size determinations are completed. This kind of a technique might be practical to determine the pore size distribution of particle beds.

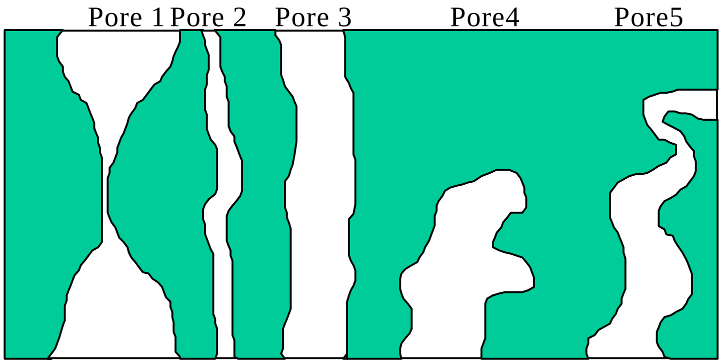


Figure 1.2. The typical pore types found on the surfaces (Porous Materials Inc, 1997).

Figure 1.2 shows the types of pore structures. Some pores are considered “trough pores”, which means they travel continuously through the material from one side to the other (pores 1, 2 and 3). The pores that do not pass all the way through the substance are called the “dead-end” pores. Pore type 1 is called a “bottleneck” pore. The diameter of this pore is measured at its narrowest restriction. A porometer will determine that, pore 2 has a larger diameter than pore 1 although the total volume of pore 1 is greater than that of pore 2. This is because the narrowest restriction of pore 2 is larger than the narrowest restriction of pore 1. Pore 3 is the largest pore and shall be the first one that opens under pressure. Pore 3 thus yields the “bubble point” of the sample. Neither of pore types 4 or 5 is measurable by a porometer since they are not “through pores”. A porosimeter can be used instead.

The chemical characterization of coal requires a knowledge of the functional groups on the surface. For a freshly ground coal sample, functional groups on the particle surfaces would reflect the bulk distribution of the chemical structure. Figure 1.3 shows the major functional group types identified in coal (Whitehurst, 1978). These are oxygen, sulfur, nitrogen compounds of carbon and metal salts. Oxygen occurs on the coal surface predominantly as phenolic or etheric groups with fewer amounts of carboxylic acids or esters; some carbonyls have also been identified. Sulfur has a similar chemistry to oxygen. Nitrogen occurs in the coal structure mainly as pyridine or pyrrolic type rings and metals are found as salts or associated with porphyrins. The ratio of these groups varies depending on the rank and the physical formation environment of the coal.

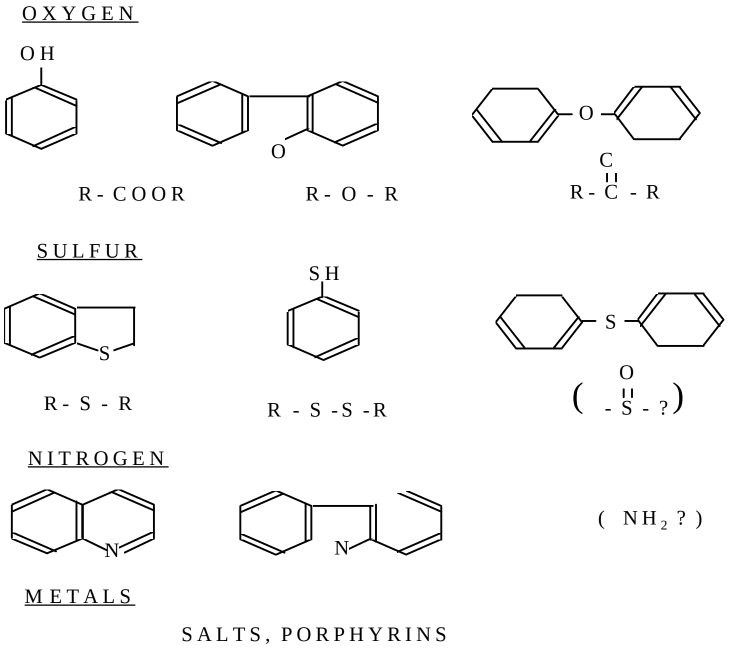


Figure 1.3. Functional groups in coal (Whitehurst, 1978).

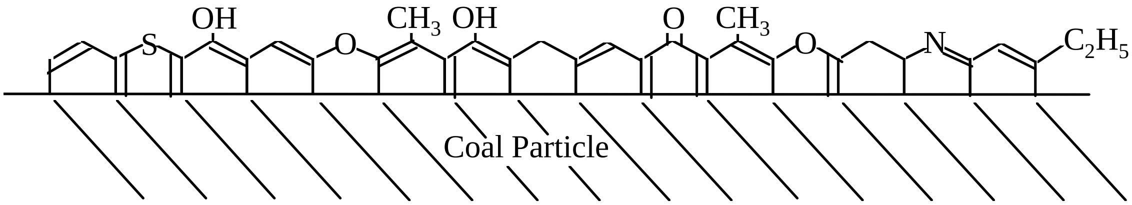


Figure 1.4. Schematic Representation of possible portion of coal surface (Oven, 1988).

A schematic representation of a portion of coal surface is seen on Figure 1.4 (Owen, 1988). The basic surface heterogeneity is apparent in this representative model. The aromatic, aliphatic, and heteroatomic entities on coal surface present a wide variety of polar and non-polar adsorption sites to which dewatering additives might adsorb. A surfactant molecule could bind either its hydrophilic head or its hydrophobic tail to such a surface. In practice, both kinds of behavior have been reported.

Table 1.1. Percentage Oxygen Functionality in Coal (Ruberto and Cronauer, 1978).

Functionality	**Subbituminous Coal**	**Bituminous Coal**
Hydroxylic (-OH)	5.6	2.4
Carboxylic (-COOH)	4.4	0.7
Carbonylic (=C=O)	1.0	0.4
Etheric (-O-)	0.9	2.8

Among all the groups on the coal surface, oxygen is the most abundant one. Moreover, it has the greatest effect on coal reactivity, so it should also be important in physical interactions. Table 1.1 illustrates the distribution of oxygen-containing functionality in a bituminous and subbituminous coal (Ruberto and Cronauer, 1978). As the coal gets older, the surface groups changes under the effect of oxidation. A detailed study on the surface changes of coal after oxidation took place was performed by X-ray photoelectron spectroscopy (Grzybek and Kreiner, 1997). The influence of liquid phase oxidation on the surface heterogeneity of coal was examined on coals of different ranks in this study. Upon oxidation the changes in organic matter were reported as follows:

i. Carbon content decreases while the amount of oxygen strongly increases.
ii. The amount of nitrogen on the surface changes similarly as carbon, which was proven by the constant value of nitrogen/carbon ratio throughout the treatments.
iii. Sulfur content increases for the coal types that contain mobile sulfur. Sulfur oxidizes and moves to the surface of the coal.

It was concluded that the increase in oxygen amount was connected with the formation of CO^-, C=O, and COO^- groups. The surface oxygen-containing groups were formed by oxidation of aliphatic chains and/or already present phenol/alcohol-like species. The oxygenated group population varied depending on the coal rank. For the high rank coals, the process leaded mostly to the formation of C=O. The lower rank coals were observed to get oxidized more easily compared to the high rank coals.

Water in Coal

The moisture in coal can be classified into two parts, surface moisture and inherent moisture. The surface moisture of coal is the type of moisture that can be removed partially by using mechanical dewatering methods, whereas the inherent moisture is a part of the coal structure and it can only be reduced by thermal drying. The surface water of a bed of coal particles vary from zero to over 30%, depending on the following factors (Keller at al, 1979):

i. Rank of coal
ii. Ambient relative humidity
iii. Mineral composition and concentration in the sample
iv. Particle size distribution
v. Temperature
vi. Mechanical treatments
vii. Water impurities
viii. Prior treatments, e.g., oxidation, chemical additives, etc.

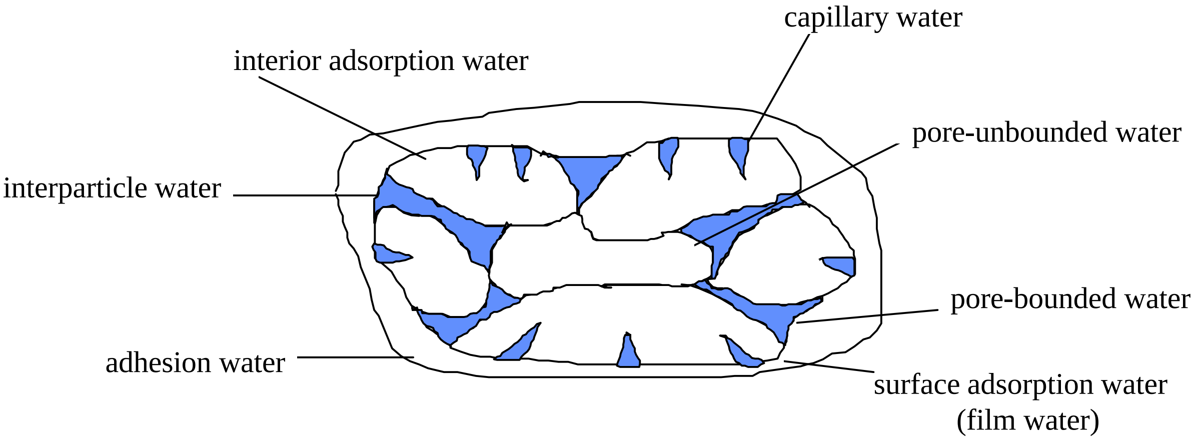


Figure 1.5. Water associated with coal (Gala, 1989).

Figure 1.5 shows the types of water associated with coal (Gala, 1989). These are,

- *Interior adsorption water:* water deposited during coal formation and contained in micropores and microcapillaries within each coal particle.
- *Surface adsorption water (film water):* a layer of water molecules adjacent to coal molecules, but only on the particle surface. This water may be chemically bound to the coal particle.
- *Capillary water*: contained in small crevices found on the surface of coal particles.
- *Interparticle water:* contained in capillaries and small spaces found between two or more particles. It can be divided into two parts as pore-bounded and pore-unbounded moisture. Pore-bounded moisture is entrapped between the contact points of solids; its motion depends on the ratio of capillary forces to the air pressure gradient at dewatering. Pore-unbounded moisture is located in the space between solid particles; its motion also depends on the action of molecular forces.
- *Adhesion water:* a layer or film of water around the surface of individual or agglomerated particles.

Interior and surface adsorption water are considered as inherent moisture because they are tightly bound to the coal molecules. Some part of the capillary water can also be considered as the inherent moisture since it is very difficult to remove the moisture in small capillaries, especially in microcapillaries. All of the others are considered as the surface moisture of coal that can be removed by mechanical techniques up to some extent.

Filter cake Formation and Water in Cake Structure

The efficiency of the dewatering process is strongly related to the properties of the filter cake structure. The deposition of the particles during filtration leads to the build up of a filter cake, which consists of a packed bed of particles with a complex system of interconnected, inter-particle voids. The final moisture content remained in cake varies depending on the structure of the filter cake, the physical properties of the fluid, the solid/fluid interfacial properties, and the temperature and the pressure gradient (Wakeman, 1975).

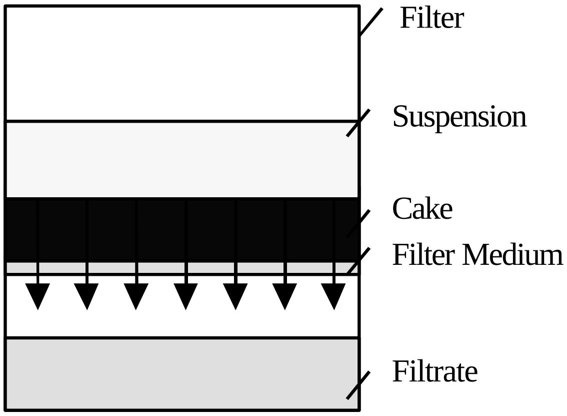


Figure 1.6. Operating scheme of a filtration process (Cheremisinoff and Azbel, 1983).


(a) (b)

Figure 1.7. Postulated interaction between the cake particulates and the filter media (Willis, 1986).

Figure 1.6 shows the formation of a coal filter cake. The fine coal particles in the form of slurry are subjected to the pressure in a filtering system. Under the effect of the applied pressure, a flow through the filter medium starts and the particles settle on the filter medium forming the filter cake. Meanwhile the filtrate is collected separately. Initially, there is only filter medium resistance against the applied vacuum. As the filter cake builds up, the total resistance to flow increases due to the resistance of the forming filter cake. If the filter medium is blocked by the coal particles as shown in Figure 1.7, the filter medium resistance goes to a maximum and filtration stops. This implies the importance of selecting a proper size filter medium considering the particle size distribution of the coal sample. When the particles are very much larger than the media openings (a), no clogging occurs. However, when openings have almost the same size with the particles, filter media clogs (b) and the filtration stops immediately (Willis, 1986).

During the cake formation, filter cakes stratified with large particles at the bottom of the cake and fines in the top layers. The segregation results in a cake of non-uniform structure with a wide variation in properties from one layer to another. The compact, low-permeability nature of the top section makes removal of water during the dewatering stage more difficult. Mixing the coal slurry during the cake formation was found to be an effective solution to form a uniform cake structure (Sung at al., 1994). Pressure filtration experiments were conducted on a coal slurry sample with a large particle size distribution. The slurry was mixed at the upper level in the funnel to prevent the deformation of the filter cake. The results showed that the moisture contents of the cakes with a uniform structure were lower than the moisture content of the cakes with a non-uniform structure that formed without mixing.

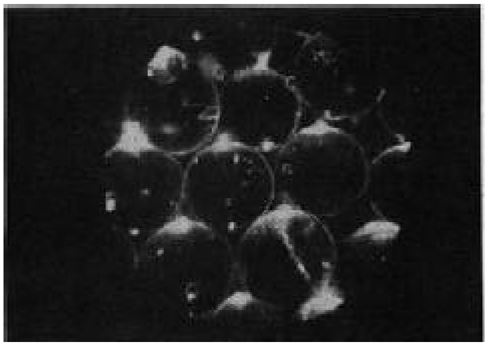
There are some models developed to simulate the build-up of a filter cake (Wakeman, 1981 & Theilander and Fathi-Najafi, 1995). In the model presented by Wakeman, the formation of the cake was examined as a function of the pre-determined time intervals. Theilander and Fathi-Najafi used a slightly modified approach to Wakeman's work and they also developed a computer program. The later model discretised the build up of the filter cake based on the volume of slurry filtered. Since the amount of deposited solid material per unit time decreases during the filtration, this model represented the build up of the filter cake more realistically. The computer program developed by Theilander and Fathi-Najafi can be used to compute the cake height, the pressure profile and the local specific filtration resistance. It was advanced on the basis of several assumptions. During the filtration process the applied pressure and the filter medium resistance were assumed to remain constant, i.e. no blinding occurred on the filter medium by the fine particles. The second assumption was related to the porosity of the cake. It was assumed that the layers were formed uniformly and the porosity remained constant in all the layers. This model can be explained simply as follows: at the start (t=0) no filter cake forms, and the fluid flows through the filter medium only. After a short time, t_1, the first layer $\Delta x_1(t_1)$ of the filter cake forms, and a filtrate volume corresponding to V_1 is obtained. As a result, the flow rate of the filtrate decreases and the pressure drop is divided between the filter medium and the first layer. As the layers build up, the same procedure continues. However, since the cake structure is not uniform this model underestimates the resistance in the region closest to the filter cloth, and overestimates the resistance in the region closest to the top of the filter cake. While several assumptions are always required to be made for modeling, the real conditions in cake formation are more complicated and should be studied more carefully. As it was mentioned previously, filter cakes have very heterogeneous structures depending on their particle size distributions. Mixing the slurry might be helpful to keep the porosity uniform at each forming layer as the filter cake builds-up but it increases the cake formation time and such a procedure is not applicable to all types of filtration processes.

A filter cake may be defined as compressible or incompressible depending on the change in the cake structure and porosity during the filtration process. When the cake structure is composed of particles that are readily deformed or become rearranged under pressure, the structures generally are termed compressible cakes. Those that are not readily termed are called semi-compressible, and those that deform only slightly are considered incompressible. The filter cakes of fine coal are generally regarded as incompressible, since they are relatively dense as formed and not easily compacted under the pressures involved in filtration. This implies that the formed cake structure and porosity will remain the same during the filtration process.

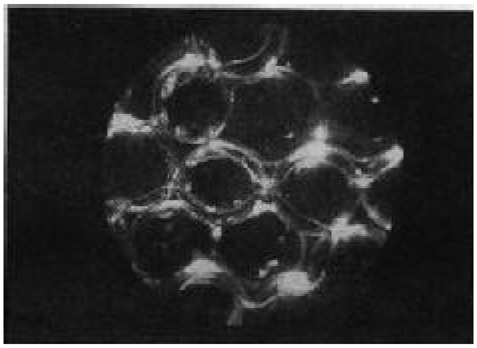
The porosity of a filter cake can be defined in two ways (Tosun and Sahinoglu, 1986). It can be defined as the fractional free area available for flow divided by the total cross-sectional area, or as the volume of fluid in the structure filling the pores divided by the total volume. In the filtration literature these definitions are used interchangeably. The volumetric approach to determine the cake porosity ε, can be represented by equation 1.1,

$$e = \frac{v_0}{V + v_0} \qquad [1.1]$$

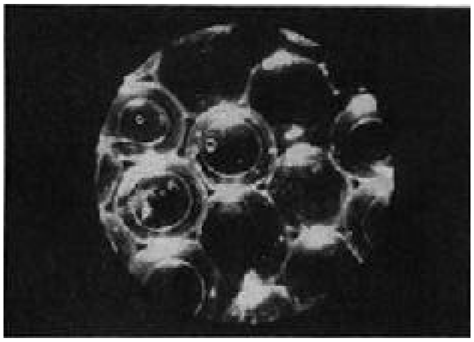
where V and v_0 are the volumes of the solid and void space, respectively. It is possible to calculate the porosity of an ideal filter bed made of equal spheres in regular arrangements by utilizing this equation (Harris and Smith, 1957). The porosity of randomly packed particles was also studied on a particle bed of glass spheres with a log-normal radii distribution (Wise, 1952). However, the fine coal filter cakes have a very complex pore structure and these kinds of approximations can only give an empirical idea about the real porosity of the cake. It is more convenient to use the techniques explained to measure the porosity of the particle beds in the previous section to determine the cake porosity when an accurate data is necessary.



Capillary



Funicular



Pendular

Figure 1.8. The three possible equilibrium conditions of a fluid in a filter cake (Phillips and Thomas, 1955).

Including the cake formation, a typical filtration cycle involves three stages where the fluid flow through the cake changes continuously (Ranjan and Hogg, 1996). These are:

- cake formation, characterized by particle deposition and single-phase flow of water through the cake
- initial dewatering in which there is two-phase flow of air and water
- final dewatering where flow is again primarily single-phase (air)

During these stages of the filtration process, the fluid remaining in the cake structure takes three well-defined positions (Phillips and Thomas, 1955) as illustrated in Figure 1.8. These are known as: capillary state in which the voids in the structure are completely filled by the fluid to a fairly well defined height; the funicular state in which the voids are not filled but there is a continuous network of fluid in solid, and the pendular state in which fluid is present only at the points of contact of the particles.

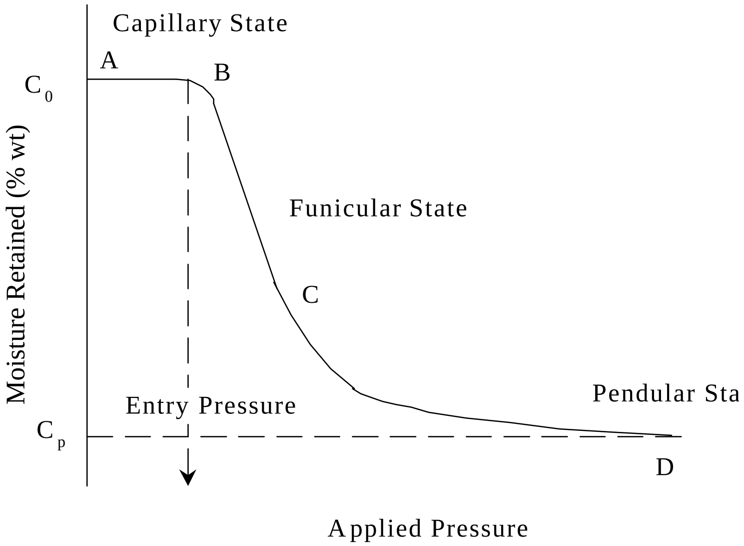


Figure 1.9. Generalized pressure/moisture relationship for cake filtration (Veal at al., 1995).

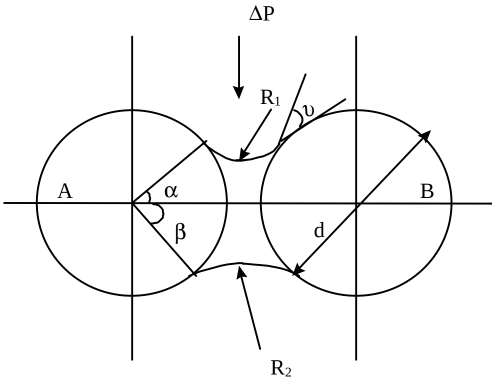


Figure 1.10. Equilibrium state between pressure gradient and capillary forces for pore moisture in a cake (Cheremisinoff and Azbel, 1983).

At the capillary stage, the atmospheric pressure is in equilibrium with the capillary forces present at the air/water interface. When the applied pressure level overcomes this capillary pressure, drainage starts. The flow of liquid continues until the limiting pendular state is reached. Further application of pressure leads to channel of air through the cake. The pressure moisture relationship based on this mechanism is represented in Figure 1.9 (Veal at al., 1995).

Figure 1.10 illustrates schematically the filtration process between two spherical particles A and B (Cheremisinoff and Azbel, 1983). These particles are in the capillary state with a small film of liquid trapped between them. The arrows indicate that the film of moisture is acted on by a pressure gradient ΔP, which is in equilibrium with the capillary forces that are in the opposite direction. This results in a displacement of the moisture from its initial position, so that the radius of the curvature of the meniscus R_1 becomes less than R_2, and the angle of wetting α becomes less than angle β. In this case, gravity forces are much less than the pressure gradient and may be neglected. According to the Laplace equation, wetting angle υ is zero and the required pressure difference is,

$$\Delta P = \frac{2\gamma}{d}\left(\frac{\cos\alpha}{1-\cos\alpha} - \frac{\cos\beta}{1-\cos\beta}\right) \qquad [1.2]$$

where ΔP is the pressure gradient (N/m^2), γ is surface tension (N/m) and d is the diameter of solid particles (m). It is obvious that as α decreases and β increases, ΔP increases; however, the limiting value of pressure gradient is not infinity, because at some small value of α air breaks through the film between the solid particles, and the above equation becomes no longer valid. If the distance between the particles approaches to zero, the limiting value of ΔP goes to infinity due to a decrease in α, at which point air breaks through. Thus, the pressure drop required in cake filtration increases with the decreasing pore size.

Although the pressure is applied to the filter cake uniformly, not all the pore-bonded moisture can be acted equally. As it is shown in Figure 1.11, for a pressure gradient that changes in the direction shown by the arrow, only the moisture located in zones 1 and 2 can be displaced by air. The moisture in zones 3 and 4 will stay in the porous layer because it is not affected by the pressure gradient. Therefore, only part of the moisture can be removed. It is possible to reduce the final cake moisture content to further down by surface chemical treatment of the particles in the presence of the dewatering aids. The significance of the alterations in the surface chemistry of the coal in improving the efficiency of dewatering will be discussed in detail in the following section.

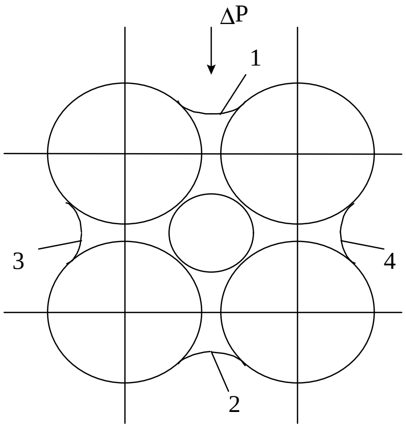


Figure 1.11. Distribution of pore-bonded moisture in different zones (Cheremisinoff and Azbel, 1983)

1.2.1.2. Surface Chemistry of Coal Dewatering

Adsorption on the Coal Surface

The limited efficiency of the mechanical dewatering techniques in reducing the moisture content of the filter cakes increased the usage of chemicals as dewatering aids. The dewatering aids can be classified as surfactants, flocculants and hydrophobizing agents. Adsorption of these reagents onto the coal surface is a complicated and important phenomenon. The facility and strength of adsorption is very largely controlled by three factors (Myers, 1988):

i. the chemical nature of the species being adsorbed, including the nature of the head group (anionic, cationic, nonionic, etc)
ii. the nature of the solid surface on which the surfactant is being adsorbed (highly charged, nonpolar, etc)
iii. the nature of the liquid environment (in water the pH, electrolyte content, temperature, additives, etc).

Based on these three main factors, the response of the coal-in-water slurry to a dewatering additive varies depending on: the type of coal, particle size distribution, degree of weathering, extend of previous beneficiation (for example, adsorbed flotation aids may still be present), water quality, process temperature, type of mechanical dewatering devices utilized, rate of production, nature and purity of the dewatering additive, dosage level, and the time of contact between additive and coal (Owen, 1988).

Usually, adsorption on the coal surface occurs in a molecular level due to the intermolecular interactions between the surface of the coal and the adsorbing molecule, like electrostatic repulsion, van der Waals and the hydrophobic attractions. Van der Waals forces are short-range attractive forces. They result from the nature of electron motion in molecular orbitals. Although they are weak forces compared to the hydrophobic attraction forces, in the absence of any other interactions, they play a very important role for adsorption of the molecules onto the surfaces. The electrical interactions are repulsive and they take place between two similarly charged particles in a solution. The attractive hydrophobic interactions are observed between two hydrophobic particles and they are effective even at long interparticular distances.

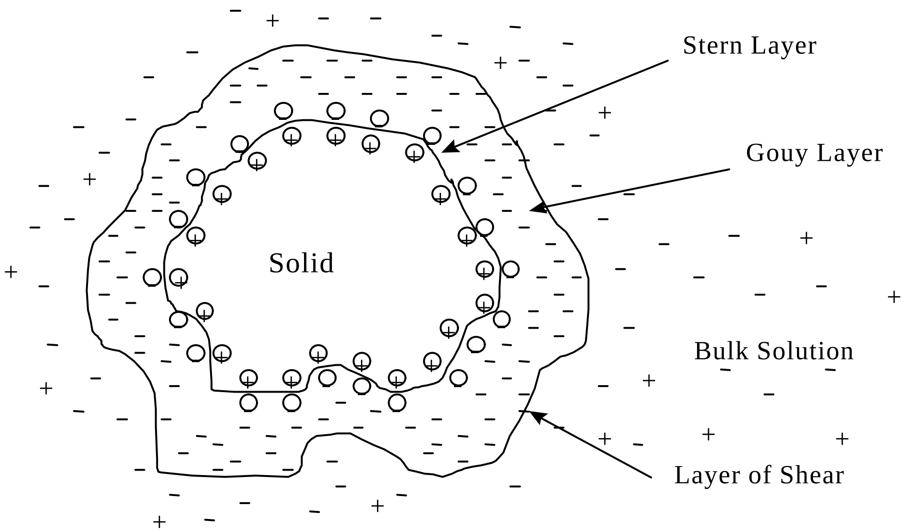


Figure 1.12. Illustration of the electrical double layer (Stelma, 1978).

When a charged solid is placed in water, the oppositely charged ions in the solution are attracted through the surface of the solid to balance the electrical charge. Such a situation gives rise to the formation of the electrical double layer (Stelma, 1978). The ions form two regions adjacent to the surface. The first consists of adsorbed ions, which is called the “Stern layer”. The second is a more diffuse layer known as the “Gouy layer”. The sum of the charges in all of the layers counter balances the charge in the solid surface. A schematic representation of the electrical double layer is shown in Figure 1.12. The formation of electrical double layer stabilizes the same type of particles in a solution since they are loaded similarly and as a result tend to repel each other.

The surface potential is at a maximum value on the particle surface. This is because, the opposite electrical charges are attracted to the surface and they form a dense layer around the particle. As the distance from the particle surface increases, the surface potential decreases. Figure 1.13 shows the change in surface potential by distance. Stern layer is represented in two layers as inner and outer Helmholtz planes (Leja, 1982). The specifically adsorbed unhydrated ions form the inner Helmholts plane. At a distance of d, which is the diameter of the adsorbed ions in the first layer, more weakly adsorbed hydrated ions are placed as a second layer, forming the outer Helmholtz plane. At longer distances the diffuse region is reached.

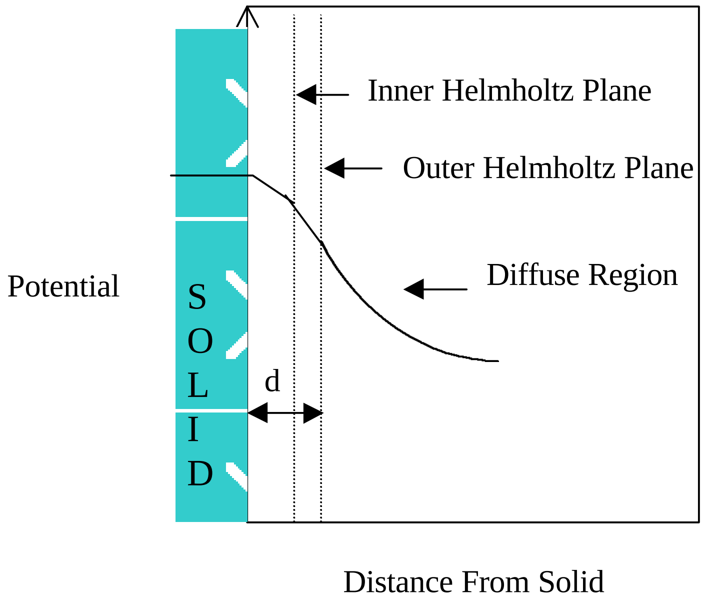
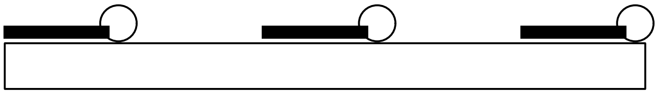


Figure 1.13. The electrical double layer variation of the potential with distance, showing the inner and outer Helmholtz planes (Stelma, 1978).

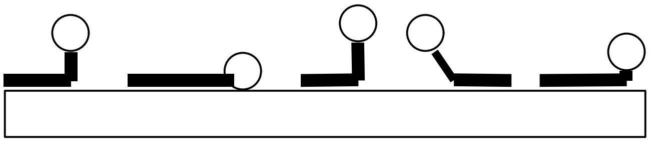
The electrical charges on the solid surface create the attraction or repulsion between the particles and the charged additives. When the charge of the additive is the same with the surface charge, there occurs a repulsion between the particle and the additive. This repulsion stabilizes the additives in the solution, and they do not adsorb onto the particle surface. On the other hand, when there are counter charges between the surface of the particle and the head group of the surfactant, adsorption occurs either by ion pairing or ion exchange. In ion exchange, the adsorbing molecule displaces the similarly charged counter-ion adsorbed on the substrate surface, while in ion pairing, molecule adsorbs onto an oppositely charged site, which was not previously occupied by a counter-ion.

The third type of adsorption on the surfaces takes place as a result of the hydrophobic interactions. The hydrophobic groups of the dewatering aids tend to escape from the aqueous environment or in other words aqueous environment pushes the hydrophobic groups out (Myers, 1988). In that case, the hydrophobic tails of the surfactant molecules tend to adsorb onto the hydrophobic portions of the surfaces. This orientation of the surfactant molecule causes the head group face through the solution. If the surfactant has a polar (ionic) head group, this type of an adsorption converts the hydrophobic particle surface to a charged, hydrophilic surface.

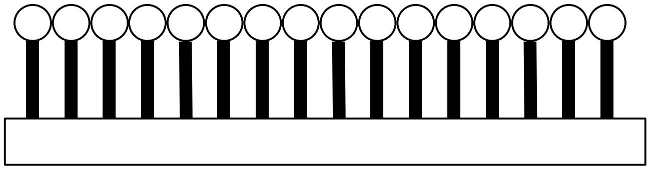
The surface of the coal is naturally hydrophobic. Although it is negatively charged in water due to the tendency of the OH^- ions to go onto the surface, the adsorption on the coal surface may occur under the effect of Van der Waals and hydrophobic forces. In such a case the orientation of the adsorbed molecules will be such that the hydrophobic groups are associated with the solid surface with the hydrophilic head group directed toward the aqueous phase as explained above. In the early stages of adsorption, it is likely that the hydrophobic tail will be lying on the surface much like trains or L’s. As more molecules adsorbed, surfactants start to orient perpendicularly and result in approximately close-packed monolayer as seen in Figure 1.14. To keep the coal surface hydrophobic, it is required to use non-ionic surfactants or otherwise this type of an orientation strongly alters the coal surface hydrophilic.



Low surface coverage: adsorption primarily in “trains”.



Intermediate surface coverage: adsorption in “trains” and ‘L’s”.



Surface saturation: close-packed with minimum area per molecule.

Figure 1.14. Adsorbed surfactant orientation as a function of total surface coverage (Myers, 1988).

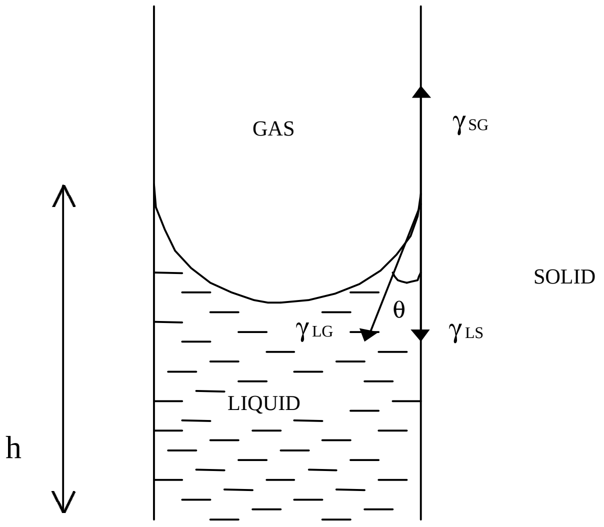


Figure 1.15. Capillary rise and interfacial tensions at the point of three phase contact in a capillary (Stelma, 1978).

The manner of adsorption of the surfactants, flocculants and the hydrophobizing agents alters the chemistry of the coal surface and the solution environment in different ways. These effects will be explained by discussing the capillary theory of dewatering and considering the variations in the surface tension, change in hydrophobicity and tendency to flocculation of the coal particles in the presence of these different additives.

Capillary Theory of Dewatering

It is a very common practice to consider a filter cake as a bundle of capillary tubes. Although the filter cake structure is quite complicated as it was explained in the previous section, this assumption is very useful to explain the theory of filtration in general. Figure 1.15 shows the rise of liquid in a single capillary tube. For the large capillaries, the capillary rise continues until an equilibrium is reached between the gravitational pull on the mass of the column of fluid and the force exerted by the surface tension (Phillips and Thomas, 1955). This equilibrium can be expressed as in equation 1.3.

$$\Delta\rho gh = \frac{2(\gamma_{LG}\cos\theta)}{r} \quad [1.3]$$

In this equation, $\Delta\rho$ is the difference between the liquid and the gas densities, g is the gravitational constant, h is the capillary rise, γ_{LG} is the interfacial tension of the liquid at the gas interface and θ is the receding contact angle between the capillary wall and the liquid. The relationship between the interfacial tensions can be given by the Young's equation as,

$$\gamma_{SL} = \gamma_{LG}\cos\theta - \gamma_{LS} \quad [1.4]$$

where γ_{LG} and γ_{LS} are the interfacial tensions between the liquid-gas and the liquid-solid surfaces.

In the small capillaries, gravitational forces become negligible and the equilibrium in the capillary is reached between the atmospheric pressure and the force exerted by the surface tension. The capillaries formed between the particles of a fine size coal bed are small enough as it was mentioned in section 1.2.1.2, as a result the gravitational forces can be neglected. To remove the moisture in the capillaries, a pressure differential ΔP, must be applied. The force per unit area necessary to counteract the capillary pressure within a capillary of radius r is given by the Laplace equation. Equation 1.2 representing the Laplace equation can be rewritten as expressed in equation 1.5 for a regular capillary. According to this equation, for a capillary of radius r, the required pressure to decrease the liquid level goes down as the surface tension decreases, capillary radius increases and cos θ decreases. This means the receding contact angle θ should be increased to reduce the required pressure to decrease the level of water in the capillary. When the contact angle is zero, the liquid in the capillaries will drain with the application of an applied pressure differential, ΔP, just greater than $2\gamma/r$.

$$\Delta P = \frac{2\gamma_{LG}\cos\theta}{r} \qquad [1.5]$$

The dewatering aids help reducing the moisture content of the filter cake by modifying the particle surfaces in a way that the required pressure differential decreases. Surfactants play an important role in decreasing the surface tension of the liquid-air interphase and the hydrophobizing agents increase the contact angle of the surface. The contact angle of the coal surfaces increase by using surfactants when they adsorb onto the particle surface by orienting their hydrophobic tails facing through the solution. The flocculants on the other hand, increase the particle size by bringing the particles together and as a result the radius of the capillaries in the cake structure increases. According to the Laplace equation increase in the capillary radius is another factor helping to decrease the capillary pressure differential.

Surface tension lowering

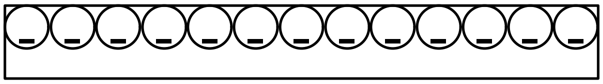
Surfactants are usually used to reduce the surface tension of the coal-water slurry. Although the coal is naturally hydrophobic, coal particles are negatively charged in water due to the tendency of the OH^- ions to go onto the particle surface. This negative charge on the surface of the coal creates an electrical interaction between the charged surfactants and the surface. Under the effect of the electrical interactions, surfactant molecules either stay in water or adsorb onto the coal surface. The negatively charged anionic surfactants are expected to stay in the solution as a result of the electrical repulsion and reduce the liquid-air surface tension, whereas the positively charged cationic surfactants are expected to adsorb onto the coal surface. This was observed in a recent study where the performance of sodium dodecyl sulfate (SDS) and dodecyl amonium bromide (DAB) were compared as dewatering aids for fine coal (Singh, 1997). It was noticed that the anionic surfactant SDS, was more effective in reducing the surface tension of the solution than the cationic surfactant DAB. That was due to the adsorption of the DAB molecules onto the surface of the coal, so that a less amount of the reagent was left in the solution to cause a reduction in the surface tension.

Most of the surfactants were observed to decrease the contact angle of the surfaces as well as they decrease the surface tension of the solution. As a result, they are used as to wet the coal surface in the coal mining industry. The sulfosucconates are the best examples of these wetting agents. An early work on surface tension effect showed that for low concentration of Aerosol OT (sodium 2-ethylhexyl sulfosucconate) the dewatering effect due to the decrease in the surface tension of the slurry was significant but small (Silverblatt and Dahlstrom, 1954). It was also concluded in the same study that the viscosity had a greater effect on reducing the moisture content than the surface tension. In another work, it was shown that the contact angle of coal decreased to 25^o from 45^o in the presence of a wetting agent in the solution but there was no reduction in the moisture content of the coal filter cake (Phillips and Thomas, 1955).

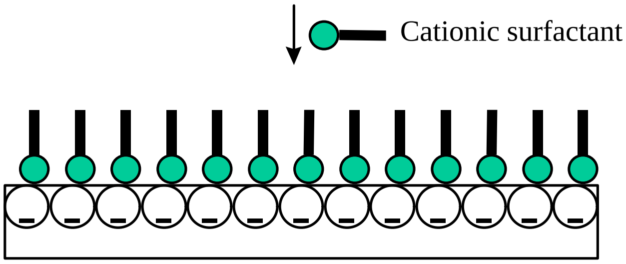
Change in Hydrophobicity

The hydrophobicity of the surfaces is measured as a function of the change in solid-liquid contact angle. If the contact angle between the solid and the liquid phases is greater than zero, a substance is called hydrophobic. The contact angle of the coal usually varies between 45^o to 65^o, depending on its rank. According to the Laplace equation, the higher the contact angle the easier the dewatering becomes. The contact angle of the coal surfaces can be increased by the adsorption of surfactants or hydrophobizing agents onto the surface properly. Adsorption phenomena on the hydrophobic surfaces were explained in the previous section. Although coal is a naturally hydrophobic substance, there are some electrical interactions taking place on the coal surfaces. This is partially due to the charged impurities existing on the surface of coal and partially because of the formation of a negative charge on the coal surface in water environment due to OH^- ions.

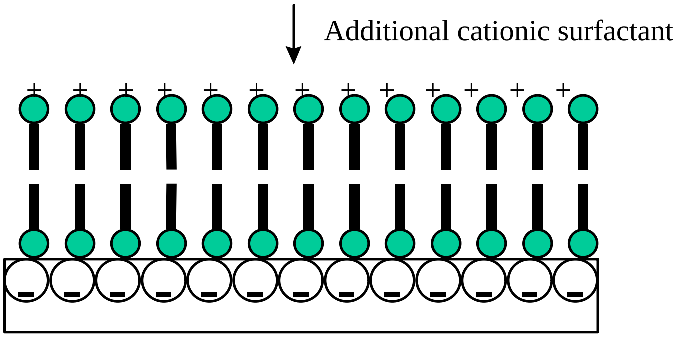
Figure 1.16 shows the adsorption of a cationic surfactant onto the coal surface. At an appropriate dosage, surfactants form a monolayer and at that stage they have a close-packed orientation with a minimum area per molecule. The cationic head groups of the surfactants adsorb onto the coal surface as a result of the electrical attraction. The hydrophobic tails of the surfactant molecules face through the water and this type of an orientation enhances the hydrophobicity of the surface. However, at higher concentrations, a second layer forms at which surfactant associate with the first layer through the hydrocarbon tails. That bilayer formation drives the surface hydrophilic again since the hydrophilic heads of the surfactants face water. As it can be concluded from all these explanations that, the selection of the surfactants to help coal dewatering is a very critical concept. The properties of a good surfactant to decrease the moisture content of the coal can be summarized as follows (Owen, 1989);



(a) Net negatively charged surface (counterions omitted)



(b) Surface charge zero: allsurface charges neutralized
by adsorbed surfactant.



(c) Surface charge reversed due to second layer of adsorbed
surfactant.

Figure 1.16. Schematic representation of surface charge reversal due to bilayer adsorption of the oppositely charged surfactant molecules (Myers,1988).

i. The surfactant molecule should adsorb on the coal oriented with the hydrocarbon tails extended to the water phase
ii. The hydrocarbon tail should be as hydrophobic as possible and still retain a degree of water solubility of the whole molecule.
iii. The head group should be the least hydrophilic group possible and still retain the solubility of the whole molecule.
iv. The adsorbed molecule should have a large effective area so that fewer molecules are required to effect the interfaces.

v. The surfactant should decrease the interfacial tension at the liquid-air interface as much as possible at the lowest concentration.

vi. High concentrations, causing bilayer formation should be avoided when the surfactant is added into the system.

In several studies with the bed of glass spheres, it was observed that the pendular state was reached faster when the contact angle of the surfaces increased (Harris and Smith, 1957, Phillips and Thomas, 1955). The water remained in the pendular state decreased from 11% for 0^o contact angle to 1.2% for 90^o contact angle after the treatment with a hydrophobicity enhancing surfactant (chloro-silane compound). Harris and Smith also studied the effect of contact angle on moisture reduction of a bed of coal particles. They used a wetting agent (concentrated solution of potassium permanganate) to reduce the advancing contact angle of the polished coal surface from 60^o to 0^o. According to the capillary rise theory, a contact angle of 60^o was supposed to decrease the moisture content of a bed of particles by 50%. However, they did not observe the same trend experimentally between the samples with 0^o and 60^o contact angles. They concluded that the poor improvement in the reduction of the moisture content was due to the complex porous structure of cake and the decrease in the contact angle by slime coating on the coal particles.

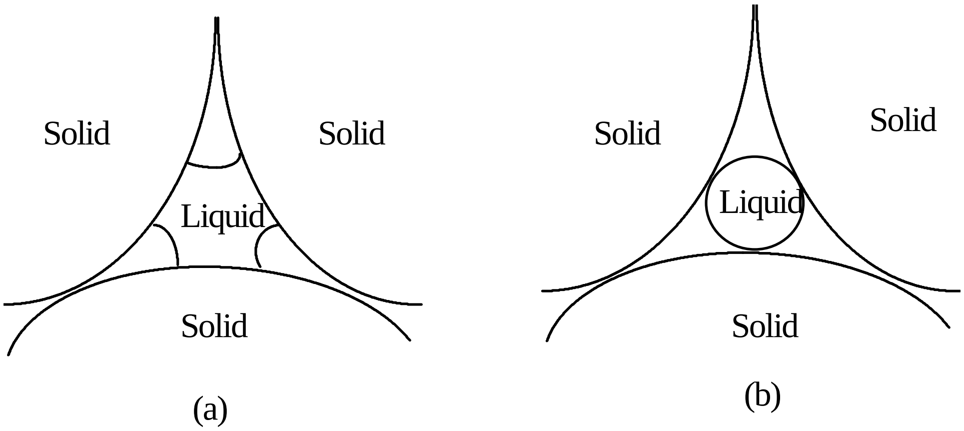


Figure 1.17. Moisture retention between non-wetting particles, (a) low contact angle, (b) high contact angle (Veal at al, 1995).

Figure 1.17 shows the moisture remaining between the hydrophobic particles for the surfaces with low (a) and high (b) contact angles (Veal, at al, 1995). In a solid-water-air system, when the contact angle is very low, complete wetting occurs. This causes the water to spread all over the particle surfaces as a film. On the other hand, when contact angle is high enough (~90^o), the residual water exists in the form of droplets on the surfaces. If such a system is drained naturally, the pendular state is reached at lower moisture content values since the capillary rise will be lower between the particles. In addition, under the effect of the applied pressure, it is easier to remove the water in the form of droplets from the coal surface compared to the film water which will always exist on the surface up to some extend.

Flocculation effect

The usage of coagulants and flocculants for dewatering is very common in the coal preparation industry. Both coagulation and flocculation processes bring the fine size coal particles together and as a result increase the particle size. As the particle size increases, the filter cakes form larger pores giving rise to the formation of larger capillaries through the cake. According to the Laplace equation, the required pressure to decrease the level of moisture in the capillary decreases with the increasing capillary radius.

As explained previously in this section, the coal particles are negatively charged in water and they tend to repel each other since they are similarly charged. To make the particles approach each other, it is necessary to reduce these repulsive interactions. The coagulants are used to neutralize the charges on the particle surface and decrease the thickness of the electrical double layer. This brings the particles closer and they coagulate under the effect of the attractive dispersion or hydrophobic forces. The typical coagulants are multivalent salts, such as alum, iron salts, lime and inorganic acids. The flocculants are different from the coagulants in that, they are polymeric substances. There are two types of flocculants, natural polymers like starch and gelatin and synthetic polymers like polyacrylamides and their derivatives. The synthetic flocculants require lower concentrations to flocculate the slurry compared to the natural polymers since they have much higher molecular weights.

Two basic mechanisms relevant to polymeric flocculations are bridging and charge patch mechanism (Owen, 1988). In the bridging mechanism, the polymer chain is adsorbed on the particle surface at only a few points, leaving the other part of the polymer chain extending into the solution for contacting other particles. This is observed when non-ionic, high molecular weight (>1 million) polymers are used. These types of polymers adsorb onto the coal surface by hydrogen bonding, van der Waals interactions or hydrophobic bonding. The charge patch mechanism applies to the adsorption of cationic, lower molecular weight (<1 million) polymers. They adsorb onto the negatively charged coal particles by electrostatic bonding. Since the polymer molecule is small relative to the particle, it covers only a small portion of the surface area, but since it is highly charged it effectively neutralizes a large area of negative surface sites. This cationic patch on the particle provides an area for direct electrostatic attachment to negatively charged sites on another particle. It should also be noted that the polymer adsorption usually takes place as a result of the increase in the entropy of the system by replacing the water molecules on the particle surfaces with the adsorbing polymer molecules. This creates a very large negative free energy in the system, which gives rise to the adsorption of the polymer spontaneously. Although the charge of the polymer plays a role in adsorption, usually the effect of the entropy change is much more dominant compared to the electrical interactions and regardless of their charge, the polymers easily adsorb onto the particle surfaces and bridge them to each other.

Although flocculation is a fast and effective method for solid-liquid separation, flocculants usually increase the final moisture content of a filter cake. That is because water is trapped into the particle flocs and it is not possible to remove this trapped water by using the mechanical dewatering techniques. New studies have shown that polymers can have other usage in dewatering. Synthetic cross-linked copolymers were used as superadsorbents for fine coal slurries (Dzinomwa and Wood, 1995). These kind of polymers can absorb water up to several hundreds of times of their own mass and expand in size, but still retain individual identity. When mixed into the slurry, they absorbed most of the water and they are then separated from the slurry by the help of the screens. It was discussed that the regeneration of the polymer from the system was also possible by changing the slurry pH or applying an electrical field.

1.2.1.3. Dewatering Kinetics

The kinetics of dewatering can be explained on the basis of the common assumption of considering the filter cake as a bundle of capillary tubes. Poiseuille studied the flow of fluid through a capillary tube in 1846 (Phillips and Thomas, 1955). His equation is given as,

$$\frac{dV}{dt} = \frac{\pi r^4}{8\eta}\frac{\Delta P}{L} \qquad [1.6]$$

where dV/dt is the volumetric flow per unit time, r is the radius of the tube, ΔP is the pressure drop across the bed, L is the bed (cake) thickness and η is the fluid viscosity.

In 1856, Darcy stated the basic concept for laminar flow through a porous bed. He assumed the fluid flow to be laminar since the pore sizes in the filter cake and the filer medium are small enough. The equation developed by Darcy is the equivalent of the Poiseuille's equation (Brownell and Katz, 1947) as expressed in equation 1.7,

$$\frac{dV}{dt} = K\frac{A}{\eta}\frac{\Delta P}{L} \qquad [1.7]$$

where A is the cross-sectional area of the cake and K is the permeability constant. The unit of permeability is "Darcy". A Darcy is defined as a unit of permeability equivalent to the passage of one cubic centimeter of fluid of 1 centi-poise viscosity flowing in one second under a pressure of one atmosphere through a porous medium of one square centimeter area and one centimeter length (Smith, 1985). According to Poiseuille's and Darcy's equations, permeability constant K is proportional to $r^2/8$, for the flow through a circular capillary. Kozeny improved the permeability concept to avoid representing a porous medium by a simple bundle of capillaries. He defined the mean hydraulic radius m, which is expressed as the total volume of liquid divided by the total wetted area (Gray, 1958). Kozeny's equation can be given by equation 1.8,

$$K = \frac{m^2\varepsilon}{k} \qquad [1.8]$$

where
$$m = \frac{\varepsilon}{S(1-\varepsilon)} \qquad [1.9]$$

ε is the porosity of the cake, S is the specific surface area (surface area of particles per unit volume) and *k* is the Kozeny's constant. Combining the equations 1.8 and 1.9, Kozeny's equation can be represented by equation 1.10.

$$K = \frac{\varepsilon^3}{S^2 k (1-\varepsilon)^2} \qquad [1.10]$$

Theoretically, *k* should be 2 for an ideal filter cake. An ideal cake is a porous medium made of capillary tubes of radius r. Carman experimentally determined *k* as 5 for filter cakes made of monodispersed solids. However, for many industrial filter cakes formed in the presence of flocculants, *k* is often greater than 5 and it can be as large as several thousands (Gray, 1958).

In the above equations, flow through a porous medium is defined by assuming the porous medium as the filter cake for the filtration operations. Cake permeability is considered as the only resisting factor to the fluid flow. However, in filtration processes the resistance to flow is not only created by the filter cake but there is also a resistance applied by the filter medium. Although the filter medium resistance can be assumed negligible for many applications, it plays a very important role especially for the filtration of very fine size material. The filter media openings can be easily blocked when the particle size is very close to the pore size of the filter medium. Darcy's equation can be written based on this approach by including the filter medium resistance as in equation 1.11 (Cheremisinoff and Azbel, 1983),

$$\frac{dV}{dt} = \frac{A}{\eta} \frac{\Delta P}{(R_c + R_m)} \qquad [1.11]$$

where R_c is the cake resistance and R_m is the filter medium resistance. The filter cake resistance is related to the specific cake resistance α, as expressed in equation 1.12. The specific cake resistance is inversely related to the cake permeability and this relation is given by equation 1.13 (Tiller and Crump, 1977).

$$R_c = \rho_w \alpha C L \qquad [1.12]$$

$$\alpha = \frac{1}{K\rho_s(1-\varepsilon)} = \frac{kS^2}{\varepsilon^3}\frac{(1-\varepsilon)}{\rho_s} \qquad [1.13]$$

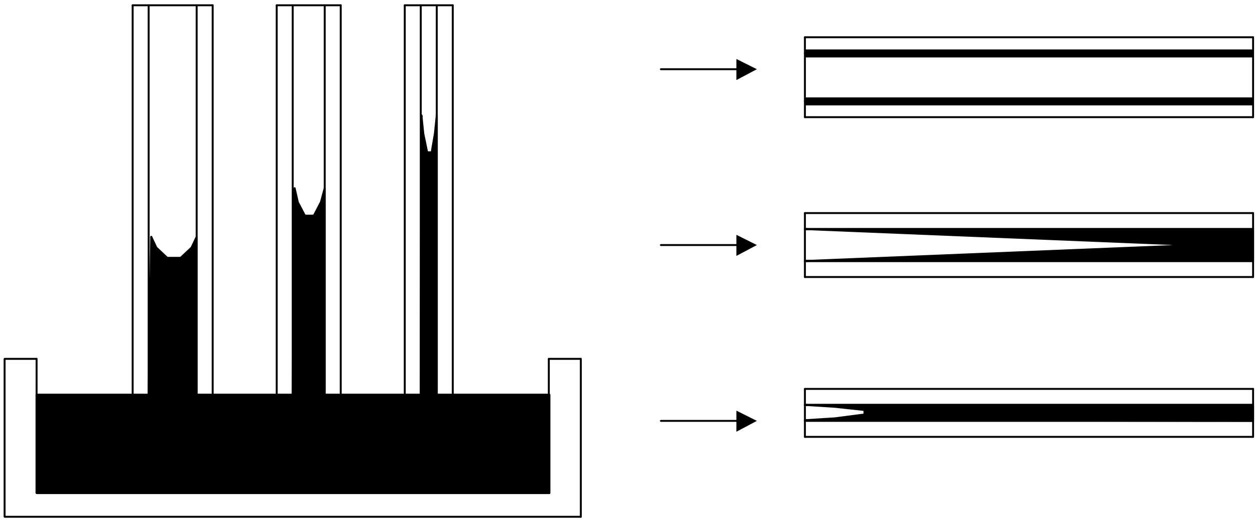
In these equations, ρ_s is the density of solid, ρ_w is the density of water (or draining fluid) and C is the concentration of solids in the slurry. The following relationship can be obtained for the rate of fluid flow by combining equations 1.11 and 1.12.

$$\frac{dV}{dt} = \frac{A}{\eta}\frac{\Delta P}{(\rho_w \alpha c L + R_m)} \qquad [1.14]$$

Equation 1.13 indicates that the specific cake resistance α, varies depending on the particle size, shape, and the cake’s particle size distribution. These factors effect the porosity and the specific surface area of the filter cake. Specific surface area increases with decreasing particle size. As a result, for filter cakes containing very fine particles, filter cake resistance is expected to be very high. The filter medium resistance also depends on the particle size, since the fine particles tend to block the pores of the filter medium and increase the resistance. In summary, the total cake resistance against the flow varies depending on the particle size distribution of the coal sample, the structure of the filter medium, the rate of deposition of the particles and the nature of the driving force.

As it is clear from equation 1.14, the flow rate through the filter cake increases as the total resistance and the viscosity of the fluid decrease and the applied pressure increases. For filter cakes composed of very fine particles, it is very difficult to reduce the resistance applied by the filter cake and the filter medium. To increase the filtration rate, either the viscosity of the fluid must be decreased or the applied pressure must be increased. The viscosity of the fluid can be decreased by increasing the slurry temperature. It is possible to heat the slurry by applying steam to the filter cake. This method was observed to be more economical than heating up the whole slurry (Gerl and Stahl, 1996). Surfactants can also be sprayed onto the filter cake as hot solutions. It was found that the filter cakes formed in the presence of the surfactant (a mixture of sodium alkyl sulfates) and heated with steam application at the same time, contained about 9 percentage points less moisture than unheated cakes formed in the absence of any surfactant (Baker, 1976).

Filtration is a three-phase flow operation where solid, liquid and gaseous phases exist all together. These phases interact with each other in various ways during the dewatering of the coal slurry. Figure 1.18-a shows the result of capillary forces in the capillary tubes with different radii under static conditions. According to the Laplace equation, the liquid rises higher in the smaller capillaries. However, if this bundle of tubes are placed in a flowing fluid, the movement of the fluid obeys Poiseuille's low as seen in Figure 1.18-b. The air, which is the drying fluid in the system, replaces the wetting fluid (water), under the effect of the applied pressure. For a given pressure gradient, there will be some size of tube in which the pressure gradient just matches capillary force and wetting fluid cannot be replaced for the capillaries of this size and the further smaller sizes (Brownell and Katz, 1947).



A. Capillary Tubes in Contact with a Liquid

B. Progressive Flow of a Second Fluid in Capillary Tubes

Figure 1.18. Effect of diameter on inhibition and two-phase flow in capillary tubes (Brownell and Katz, 1947).

According to the explanations given above, at a particular pressure gradient the cake saturation (the ratio of the volume of voids filled with the wetting fluid to the total volume of voids) can only be decreased down to a limiting value. Although the drying fluid is kept flowing through, that limiting saturation level cannot be reduced any further. This level of saturation is called the irreducible saturation. The irreducible saturation level of a cake varies depending on the properties of the fluid and the geometry of the voids in the filter cake. Dimensionless analysis predicts that the irreducible saturation is a function of the capillary number. The capillary number $N_{cap,}$ is defined as the ratio of the desaturating forces to the retention forces in dewatering (Wakeman, R. J., 1979). Equation 1.15 represents the relationship between the irreducible saturation S_∞, and the capillary number N_{cap}.

$$S_\infty = cons\tan txN_{cap}^{\alpha} \quad [1.15]$$

For the pressure drained beds,

$$N_{cap} = \frac{\varepsilon^3 d^2(\rho gL + \Delta P)}{(1-\varepsilon)^2 L\gamma} \qquad [1.16]$$

and for centrifuged cakes,

$$N_{cap} = \frac{\varepsilon^3 d^2 \rho N^2 r}{(1-\varepsilon)^2 \gamma} \qquad [1.17]$$

Irreducible saturation related to the N_{cap} can be given as in the following equations, based on the Kozeny-Carman permeability approximation

$$S_\infty = 0.155(1 + 0.03N_{cap}^{-0.49}), \quad N_{cap} \geq 10^{-4} \qquad [1.18]$$

for vacuum drained cakes, and by

$$S_\infty = 0.0524N_{cap}^{-0..19}\ , \quad 10^{-5} \leq N_{cap} \leq 0.14 \qquad [1.19]$$

$$S_\infty = 0.0139N_{cap}^{-0..86}\ , \quad 0.14 \leq N_{cap} \leq 10 \qquad [1.20]$$

for centrifuged cakes.

Filtration operations are usually limited to a pre-determined drying cycle time. This period of time depends on the operation of the selected filter type and the required throughput of the plant. As a result, the cake moisture cannot be reduced down to the irreducible saturation level in general. The amount of moisture remained in the filter cake at the end of the filtration cycle is defined as the residual saturation, S_R. The residual saturation of the filter cake can be given as a function of the initial saturation of the cake S, and the irreducible saturation as expressed in equation 1.21.

$$S_R = \frac{S - S_\infty}{1 - S_\infty} \quad [1.21]$$

Based on the knowledge of the filtration being a two-phase flow process through the filter cake, the permeability concept was further extended (Wakeman, 1978). A filter cake is relatively permeable to the drying fluid air, and the wetting fluid water. These relative permeabilities of water Kr_{liq} and air Kr_{air} are expressed by the equations 1.22 and 1.23 as a function of the residual saturation.

$$Kr_{lig} = \frac{\int_0^{S_R} \frac{dS_R}{[Pc(S_R)]^2}}{\int_0^1 \frac{dS_R}{[Pc(S_R)]^2}} \quad [1.22]$$

$$Kr_{air} = \frac{\int_{S_R}^{1} \frac{dS_R}{[Pc(S_R)]^2}}{\int_0^1 \frac{dS_R}{[Pc(S_R)]^2}} \quad [1.23]$$

where, Pc is the capillary pressure.

Finally, assuming that all the properties of a filter cake, such as porosity, saturation, permeability, etc., can be presented by an average number, the change in the moisture content of a filter cake X_w, at a capillary pressure of Pc,l at this moisture level can be given by the equation 1.24 (Gala, 1989).

$$\frac{dX_{lig}}{dt} = \frac{\rho_w}{\rho_s} \frac{KKr_{liq}(\Delta P - Pc,l)}{\eta L^2 (1 - \varepsilon)} \qquad [1.24]$$

1.2.2. **Dewatering Techniques**

The performance of the fine coal dewatering is closely related to the selected dewatering technique. The most common techniques used for fine coal dewatering are vacuum filtration and centrifugation. These techniques will be briefly explained here since these two methods were utilized to conduct the dewatering tests in this work.

1.2.2.1. Vacuum Filtration

Filters in the coal industry are assisted with vacuum to speed the passage of the material through the filtration medium. The filter cake is formed under the effect of the applied vacuum pressure on the filter media as was explained in detail in section 1.2.1.1. The industrial vacuum filters work continuously. To remove the formed cake at the end of the operation, the pumps creating the vacuum are reversed to help discharging the cake. This air blow cleans the filter medium and prepares it for another cycle. It is also common to use some mechanical devices to help removing the filter cake. Especially, fine coal filtration requires such a system, because it is highly possible to have the filter media blocked by the fine coal particles if they are not discharged properly. In summary, it is clear that the performance of the vacuum filters depend on the applied vacuum level and the efficiency of the filter media cleaning.

Four types of vacuum filters are used in the coal industry (Green, 1981). These are disc, drum, belt and pressure filters. The selection criteria vary on the basis of the required production rate, the required final moisture content of the filter cake, the available space in the plant and the investment budget of the company. The disc filters are widely used in coal industry since they cost less than the other types of vacuum filters. They have a large filter surface area compared to the volume they occupy. They can give higher throughput. However, for the disc filters the cake thickness can only be controlled by changing the percent solid content of the feed slurry or by adjusting the rotational speed. When the speed is increased both the cake thickness and the drying cycle time decreases or vice versa. These configurations are not favorable to reduce the cake moisture content.

The pressure filters have the disadvantage of being expensive and having high maintenance costs. The drum and the belt filters require both a large space in the plant and high initial investment but they give the advantage of arranging the cake thickness and the drying cycle time. For the drum filters, the depth of the slurry can be changed to adjust the cake thickness and the drying cycle time. The belt filters are advantageous in that the feed rate and the rotational speed can be changed independently to arrange the cake thickness and the drying cycle time. It can be concluded that the vacuum belt filters are the most suitable equipment for fine coal dewatering.

1.2.2.2. Centrifugation

Centrifuge is a device providing a mechanical acceleration field via centrifugal force (Warring, 1981). These forces separate the solid particles suspended in a fluid of lower density. The centrifugal action is similar to the gravitational settlement but a lot faster since centrifuges create 5000-8000 times greater acceleration than the gravity. They spin wet materials at high speeds and tear the water from the surfaces of the particles.

The body of the centrifuge is a drum or a cylinder. The centrifuged materials are forced against the inside of the spinning drum, where the difference in specific gravities separates the solids from the fluids. The stresses applied onto the solid particles are a combination of the stresses created by the mass of the rotating bowl and the stresses due to liquid pressure. The performance of a centrifuge depends on the bowl size and the rotational velocity. As the bowl size increases, both the stresses due to the bowl mass and the liquid pressure increase. However, the maintenance cost of the centrifuges is usually high because of the wearing due to these high stresses applied during the centrifugation operation.

The main types of centrifuges used for fine coal dewatering are solid bowl and screen bowl models. The solid-bowl centrifuge is a horizontal cylinder rotating fast enough to spin its contents outward against the inside surface. A scroll conveyor rotates inside the cylinder at a lower speed to push forward the solids accumulating against the inside of the cylinder (Green, 1981). Solid-bowl centrifuges work without screens or filter media and can recover solids down to 10-micron size. Figure 1.19 is the schematic representation of the industrial scale Bird countercurrent flow Solid Bowl Decanter Centrifuge. This equipment utilizes centrifugal force to separate a slurry into liquid and solid components. The separation takes place thousands of times faster than in a static clarifier. A stationary feed pipe feeds the slurry into the acceleration chamber where it is brought up to the same speed as the rotating assembly. Then the slurry is discharged out of the acceleration chamber into the separation section of the bowl. By spinning the bowl at a high speed, thousands of G's are exerted on the solid particles suspended within the slurry. This force pushes the particles outward from the centerline of rotation until they collect along the bowl wall. As the solids compact on the bowl, they displace the lighter liquid toward the axis of rotation.

The solid bowl is made up of a cylindrical section and a tapered section. The screw conveyor, which is located inside the bowl, turns at a slightly different speed from the bowl. The solids that have compacted along the bowl wall are advanced by the screw conveyor to the tapered end of the wall where they are discharged from the centrifuge. The differential speed between the bowl and conveyor is controlled by a planetary gear unit, or an automatic backdrive unit operation with or without the gear unit. This conveyor to bowl differential speed controls the rate of advancement of the solids within the centrifuge. The automatic backdrive senses conditions within the centrifuge and adjusts the differential speed to maintain optimum centrifuge performance. The clarified process liquid flows in the opposite direction from the solids and overflows adjustable weirs and is discharged from the centrifuge as a clear centrate.

Horizontal screen-bowl centrifuges are similar to the solid bowls in that they separate liquid and solid materials inside a cylinder containing a scroll conveyor. The difference is in the profile of the cylinder. Figure 1.20 represents the Bird Screen Bowl Centrifuge schematically. This centrifuge provides an effective moisture removal and excellent washing efficiencies. Operation of the screen bowl involves a solid bowl clarifier combined with a centrifugal filtration section. The solid bowl section of the equipment functions based on the same principles explained for the solid bowl centrifuge previously. The next stage involves a specially designed screen section that uses the centrifugal filtration principles. Additional moisture and solids are spun from the centrifuge in this section. The fully dewatered cake is then discharged at the exit end of the screen. Drainage from the screen section may contain valuable materials, which can be recycled back into the process stream, thus reducing product loss to an absolute minimum.

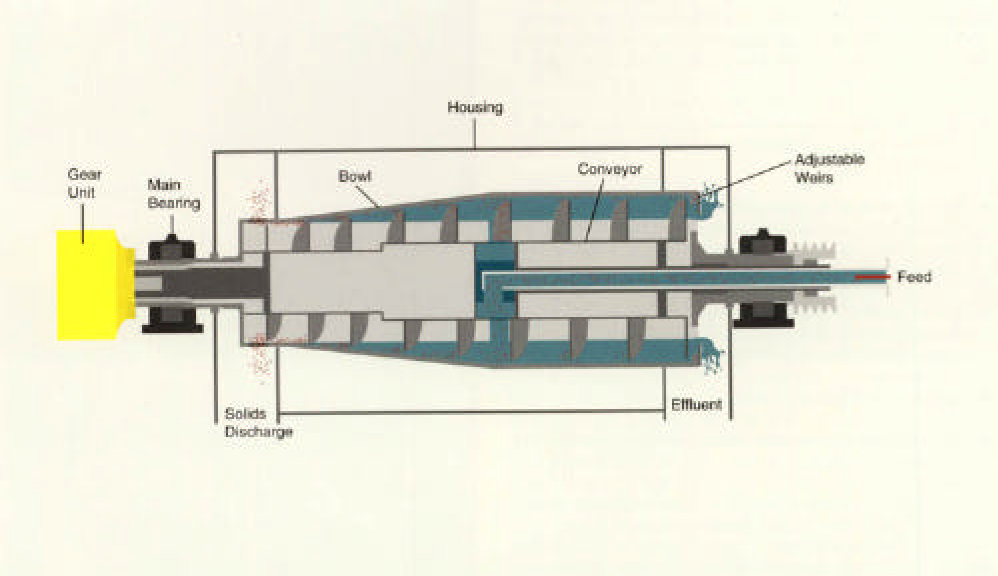


Figure 1.19. Bird countercurrent flow Solid Bowl Decanter Centrifuge.

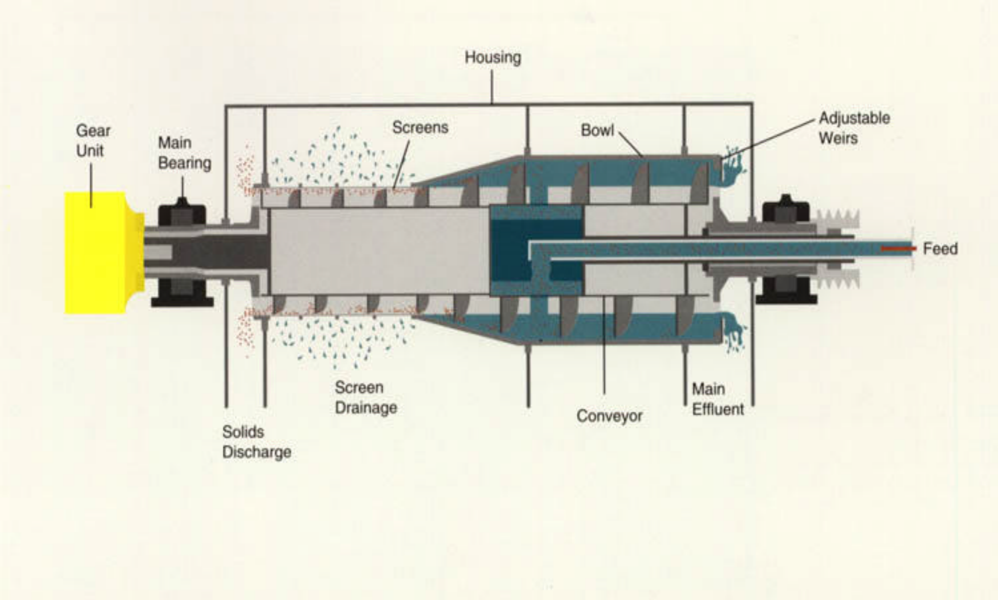


Figure 1.20. Bird Screen Bowl Centrifuge.

1.3. Objectives

The main objective of this investigation was to increase the efficiency of the mechanical dewatering methods in fine coal dewatering by using the novel reagents developed at Virginia Tech. The best performing reagents were selected by using laboratory vacuum filtration technique. The moisture contents of the fine coal slurries from different plants were decreased approximately by 50% in the presence of the novel additives. The second objective was to determine the degree of improvement that these reagents can make with different mechanical dewatering methods. The efficiencies of the selected reagents were tried on the centrifuge dewatering to achieve this objective. The last objective was to show the relationship between the change in surface chemistry of coal and moisture content reduction in the presence of these novel dewatering aids. This objective was achieved by comparing the contact angle and surface tension measurement results with the change in the moisture content of the coal slurries.

1.4. Organization

This thesis is arranged into individual chapters that are focusing on different specific objectives of this investigation. Chapter 1 contains a literature review on the characteristics of the coal and the moisture in coal and additionally the theory of the fine coal dewatering. Chapter 2 focuses on vacuum filtration of fine coal sample slurries in the presence of the novel dewatering reagents. Furthermore, a statistical analysis was represented in this chapter to determine the effects of various factors in moisture content reduction of fine coal filter cakes. Chapter 3 is about the centrifuge test results, conducted at Bird Machine Company’s laboratories. The comparison of the results with the vacuum filtration was also explained in this chapter. Chapter 4 presents the surface analyses on coal and relative changes in the moisture content of the filter cakes. Chapter 5 gives a general conclusion on the results of the tests conducted in this study and recommendations for the future work in this area.