COAL BENEFICIATION BY MEANS OF CONVENTIONAL FLOTATION TREATING THE SIZE RANGE OF + 0.5 MM – 1.4MM

GABRIëL JOHANNES PRETORIUS

A research report submitted to the Faculty of Engineering and the Build Environment, University of Witwatersrand, Johannesburg, in partial fulfilment of the requirements for the degree of Master of Science in Engineering.

Johannesburg, South Africa 2010

DECLARATION

I declare that this research report is my own unaided work. It is being submitted to the degree of Master of Science to the University of Witwatersrand, Johannesburg. It has not been submitted before for any degree or examination to any other University.

Signed : ______________________________

Gabriël Johannes Pretorius

Date : _____ day of __________________ 2011

ABSTRACT

Tshikondeni Coal has utilized froth flotation to treat the fine fraction of the feed to plant since the early 1980's. The latest plant in use was commissioned in 1996 treating the – 1.4mm material in Wemco Smart Flotation Cells. Various problems were experienced floating the + 0.5mm material which initiated improvement projects over a period of 13 years to improve the recovery of this fraction.

New conventional froth flotation technology was developed by a company named Ultimate Flotation for the copper industry. Great successes have been achieved recovering copper from slurry ponds in various countries in Africa.

The Ultimate flotation technology was tested in the Tshikondeni flotation plant and proved to be a viable option as replacement flotation cells. The main aim of the project was to replace the first three flotation cells in each module with Ultimate flotation cells and to increase the percentage recovery of the + 0.5mm – 1.4mm size range with at least 5%. The recovery increase achieved in this fraction after installation amounts to 8.1%, with a saving in energy consumption of 47%. It is recommended that the fourth flotation cell be replaced with an Ultimate Flotation Cell. Cell number five and six can be removed from the circuit.

DEDICATION

The work done and the many hours spend to complete this research report is dedicated to my wife, Carina, and my two children, Danelle and Estiaan, for their support and numerous sacrifices they had to make to allow my further studies.

ACKNOWLEDGEMENTS

- I would like to give special thanks to the Lord, our Creator, for blessing me with the talent and ability to work and study hard enabling me to become a better person overall.
- Thanks to my mentor, Mr Fanie Boshoff, for developing me from an artisan to a Production Manager with an Advanced Coal Preparation certificate.
- To Mr David Powell, Plant Manager at Tshikondeni, I would like to give special thanks for introducing me to the GDE programme as well as for his assistance in completing the course. Thank you for being a dear friend who believed in my abilities throughout.
- To Mr Stephan Badenhorst, Mine Manager at Tshikondeni, thank you for your assistance in upgrading the Tshikondeni flotation plant as well as supporting my judgement in opting for the Ultimate Flotation System.
- To Dr Adrie Conradie, Projects Manager Exxaro, special thanks for believing in me and making further studies possible, even in a challenging economic environment.
- To Prof Rosemary Falcon, thanks for all your assistance during the past few years and for helping me to complete the GDE course which enabled me to enrol for the Msc.

TABLE OF CONTENTS PAGE

LIST OF FIGURES PAGE

LIST OF TABLES PAGE

LIST OF ATTACHMENTS PAGE

NOMENCLATURE

Cubic Meter per Hour	m^3/h
Contact Angle	θ
Degree	°
Degree Celsius	°C
Ecart Probable (moyen)	epm
Hertz	Hz
Iron	Fe
Kilowatts :	kW
Meter (meters)	m
Millilitres per minute	ml/min
Millimetre :	mm
Micron	µ
Percentage	%
Revolutions per minute	rpm
Tons per hour	t/h
Work of Spreading	γ_s
Liquid to air bubble interface surface energy	γ_{1v}
Solid to air bubble interface surface energy	γ_{sv}
Solid to liquid interface surface energy	γ_{s1}
Hidrophile-lipophile balance	HLB

CHAPTER 1: Introduction

1.1 Geological Background

Tshikondeni is situated in the far North East sector of South Africa and is currently the only mine extracting coal from the Soutpansberg coal reserve. The mine borders the Kruger National Park, which can be described as a very sensitive environmental area. Figure 1.1 indicates the regional location of Tshikondeni mine which is being managed by Exxaro on behalf of Arcelor Mittal Vanderbijlpark.

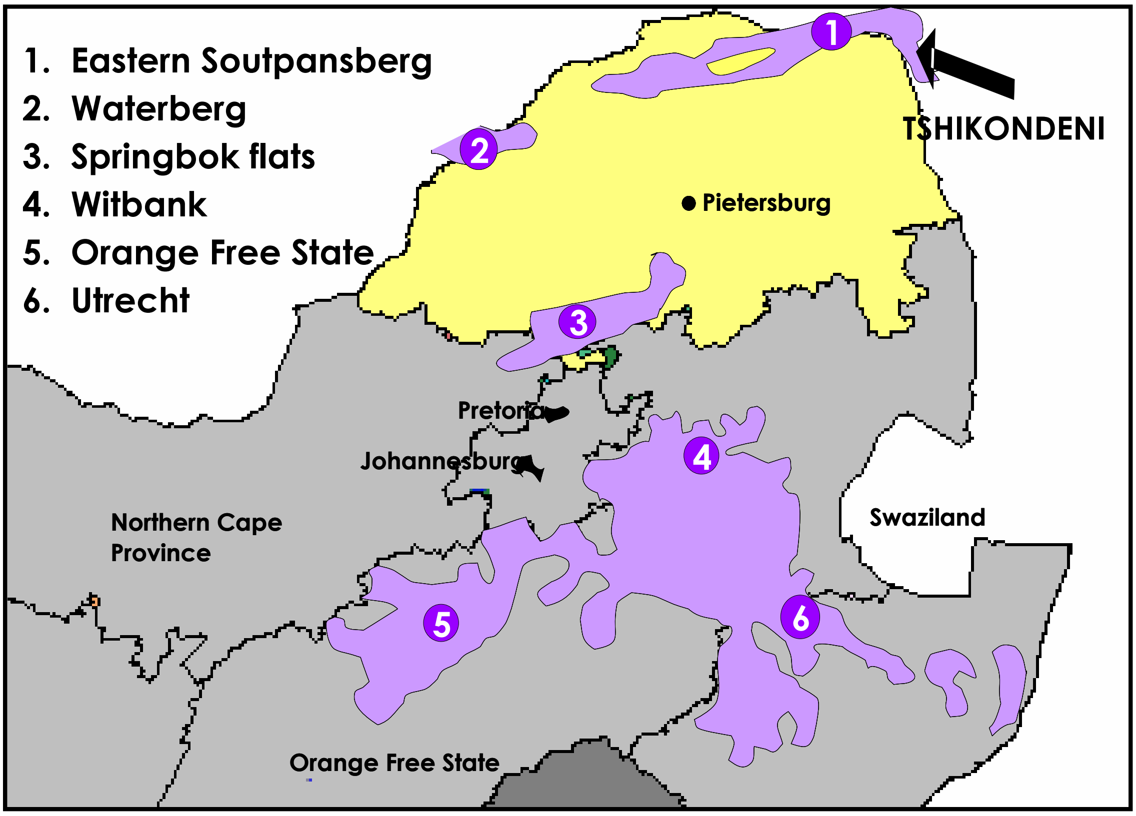

Figure 1.1: Regional location of Tshikondeni Coal

The coal being mined is of high vitrinite content and is commonly known as “hard coking coal”. The value of coking coal is significantly higher than steam coal. It is therefore important to optimize the plant yield and recover the maximum percentage from coal mined at very high expenses due to difficult geological conditions. Vitrinite rich coal is also brittle and the plant design must cater to treat a high percentage of fine coal. The impact of the formation of the Soutpansberg coal field on the quality of the coal is very important to understand and will be discussed briefly.

Tectonic activities were present during various geological periods in the Tshikondeni Coal Mine area. The coal seam overlies the Pretorezoic Limpopo Mobile Belt consisting of extensive east-north-east trending linear zones of high-grade metamorphic tectonites. The Kaapvaal Kraton was down-faulted on the North-Eastern margin into a graben-type structure into which the pre-Karoo Soutpansberg Group was deposited. During the deposition of the Karoo sediments this faulting continued and was again reactivated in the post Karoo times, resulting in a very complex structural setting. Two main fault systems have been identified, one North-North-West trending and the other East-North-East. The main faults encountered in the Tshikondeni coal field caused numerous smaller faults in the area varying between 0.5m and 10m. These faults are being encountered in the coal seams being mined and can be in an upward or downward direction causing some major production and planning disruptions.

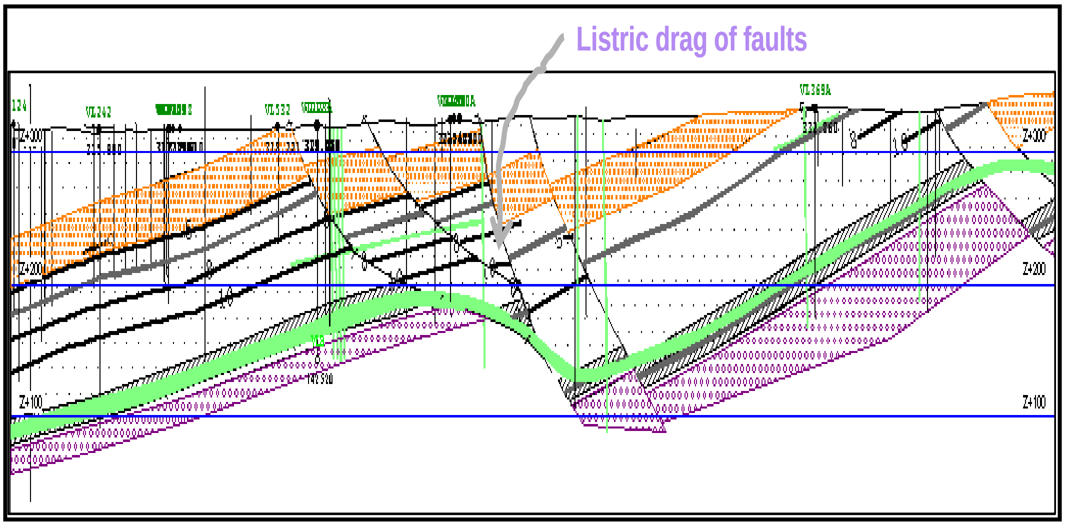

Figure 1.2: Listric drag of faults

The Karoo series was faulted into a series of horst and graben blocks with displacements exceeding 1000m in some cases. The faults are normal faults and have a listric nature, which is an indication of extensional tectonics as can be seen in figure 1.2. The blocks between faults are tilted and the dip of the strata varies between 2° and 18°, but increases to 22° near the faults. The mine operates from surface to a maximum depth of 400 meters below surface due to the dip of the coal seam. The coal seams were deposited similarly to those of the Vryheid formation and are locally known as the Madzaringwe formation.

It is a challenging coal mine as far as mining conditions are concerned due to the extensive faulting and intensity of dolerite dykes and sills in the area. Small scale faults of less than two meters are common in the production areas of the mine. Dolerites occur as sills and dykes where the dykes are up to 20m wide and the sills 30m thick. Contact aureoles vary, but can reach up to 60m on either side of the dykes, but the average influence is about the thickness of the dyke. The coal in this area is usually heat affected with drastic changes to the surface properties and volatile content. The heat affected coal is very difficult to beneficiate in flotation cells as the surface properties are similar to a coal which has been oxidised. The rocks found in the coal seam are fine to crystalline and green to dark green and gray. Yellowish chill, flow banding and flow laminations are common. Some dykes have spinifex textures. Brecciation and fracturing at the contacts are common, as are slickenside's associated with small scale faulting (< 2 meters) (Viljoen, H. *pers comm 2009).*

The major dykes in the Tshikondeni coal reserve can clearly be seen in figure 1.3. These dykes vary in thickness between 5m and 30m. The dykes were observed by making use of Geo Magnetic technology which indicates mostly the dykes thicker than 5m. There are however; numerous dykes and sills less than 5m which could not be detected by making use of the Geo Magnetic technology. It is of utmost importance to make use of horizontal drilling in the sections to ensure that the area to be mined is being explored prior to development. The horizontal drill will in most cases indicate the presence of dyke and sill intrusions. Up throw and down throw faults are usually present in close proximity of dykes and sills and will also be indicated by the horizontal drill. Horizontal drill information forms a critical part in the mine development planning process.

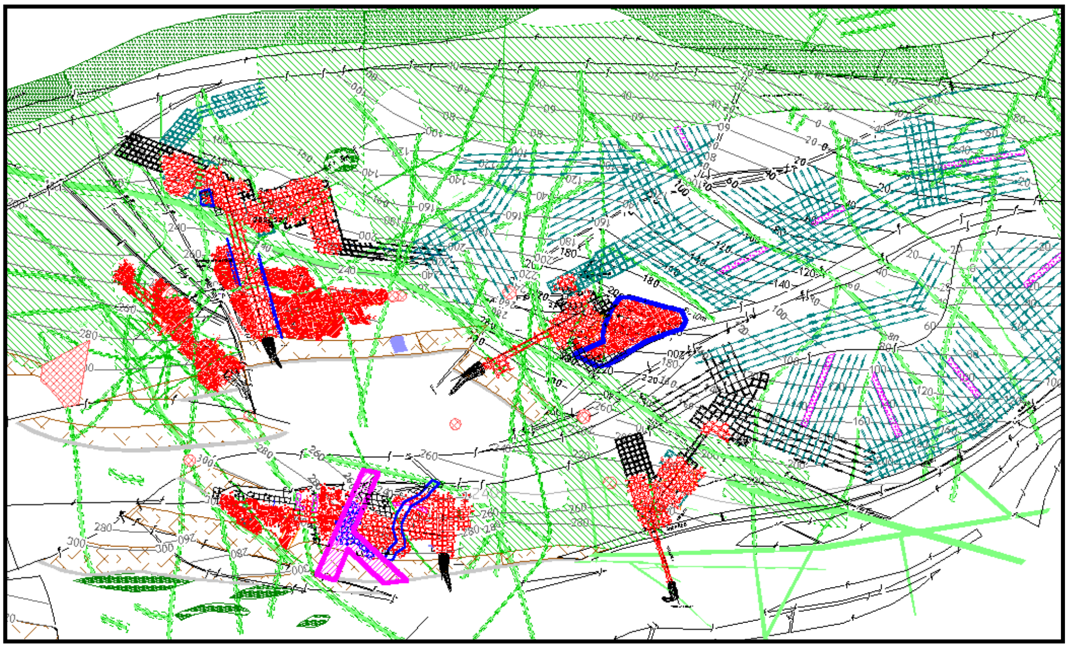

Figure 1.3: Dyke and Sill intrusions in Tshikondeni coal reserve.

The conditions during the formation of coal determine in most cases the maceral composition of a coal reserve. South African coal consists mainly of inertenite rich coal which is high in mineral content. Fern like plant material that grew in great swamps and low-lying terrain, as was the case in the Northern hemisphere, were mostly fast growing plants helped by warm and moist weather. As the plants died, they accumulated below the swamp water surface where the decaying process continued in the absence of oxygen (Falcon, R.M.S. 1986). The decaying plant materials passed through a gel like phase after which they metamorphosed into shiny black constituents of coal. The shiny black constituents are known as vitrinite. The coal in the Northern hemisphere is low in mineral content as limited minerals were washed into the swamps by the rivers of the time.

Figure 1.4 is a typical illustration of the formation of coal in the Northern hemisphere. The plant material is accumulated in the swamps and is eventually buried under soil. In the presence of pressure and heat over an extended period of time coal is formed. The amount of pressure, time and heat determine the degree of maturity of the coal. (Falcon, R.M.S, 1986)

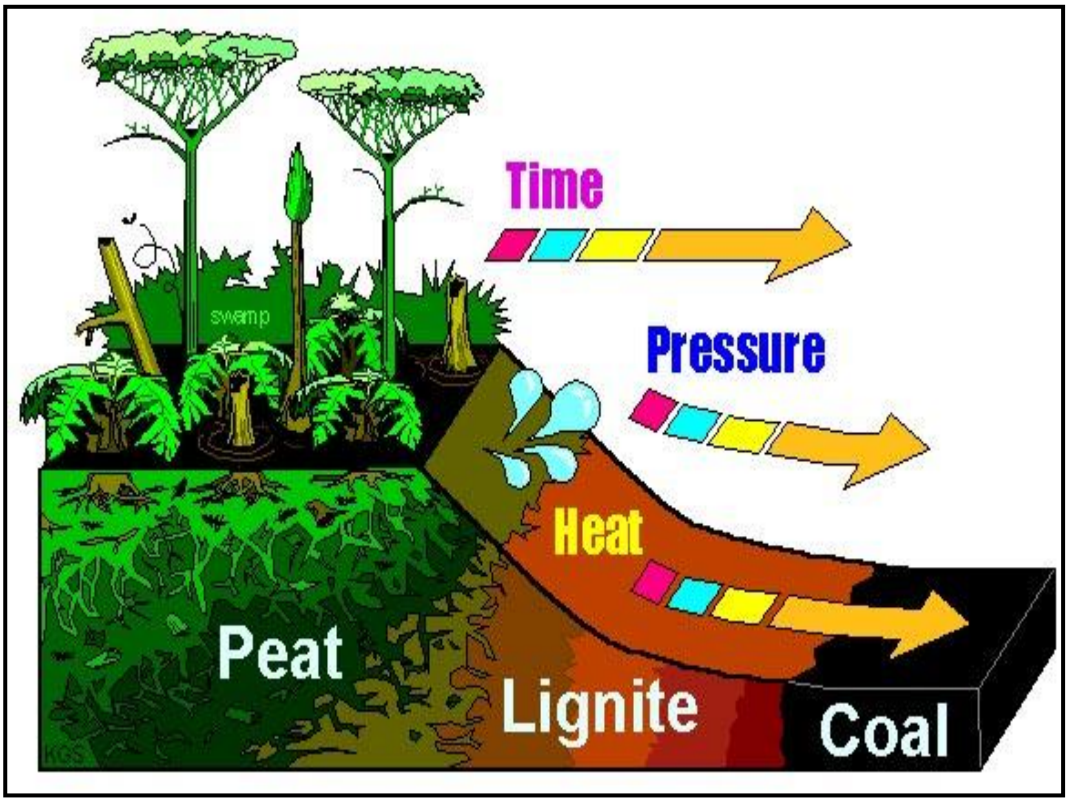

Figure 1.4: Coal formation. (Mining and environmental geology. INTERNET. http://www.geologydata.blogspot.com/)

In contrast to the coal formation conditions of the Northern hemisphere, the conditions in the Southern hemisphere were dry and cold (Falcon, R.M.S, 1986). Plants grew in shallow lakes and due to the cold and dry climate the type of plants were slow growing. The lakes were fed by rivers as a result of molten ice and glaciers during the relatively short summers. Plant material accumulated in the lakes was deciduous causing smaller amounts of plant material. This allowed mineral matter to be carried into the plant material by the rivers.

The origin of the mineral matter was in the mountains where the melting glaciers followed a path to the rivers eroding rocks and soil on its way down from the mountains. This phenomenon is mainly the reason for the high percentage mineral matter found in the South African coal. The plant material was only covered by water during certain periods in any specific season by the lakes and therefore plant material was mostly exposed to aerobic decay causing South African coal to have lower proportions of vitrinite and liptonite, and much higher proportions of inertenite and mineral matter.

The Tshikondeni coal reserve situated in the Soutpansberg Coalfield is however unique. The coal reserve consists of vitrinite in excess of 80% indicating that the coal forming plant material was mainly covered in water creating anaerobic decaying conditions. The mineral percentage found in the Tshikondeni coal is however similar to the mineral content found in most of South Africa's medium to high grade coal. The product produced at Tshikondeni forms part of a mixture consisting of imported hard coking coal as well as a local semisoft coking coal which is used for the production of metallurgical coke. The coke is mainly utilized in the iron and steel manufacturing industry. Some of the Tshikondeni product is also utilized to produce market coke for the Ferro Alloy industry.

Figure 1.5 depicts a typical coke battery used for the production of coke. The minerals (ash) in the coal are not removed during the coke production process and will therefore be transferred into the blast furnace. The ash in the coke will have a negative effect in the blast furnace reducing the overall iron (Fe) yield. It is therefore critical to beneficiate the raw coal as mined to yield a high grade hard coking coal product with low ash content.

The hard coking coal price must also be competitive with coal imported from Australia and New Zealand. Mining conditions in the Soutpansberg Coalfield are not ideal and can be described as one of the most difficult and expensive areas to mine. It is therefore critical to maximize the recovery of product from the raw coal as mined to ensure that operating costs do not escalate to a level where the product is more expensive than coal imported from abroad.

Figure 1.5: Coke Battery in Wales (INTERNET. http://www. en.wikipedia.org/wiki/file:coke_ovens_Abercwmboi)

Low ash, low sulphur, vitrinite rich coal is used to produce coke which is a solid carbonaceous residue. The volatile constituents are driven off by heating the coal in the absence of oxygen. The temperature is ramped up to a maximum of 1 000 °C and then gradually ramped down again. It takes approximately 18 hours to produce one battery of metallurgical coke. The fixed carbon and the ash in the coal is fused together to form coke. A high quality coking coal will swell at least nine times its original size to allow the volatiles and fluids in the coal to be driven out by the heat. The coke obtained from coal is gray, hard and porous with a heating value of 29.6 MJ/kg.

Metallurgical coke is loaded in layers with iron ore into the blast furnace and act as a fuel as well as a reducing agent assisting in smelting the iron ore. The coke must be strong enough to carry the weight of the load in the blast furnace. A weak coke will collapse causing the layers of iron ore to come into contact with each other. This can be quite problematic and the outcome can be catastrophic with the furnace ending up as a solid heap of consolidated iron. Some by-products such as tar, ammonia, oils and gas can also be derived from the coke making process. Coking coal forms a critical part in the conventional iron melting methods utilized successfully over many years and will therefore be in demand for many more years to come.

1.2 Beneficiation Plant Background

The initial pilot plant built at Tshikondeni utilised a Rectangular Flotation Tank circuit to treat the fine fraction (- 1.4mm material). This fraction amounts to 30% of the total feed to plant. The Bath Flotation system consisted of a rectangular bath fitted with six flotation drive units. The fine coal entered from the one side of the rectangular tank passing over the false bottoms fitted in the tank. The pulp was lifted through the false bottoms by the draught created by the rotors of each flotation drive. The rotor then dispersed the fine particles through the disperser hoods creating air bubbles in the bubble to particle attachment zone with the aid of a chemical frother. The coal product captured in the flotation froth rose to the coal slurry surface where it was removed from the rectangular tank by means of mechanical paddles. The tailings flowed out of the tank through a drainage system fitted with a dart valve. The dart valve was hydraulically controlled to ensure that a constant coal slurry level is maintained. The Bath Flotation system was quite efficient, but was old technology even in the mid 1980's to early 1990's. The product was dewatered on a horizontal belt type filter system which worked quite well when maintained properly. The fine coal product was then mixed with the product obtained from the coarse Dense Medium Separation (DMS) cyclone.

The new plant was designed during the early 1990's and was commissioned at the end of 1996. No primary crusher was included in the run of mine circuit. The + 200mm material are removed by a scalping screen and then stockpiled to be crushed at a later stage. Crushing is achieved utilizing an American Granulator reducing the size from – 200mm to – 13mm. The – 13mm material in the feed is removed prior to the Granulator by means of a sizing screen. The liberation of the coal from gangue material prior to beneficiation is critical as the unwanted minerals are finely imbedded in the coal. The new beneficiation plant design consists of two mirror image modules. Each module is equipped with a 710mm dense medium cyclone treating the – 13mm + 1.4mm fraction. The bath type flotation system was replaced by six Wemco Flotation Smart Cells per module.

The new Wemco Flotation cells were used to treat the – 1.4mm + 0mm fraction. Figure 1.6 depicts the workings of a Wemco Flotation Smart Cell.

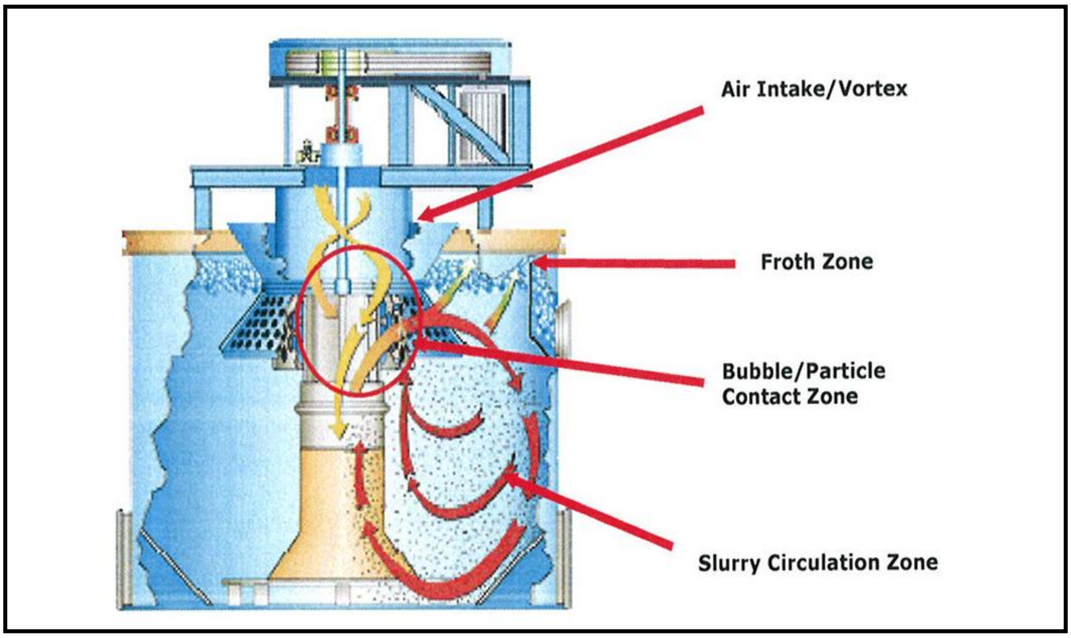

Figure 1.6: Wemco Flotation Smart Cell (Wilkes, KD)

1.2.1 Problem statement

The original rotors were designed with six blades. The expectation was that it would be able to lift the 1.4mm material from the false bottom and through the draught tube to be dispersed into the bubble to particle attachment zone. Major problems were however experienced and the – 1.4mm + 0.5mm material proved to be problematic to float. After some test work and collaboration with the company's research and development department, the rotors were replaced with eight blade rotors in the attempt to increase the air intake into the flotation cell. This modification was meant to increase the power of the draught through the draught tube. The attempt to float the coarse fraction of – 1.4mm was in the end in some way achieved, but some disadvantages were however created in the process:

- The 15kW electrical drive motors of cell number one and cell number two tripped on over current once the feed tonnage to the plant was increased closer to design capacity.
- The surface area of the water in the flotation cells were disrupted by the currents caused in the cells due to the increased energy injected by the eight blade rotors.
- Excessive wear to the flotation cell bodies as well as to the internal components were experienced due to the increased velocity of the particles.

There was one solution identified which addressed only one of the mentioned problems. The 15Kw electrical motors were replaced by 18kW motors which solved the problem of the electrical over current problems experienced with the 15kW motors. Both the other problems could not be solved without sacrificing clean coal lost to the tailings.

Another design problem experienced can clearly be seen in figure 1.7. Cell number one, two and three in each module were installed on the same level. Cell number four, five and six were installed on the same level although on a different level than cell one, two and three. The level of the first three cells was controlled with a hydraulic assisted dart valve and the second three were controlled on the same manner at cell number six. Level control turned out to be a major challenge as all three cells being controlled was build on the same level. When the level in cell number three is adjusted to 75mm below the lip of the cell, the water in cell number one and cell number two overflowed into the product launder. By adjusting the level in cell number one to 75mm below the lip, the levels in cell number two and cell number three were too low, therefore causing the product bearing froth not to flow into the product launder. This problem proves to be problematic where quality control is concerned.

Figure 1.7: Wemco Flotation cells installed in 1996

A simple solution was implemented to mitigate the level problems experienced without redesign or further capital expenditures.

The levels were increased by installing 400mm wide stainless steel rings wrapped around the flotation cell lip area. It was only necessary to install rings to cell number one, two, four and five. The level of cell number one and four were raised 100mm and cell number two and five were raised 50mm. The outcome was that the levels could be managed most of the time. No linear control was however possible with the hydraulic system installed. The dart valve installed to control the level was either in the open or closed position. The net effect was a level that was constantly varying at least 50mm. This caused the flow of froth to stop at least once every minute during operation.

In an attempt to recover more of the + 1mm material still present in the flotation tailings, spiral concentrators were installed in the late 1990's. The flotation plant tailings were pumped to the spiral plant and deslimed in two 350mm classifying cyclones prior to being dewatered on a high frequency deslime screen. The product from the spirals was routed to the flotation plant product and dewatered in a Humboldt solid bowl centrifuge. The spiral plant could however never be used on a permanent basis. The product specification at the time was 12% ash content and the spirals were not able to produce a product with ash content lower than 17%. The high ash obtained from the spirals increased the product ash to levels unacceptable for the client. The only other way to use the spirals in conjunction with the rest of the plant was to adjust the relative density of the dense medium cyclone to produce a lower ash product. The yield loss was greater than the yield gained in the spiral plant and therefore the decision was made early in 2004 to decommission the spiral plant altogether. The spiral plant was however not demolished to be put back into operation in the future if the need arises to do so.

The flotation cells installed in the new plant were utilized from 1996 to 2001. In 2001 it became evident that the flotation cell tanks had to be replaced due to high wear caused by the increased particle velocities. At the time the opportunity was not used to redesign the flotation circuit. The damaged flotation cells were replaced by identical Wemco Smart Cells. The tank liners were replaced by a 5mm polyurethane liner in the attempt to increase the lifespan of the tanks. None of the original design problems were addressed and production continued as before.

Tshikondeni coal was always easy to beneficiate in a flotation cell due to the high vitrinite content of the coal. An example of a typical monthly Petrographic report is attached as Attachment A. One can clearly see that the coal is a high grade coking coal by observing the vitrinite content of 80.6% as well as the Roga Index being in excess of 85. Major problems were experienced in cell number one and cell number two with the flowability of the froth over the cell lip into the product discharge launder.

The froth became so dense and saturated with product that it started to dry out on the surface of the water in the cell. Bubble size also contributed to this phenomenon caused by the lack of air induced into the flotation cell. The froth was removed from the flotation cell by spraying the froth with water. The net effect was that the product caught in the froth was lost back into the flotation cell. This procedure had to be repeated once every hour to ensure that the product filled froth is able to pass over the flotation cell lip.

The plant organic efficiency at this point in time averaged 92.5% on a month to month basis against a target of 92.5%. During 2005 a project was launched to increase the plant organic efficiency to 94%. The outcome of the project was to install mechanical paddles to assist with the removal of the froth from the flotation cells. The paddles were driven by an electric motor and a reduction gearbox. Three 90 degree gearboxes transferred the energy from one paddle to the next in the attempt to remove the froth around the edge of the entire cell. The paddles proved to add value and the plant organic efficiency increased from 92.5% to 94% immediately after installation.

Maintenance costs on the paddles were however quite costly and availability deteriorated after the first six months of use. Daily maintenance was needed at quite a cost. The 90° gearboxes installed had to allow the paddles to cover the entire lip area of the cylindrical tank and worked very well in the beginning. The reagents used to assist with the flotation process caused damage to the seals on the gearboxes leading to premature gearbox failures. The availability of the gearboxes was problematic and the paddles had to be removed to ensure that there is no restriction preventing the flow of the froth over the lip of the flotation cell while waiting for spares.

1.3 Project scope

Tshikondeni plant management were contacted by a company named Ultimate Flotation late in 2004 introducing a new concept rotor and disperser. Various successes have been achieved in the copper industry utilizing the new technology. A rotor and disperser was obtained from the company to be tested in the Tshikondeni plant. Regrettably the rotor and disperser were available for only one day at the time. The rotor and disperser were destined for another plant and could only be side-tracked for three days. One day was used for installation and another day for disassembly which left one day for some test work. The impact of the rotor installed in the first flotation cell of Module A was noticeable right from start-up. The air induced into the cell could be seen immediately after the rotor came into contact with the water in the cell.

The pulleys on the electric motor and the rotor bearing unit were replaced to reduce the speed of the rotor. The froth generated by the rotor and the increased airflow into the cell caused the froth to flow freely over the lip of the cell. No caking of the froth was noticed during the test day and it was not even necessary to wash the froth from the cell as was the case with the other cells. Although the froth was dense with fine coal product, no coarser material could be detected in the froth. There was unfortunately not another rotor and disperser available at the time to be installed in the second cell, which would have given a clearer indication of the ability of the rotor to lift the coarse material as well. The overall feeling was that the rotor and disperser was unable to float the – 1.4mm + 0.5mm material. It was however quite clear that the froth in the first cell fitted with the new rotor was saturated with – 0.5mm material leaving no space for the coarser material to attach to the bubbles. The one day test work was not enough to persuade the relevant people to purchase a rotor and disperser for further test work.

The second set of flotation cells installed in 2001 was nearing the end of their useful life in 2008. The flotation cells could not be replaced with another piece of equipment with similar capital cost to beneficiate the – 0.5mm material. The flotation process is also ideal for beneficiating this size fraction, especially with a feed consisting of more than 80% vitrinite. The – 1.4mm + 0.5mm material can be successfully treated with dense medium cyclones. Extensive test work indicates that the best efficiencies can be achieved beneficiating – 1.4mm + 0.5mm material by means of dense medium cyclones (Van der Merwe, D 2007).

The capital cost to build a fines dense medium plant treating the – 1.4mm + 0.5mm material at 100t/h was approximately R 20 000 000 in 2008. Such a project is not viable as Tshikondeni has only 5 years left on life of mine extracting coal from the current reserve. Even when a dense medium plant is installed to treat the + 0.5mm material, the flotation cells still need to be replaced to treat the - 0.5mm material, which will increase the capital cost even further. The decision was made to replace the first three flotation cells on each module with Ultimate Flotation Cells. These cells will also be utilized in the future if additional reserves are found to extend the life of mine. The flotation circuit can be reduced to three cells per module if the fines dense medium plant becomes a viable option to treat the + 0.5mm material. These cells will be tested and optimized by adjusting the various parameters provided by the new technology until the optimum recovery is achieved.

There was approximately 5% + 0.5mm product present in the flotation plant tailings which can be recovered. The opportunity existed to increase the product ash by 1 percent from 11 to 12 percent. This will have a positive effect on the flotation plant yield as well as on the overall plant yield. The main aim of the project is therefore to replace the first three flotation cells in each module with Ultimate flotation cells and to increase the + 0.5mm recovery by at least 5%. Another objective is to decrease the energy consumption in the new flotation plant by 10%.

CHAPTER 2: Literature survey

This chapter covers the history of conventional flotation and the challenges one can experience relating to conventional froth flotation performance. The chapter will also investigate coal characteristics and the impact thereof on conventional froth flotation.

2.1 Coal characteristics

Froth flotation is highly dependent on the differences in surface properties of coal and the minerals present in the feed to the flotation cell circuit. Coal, which is mostly composed of hydrocarbons, is naturally hydrophobic and will therefore repel water. Hydrophobic coal particles attach themselves to the air bubbles in a flotation cell to be carried to the surface and discharged into the product launder. The mineral component is generally hydrophilic and will be coated by water ensuring that no particle to bubble attachment of the minerals will be possible in the flotation environment when beneficiating coal.

The natural hydrophobicity of coal can however be influenced by several factors and will be briefly described.

- Coal Rank:

  The order of flotation for high rank coal is as follow in descending order: vitrinite, inertenite, exinite. It was found by Sarker *et al* (1984) that vitrinite is the most floatable maceral in the medium rank coal. Macerals can therefore be recovered by froth flotation provided that high levels of liberation are achieved.

- Surface functional groups:

  Surface functional groups are generally hydrophilic and the presence thereof on the coal surface will affect the floatability of the coal negatively (Beafore, 1979). Oxidation of coal will increase the presence of surface functional groups causing the coal to be less floatable (Sun, 1954).

- Oxidation of coal surface:

  Coal stored for long periods of time, or that has been mined close to the surface, will undergo physical and chemical changes due to the effect known as weathering (Beafore *et al*, 1979, Sun, 1954).

  Oxidised coal flotation is influenced by the following

  - Higher brittleness of oxidised coal which will cause an increase in the amount of fine coal being generated.
  - An increase in fines will develop a higher surface area.
  - Pulp PH will decrease.
  - Free cations in the coal pulp.
  - Surface hydrophobicity will decrease.

  (Beafore *et al*, 1979, Fuerstenau *et al*, 1987)

- Slime coating and entrainment:

  Ultra fine particles that remain in suspension can be defined as slime. When slime is present in the flotation cell, excessive use of reagents will occur together with a decrease in product recovery. Clay surfaces have a dual charge could bind to the negatively charged coal surface. This phenomenon will render the surface of the coal particle hydrophilic hampering the floatability of the coal.

Entrainment will occur in a flotation cell in the presence of ultra fine hydrophilic material. This material will not attach to the air bubbles, but will be transported to the froth bed on the water surface by the capillary forces caused by the air bubble movement. This material can be trapped in the froth contaminating the coal product.

- Particle size distribution:

  Sun and Zimmerman (1950) established that coarser particles require more than one air bubble to rise to the surface. Cell turbulence caused by the spinning rotor in a conventional flotation cell will also affect the floatability of larger particles negatively. Floatable particle size limits have to be considered according to Wheeler and Keys (1986) as a function of the coal hydrophobicity as well as the rank of the coal.

- Pulp temperature:

  It was found that the floatability of four British coals increased at pulp temperatures between 25°C and 30°C by Bailey (1953). The floatability decreased at pulp temperatures below 25°C but no influence was noted at pulp temperatures above 30°C.

- Coal pulp conditioning:

  Coal surface properties mostly dictate the conditioning of coals using oily collectors. The spreading of an oily film on a coal surface from a thermodynamic point of view requires an energy input termed the work of spreading, $\gamma_{s.}$ Oils spread more easily over high rank coal than over low rank coal and require therefore a lower γ_s Powell (1999). Aromatic oils containing surfactant impurities generally spread the most readily indicating that the structure and composition of the oil plays a role. Enhanced oil spreading does however not necessarily improve flotation recoveries.

As described it is clear that coal characteristics play a major role in the floatability of coal particles and that a highly floatable coal can be influenced by external factors affecting the floatability thereof negatively. Most South African coals are high in mineral content rendering the floatation process to become quite difficult. Grade control can become quite problematic as the minerals are finely interwoven in the coal. When floating larger particle sizes than 0.5mm, it can be expected that the product quality will deteriorate and even go beyond product quality specifications. The floatability of the coal changes quite drastically in the case where the coal has been affected by heat introduced by either dyke or sill intrusions. The coal is commonly known as burnt coal or de-volatilized coal. When treating Tshikondeni's vitrinite rich burned coal (heat affected) in a flotation cell, the frother addition needs to be stopped as the coal generates a froth bed in the flotation cell that cannot be controlled when a frother is being added to the conditioning tank. Quality control is quite problematic even without the addition of a frother and the addition of a collector need to be reduced to levels half of the normal collector addition.

2.2 Conventional flotation history

With the main aim being to commercialize concepts in pneumatic froth flotation, Maelgwyn Mineral Services was founded in Llandudno, North Wales on 21 October 1997. It is interesting to note that the Elmore brothers were installing the world's first industrial size commercial flotation process for mineral beneficiation at the Glasdir copper mine exactly one hundred years prior to the foundation of Maelgwyn Mineral Services. Even though there was only one known flotation plant treating fine coal in Europe in 2009, the Welsh can proudly say that the birthplace of flotation was in Wales in 1897. Approximately 80% of all the worlds' minerals are currently being beneficiated utilizing the flotation process (Jenkins, P.R. 1996).

Figure 2.1: Glasdir copper mine (Jenkins, P.R. 1996)

Figure 2.1 illustrates the Glasdir copper mill and flotation plant bought by the Elmore brothers, Frank and Stanley, in partnership with their father William Elmore. The Glasdir copper mine was in production since 1852. The ore was very difficult to work and recovery was inadequate to ensure a decent operating profit. The mine was constantly in and out of liquidation causing a constant change of investors. The Elmore brothers invested a considerable amount of capital to build a new mill for the conventional concentration of copper ores. There was however no improvement in recovery and experiments were then conducted by Frank Elmore. His idea of using oil as an instrument of beneficiation paid off and the mine was shortly exploiting this selective action on a commercial scale.

The process used is known as the world's first practical flotation process and collected 70% of the copper present in the ore being mined (Jenkins, P.R. 1996).

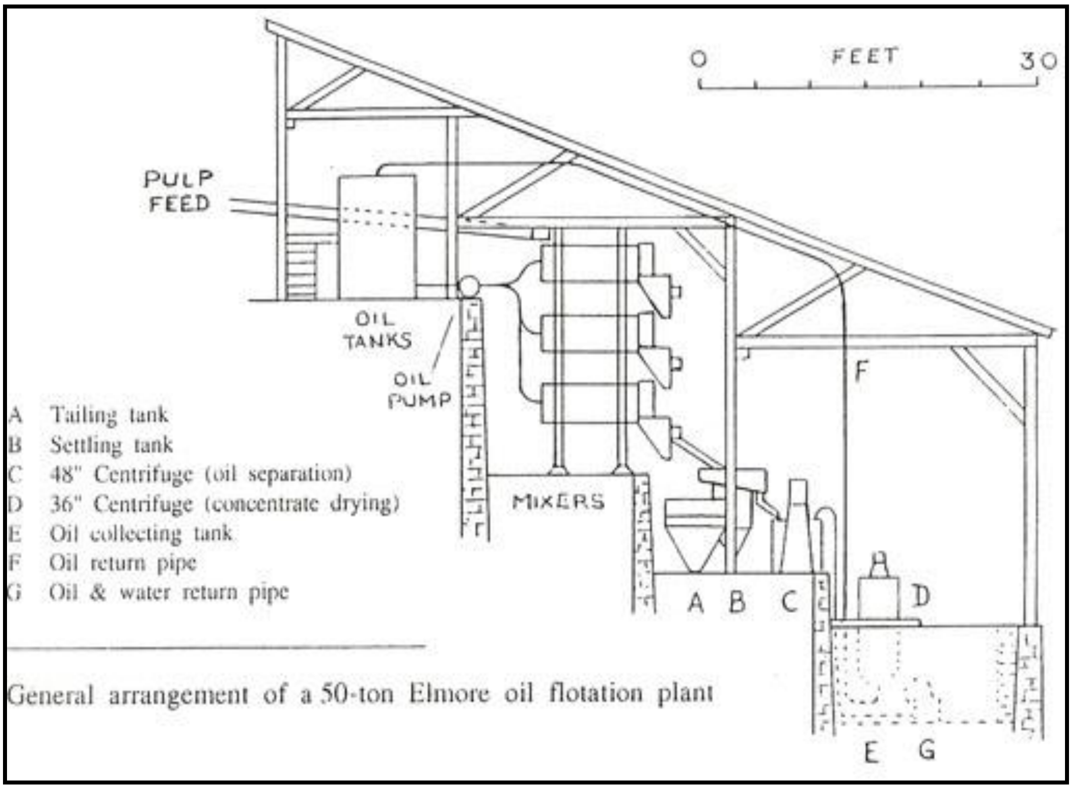

Figure 2.2: Typical Elmore oil flotation plant (Jenkins, P.R. 1996)

The ore pulp was fed into the mixers where the oil was added to mix with the pulp. The mixture was transferred by means of gravity to the settling tank where the ore separation took place. The oil was recovered utilizing centrifuges and returned to the oil tanks for reuse.

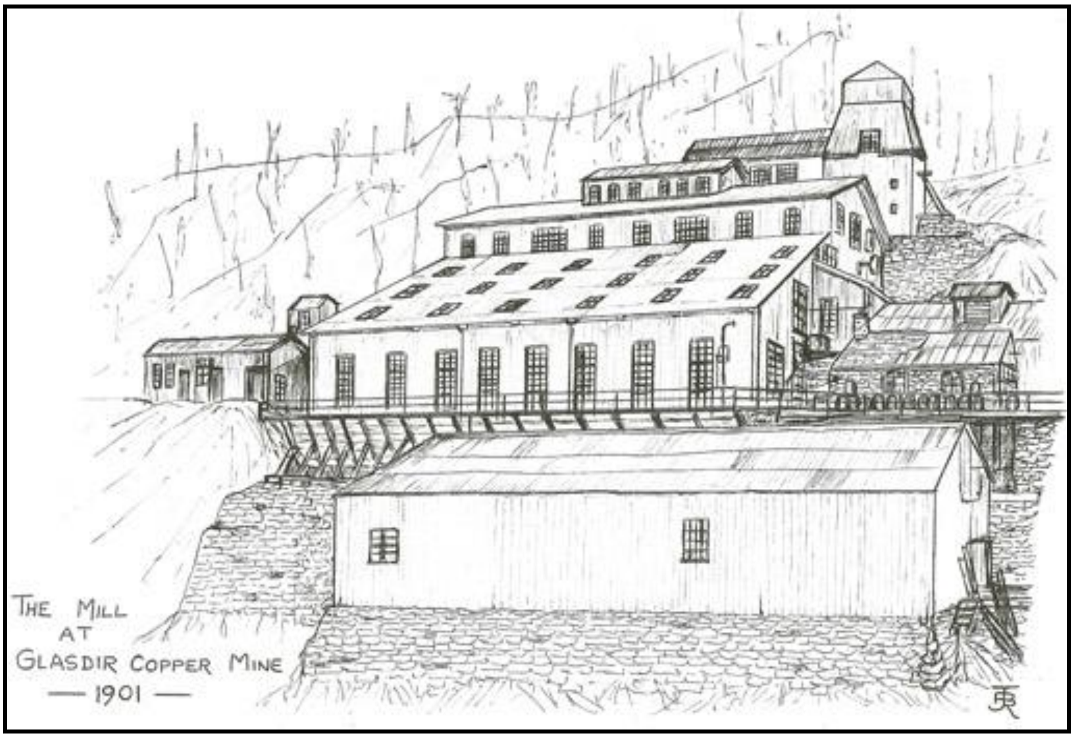

Figure 2.3: Glasdir copper mine mill (Jenkins, P.R. 1996)

The oil flotation process was patented by the Elmore brothers in 1898. They published a description of the process in the *Engineering and mining Journal* in 1903 Jenkins, P.R. 1996. The Elmore brothers improved the process after they recognised the importance of the presence of air bubbles in the oil flotation cell. The air bubbles assisted the oil to carry the mineral particles to the surface improving the recovery rate. A company named the Ore Concentration Syndicate Ltd was formed by the Elmore brothers to promote the use of the oil flotation process worldwide (Jenkins, P.R. 1996).

2.3 Froth flotation theory

Froth flotation is a physico-chemical beneficiation process making use of the difference in surface properties of the gangue minerals and the valuable minerals. The process and theory of froth flotation can be described as complex involving three phases namely; solids water and froth. The more complex plant designs have many sub processes and interactions. These plants are not easily understood and experienced operators and metallurgists are needed to run these plants.

There are three mechanisms that is crucial in the flotation process for material to be recovered. (Wills and Napier-Munn, 2006)

- Discerning attachment to air bubbles by hydrophobic material
- Suspension in the water where air bubbles are passing through
- Aggregation of particles in the froth bed in the froth zone of the flotation cell (particle entrapment).

The particle to bubble attachment of the valuable mineral is the most important mechanism of froth flotation and these particles represent the majority of particles found in the froth bed. The mechanism of selective attachment to a bubble is known as “true flotation” (Wills and Napier-Munn, 2006).

In contrast to true flotation, which is dependent upon the surface properties of the valuable mineral, unwanted gangue material can be recovered to report to the product by entrainment or entrapment. Gangue minerals in a fine state will be suspended in the particle to bubble attachment zone. These particles do not attach to the bubbles, but they are entrapped between the air bubbles and the valuable minerals. The gangue minerals are raised to the surface with the valuable minerals to be embedded in the froth collection zone (Wills and Napier-Munn, 2006). Entrainment and entrapment is a common occurrence in industrial flotation plants. It is due to this fact that single stage flotation circuits are uncommon and multiple stage flotation plants are being utilized to ensure an economically acceptable product.

Figure 2.4 illustrates the particle to bubble attachment of the hydrophobic coal particle in a typical true flotation environment. The natural buoyancy of the air bubbles will lift the particles to the surface to be collected in the froth zone. The hydrophilic mineral particles will not attach to any air bubbles and will sink to the bottom of the flotation cell to be discarded into the flotation plant tailings circuit. When the gangue mineral particles are much smaller than the hydrophobic particles and stay in suspension, entrapment can occur and cause gangue material to be raised to the surface by being trapped between air bubbles and hydrophobic particles. The gangue material can also be dragged to the surface by the capillary forces created by the upwards movement of the air bubbles.

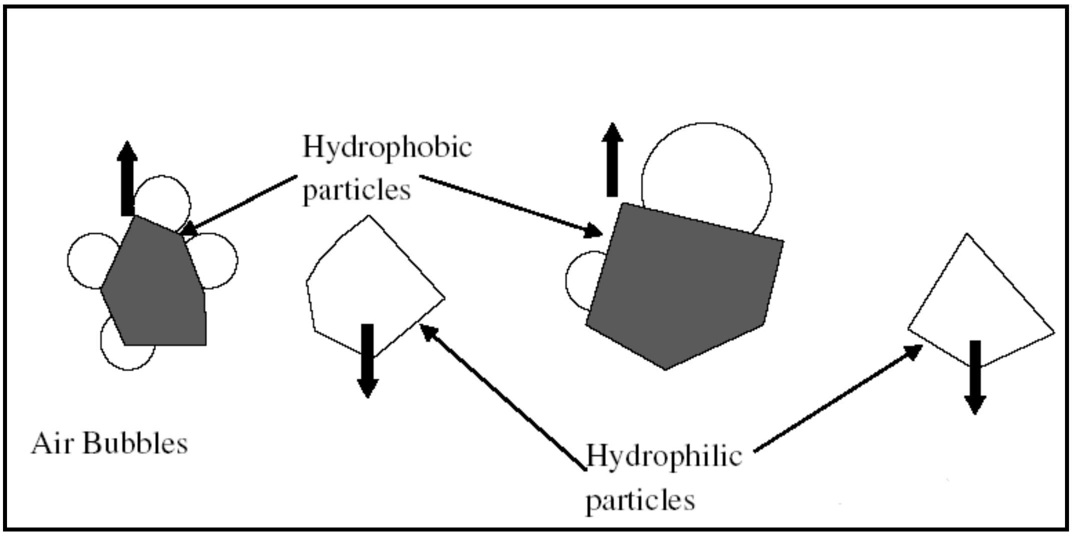

Figure 2.4: Attachment of hydrophobic material to an air bubble (Kawatra, S.K.)

Particle to bubble contact is very important in the sense that if there are no bubbles present in the direct vicinity of the valuable mineral, no attachment will be possible and the particle will be lost to the tailings. The contact zone in the flotation cell needs to be injected with sufficient air to create the bubbles needed to attach to the valuable mineral and transport it to the froth zone. The slurry feed has to be dispersed into the contact zone to ensure that each and every hydrophobic particle is given the opportunity to attach to an air bubble.

Figure 2.5 indicates what is happening in the Wemco Smart Cell while in operation. The slurry enters the cell from the bottom side of the cell. The rotor creates a draught through the false bottom and the draught tube lifting the slurry to come into contact with the rotor. Air is sucked through the air intake into the rotor compartment situated in the standpipe. The air and slurry is displaced by the rotor through the disperser and disperser hood. The disperser and disperser hood mix the air bubbles and the slurry together to ensure that they enter the bubble to particle attachment zone simultaneously and at the same angle. This all happens to ensure that the valuable mineral comes into contact with the air bubbles. The amount of air being introduced into the flotation cell is also critical. When the critical operating parameters and adjustments in the flotation cell are not being adhered to, airflow into the cell will be affected impacting negatively on the recovery of the valuable mineral.

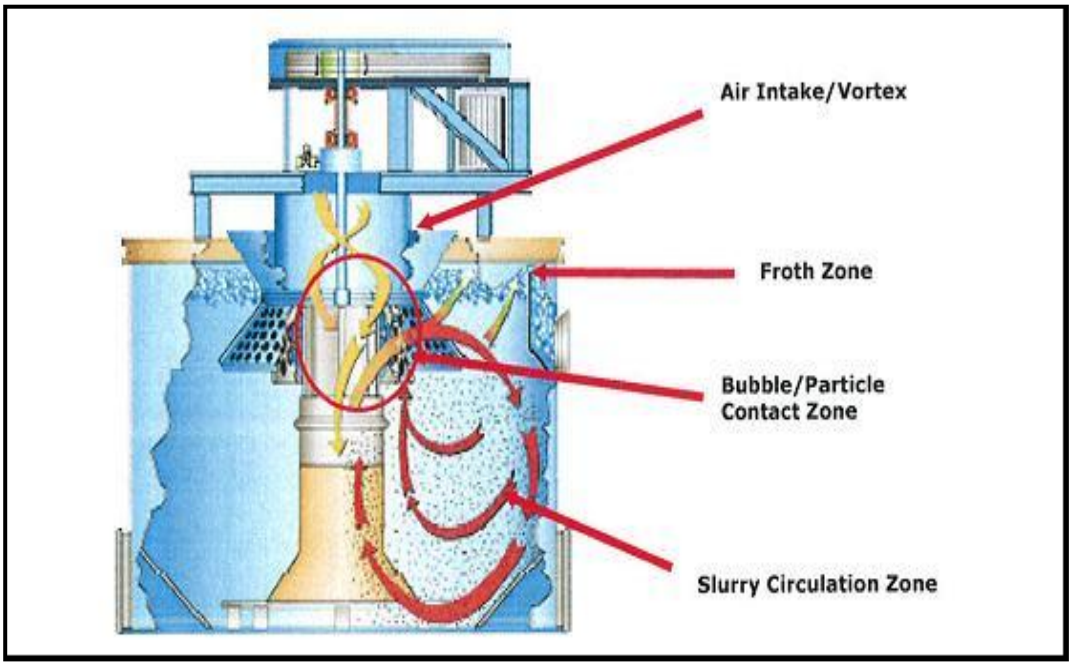

Figure 2.5: Particle to bubble contact zone (Wilkes, K.D.)

One of the most important aspects of froth flotation is the contact angle between the particle and the air bubble. The interfacial energies between the solid, liquid and gas phases will determine the attachment of the bubble to the particle surface (Kawatra, S.K.). The Young/Dupre Equation below can be used to determine this.

$$\gamma_{1v}\cos\theta = (\gamma_{sv} - \gamma_{sl})$$

In the equation γ_{1v} is the surface energy of the liquid to air bubble interface, γ_{sv} is the surface energy of the solid to air bubble interface and γ_{s1} is the surface energy of the solid to liquid interface. The angle formed at the junction between the air bubble, solid and liquid phases is known as the contact angle and the symbol therefore is θ. If the contact angle is small, the attachment will be weak and the particle will not be able to float to the surface.

In the case of a large contact angle, the attachment will be very strong and the bubble and particle will rise to the surface. Figure 2.6 indicate a contact angle close to 90° which will be sufficient for effective froth flotation in most cases (Kawatra, S.K.). Air bubble size as well as the size of the particle will also affect the contact angle and care should be taken to ensure that the air bubble size is compatible with the size fraction being beneficiated.

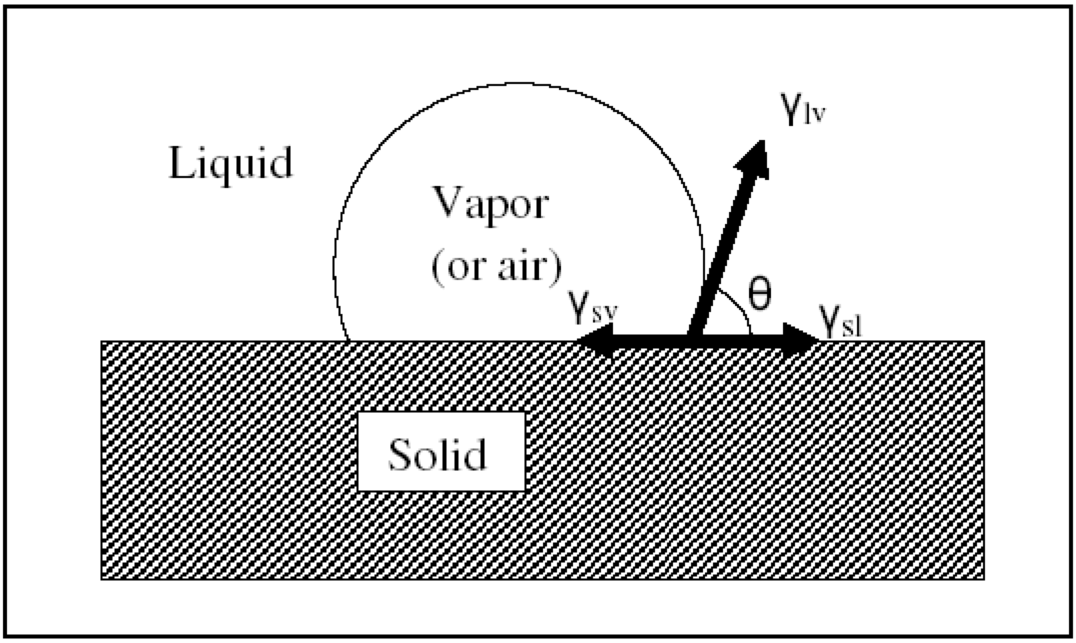

Figure 2.6: Particle to bubble contact angle (Kawatra, S.K.)

2.4 Froth flotation reagents

2.4.1 Collectors

Coal particles have natural buoyancy due to the small difference in density compared to water. This insignificant difference in density influences the attachment of the coal particles to the air bubbles. The coal particles tend to follow the streamlines created by the capillary forces which are caused by the movement of the air bubbles instead of making contact and attaching to the bubbles (Powell, D.M. 1999). The correct selection and application of flotation reagents will solve this problem and increase the product yield as well. Care should also be taken when applying reagents as the quality of the product can also be affected by certain reagents. Larger particles with embedded gangue material can be coated by the reagents and attach to an air bubble. The embedded minerals will report to the coal product reducing the quality.

Coal will attempt to attach to an air bubble or any other material less polar than water. A coal particle can therefore easily be wetted by non-polar material, such as hydrocarbon oil, rather than be wetted by water (Briscoe and Vander Veen, 1990). Minerals like shale, mudstone and clays have a polar nature and will therefore rather be wetted by water than by hydrocarbon oil.

The most commonly used collectors in coal flotation are diesel oil, liquid paraffin and kerosene. They form part of the petrochemical product list. Diesel components are almost exclusively hydrocarbons of varying molecular weights. The hydrocarbon groups found on coal surfaces is similar to the hydrocarbons found in diesel (Powell, D.M. 1999). Hydrophobic bonding between the collector molecules and associated groups on the coal surface is widely accepted as the mechanism of adsorption.

It is also important to note that the rank of the coal being beneficiated will determine the concentration of collector needed in the froth flotation process. According to Fuerstenau (1976) very little collector is needed to float bituminous coal and little to very much collector is needed to float low rank bituminous coal. Hydrophobicity is not created by adding a collector to the coal bearing pulp, but the already hydrophobic surface of the coal particles will increase in hydrophobicity when the collector is added (Nimerick *et al,* 1980). The natural hydrophobicity of the coal will determine the concentration of collector to be applied as well as the type of collector that needs to be applied.

2.4.2 Frothers

A frother is introduced to the flotation process with the main aim to create a stable froth in a flotation cell. Frother addition will increase flotation kinetics and allow the drainage of entrapped gangue material. Frothers are mainly chemically similar to ionic collectors. Many collectors, such as oleates, can be used as powerful frothers. Care should however be taken as some of these collectors can be too powerful to be used as an effective frother. The froth produced can be too stable creating downstream problems (Wills and Napier-Munn, 2006). Excessive frothing in flotation cells can cause froth build-up on the surface of thickeners is an example of a problem that can be experienced when applying a strong frother. A good frother should ideally not have collecting power and must create a froth bed just stable enough to transfer the floated material from the flotation cell to the froth collecting launder.

The strength of a frother can be measured by the “hydrophile-lipophile balance (HLB)” number. HLB is dimensionless and assumes the arbitrary scale from 1 to 20. The degree of surface activity of the molecule will be greater when the HLB number is increased (Powell, D.M. 1999). By investigating the extent of partitioning of the molecule between oil and water the HLB number can be determined. The ability to orientate itself at an interface of two different materials such as air and water is of utmost importance. The molecule orientates itself in such a way at this interface that the hydrophobic and hydrophilic tendencies can be accommodated.

The hydrophilic end moves towards the more polar water phase and the hydrophobic end orientates into the air bubble. The hydrophobic ends will orientate itself in an air bubble towards the inner centre part of the bubble and the hydrophilic part towards the bubble wall. The frothing action is possible due to the ability of the frother to absorb on the air-water interface. This will happen because of the frother’s surface activity and the ability to reduce the surface tension. The air bubble will be stabilised as a result of the interference by the added frother. (Wills and Napier-Munn, 2006)

CHAPTER 3: Process description and experimental methods

This chapter will look at the Tshikondeni Coal flotation plant process as well as the details of the Ultimate Flotation Cell. One subsection will be dedicated to dense medium fine coal beneficiation. Test work was conducted on Tshikondeni fine coal with the aim of constructing a fines DMS plant in the future. The experimental and sampling methods of the flotation optimization project will also be discussed in this chapter.

3.1 Flotation plant process description

Froth flotation has been used as a beneficiation process at Tshikondeni Coal to treat the fine coal as well as the ultra fine coal since the early 1980's. The coal being mined in the Soutpansberg coal reserve is high in vitrinite content and is the only hard coking coal left in South Africa. The beneficiation plant treats coal which has been prepared to a size range of approximately -17mm. The plant consists of two mirror image modules and is designed to treat 250t/h combined. The coal is fed to a pre-wash screen where the -1.4mm material is removed and pumped to the flotation plant. The + 1.4mm material is beneficiated in one 710mm dense medium cyclone per module.

Figure 3.1 illustrates the flow diagram of the flotation circuit and is also attached as Attachment D. The coal pulp is pumped from the pre-wash screen to the flotation conditioning tank. Collector and frother are applied to the pulp in the conditioning tank. The main purpose of the conditioning tank is to allow the coal particles to be coated by the collector prior to entering the flotation cells. The plant consists of six flotation cells per module which is connected in series. The tailing of each cell flows to the flotation cell that is next in line. The product from the product launders of all the flotation cells is channelled to a product sump. The product pulp is pumped to a 32t/h Humboldt solid bowl centrifuge where it is dewatered and discharged onto the flotation plant product conveyor. The flotation plant product quality parameter is 12 percent (%) ash. The flotation plant tailings were pumped to a spiral plant where the pulp was deslimed by two 350mm polyurethane cyclones. The deslimed pulp was dewatered and the - 0.63mm material removed on a high frequency deslime screen. The screen overflow were fed to the spiral plant in the attempt to recover some of the + 0.5mm – 1.4mm material lost in the flotation cells. The spiral product were combined with the flotation plant product and dewatered in the Humboldt centrifuge.

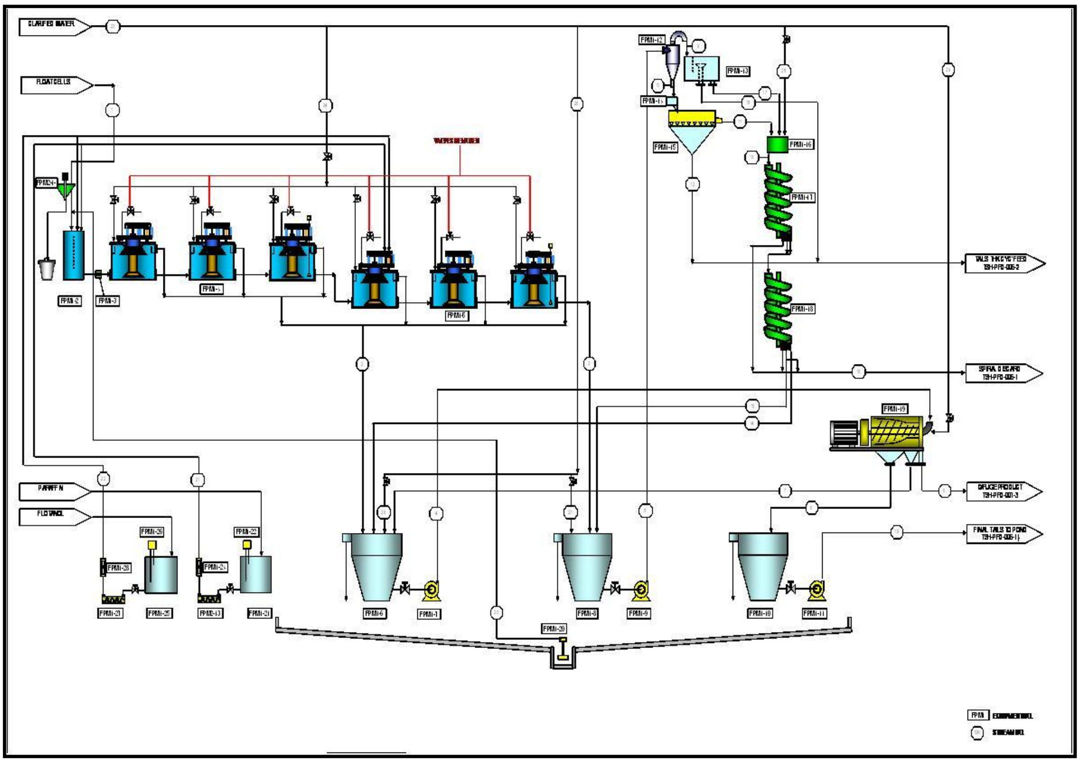

Figure 3.1: Flotation plant flow diagram

Major problems were however experienced with the quality of the combined product and after some intensive test work it became evident that the product ash obtained from the spiral plant was 17% ash. The product specification at the time was 12% ash. Densities in the cyclone plant were decreased to meet the customer requirements of 12% ash impacting negatively on the overall plant yield. The decision was made in 2004 to bypass the spiral plant which had an immediate positive impact on the overall plant yield. Even after decommissioning the spirals it was evident that between 3% and 5% product was still present in the flotation plant tailings composite sample at the end of every month. The coal present in the tailings was mainly in the – 0.5mm + 1.4mm fraction range. This indicated that inefficiencies in this size range still existed in the flotation plant. Numerous improvement projects were initiated and implemented with mixed successes as discussed in chapter one.

The product obtained from the flotation circuit plays a major role in achieving the overall predicted plant yield. The current product specification is 14% ash and the ash obtained from the flotation circuit is between 10 and 11%. The flotation feed only represents between 25% and 30% of the total feed to plant where the DMS cyclone plant treats 65% to 70% of the total feed to plant. When the recovery in the flotation plant is at its maximum predicted yield at 12% ash, the DMS product quality can be increased to 15% to achieve a combined 14% ash product. The yield impact in the DMS plant is much greater as it treats the largest percentage of the feed to plant and will therefore have the largest impact on the total product yield. The realization of the possible impact on the overall plant yield was a major motivator to improve the flotation circuit over the last 12 years. These projects were discussed in sub-chapter 1.2.1.

3.2 Fine coal DMS beneficiation

Coal was produced from three different shafts at Tshikondeni during 2007 namely Vhukati shaft, Nyala shaft and Mutale shaft. Nyala shaft was the oldest shaft at the time nearing the end of shaft life. Vhukati and Mutale shafts would produce coal to the end of mine life. Major problems were experienced beneficiating coal mined at Mutale shaft and low yields were achieved compared to yields obtained from the other shafts. It must be noted that the coal in the Mutale shaft reserve was of lower quality than the coal found in the other shafts. Higher percentage contamination was also being mined at Mutale shaft due to extensive faulting throughout the shaft together with numerous dyke intrusions and bad roof conditions. Test work indicated that coal beneficiated from Mutale shaft are problematic to treat in flotation cells and different methods of beneficiating the + 0.5mm – 1.4mm material needs to be investigated.

Samples were taken in March 2007 from the plant feed conveyor of Vhukati and Mutale coal respectively. Float and sink analysis were conducted on the different size classes after the samples were screened into the following size fractions (Van der Merwe, 2007).

- + 6.4mm
- - 6.4mm + 3.4mm
- - 3.4mm + 1.4mm
- - 1.4mm + 0.6mm
- - 0.6mm + 0.212mm
- - 0.212mm + 0.075mm
- - 0.075mm

The results obtained from the float and sink analysis were used in a LIMN simulation to predict a yield at 14% product ash. The results obtained compared favourably with Coaltech 2020 results. The LIMN simulation program with the assistance of Coaltech 2020 was used to simulate four scenarios and ultimately in the financial evaluation as well. Table 3.1 is a summary of the results and figure 3.2 gives a clear indication of the difference in washability of the coal mined at Vhukati and Mutale shaft.

Scenario	Description	Capex (R mil.)	OPEX saving (%)	Total yield (LOM)
Scenario 1	▪ All to 710mm cyclone	4.8	47.2	62.7%
Scenario 2	▪ -1.4mm to 420 cyclone ▪ +1.4mm to current 710 cyclone	6.9	41.7	61.8%
Scenario 3	▪ -0.212mm to flotation ▪ +0.212mm to 710 cyclones	4.5	36.6	63.4%
Scenario 4	▪ -0.212mm to flotation ▪ -1.4+0.212mm to 420 cyclone ▪ +1.4mm to current 710 cyclone	7.8	32.9	64.1%

Table 3.1: LIMN and Coaltech 2020 results (Van der Merwe, D 2007)

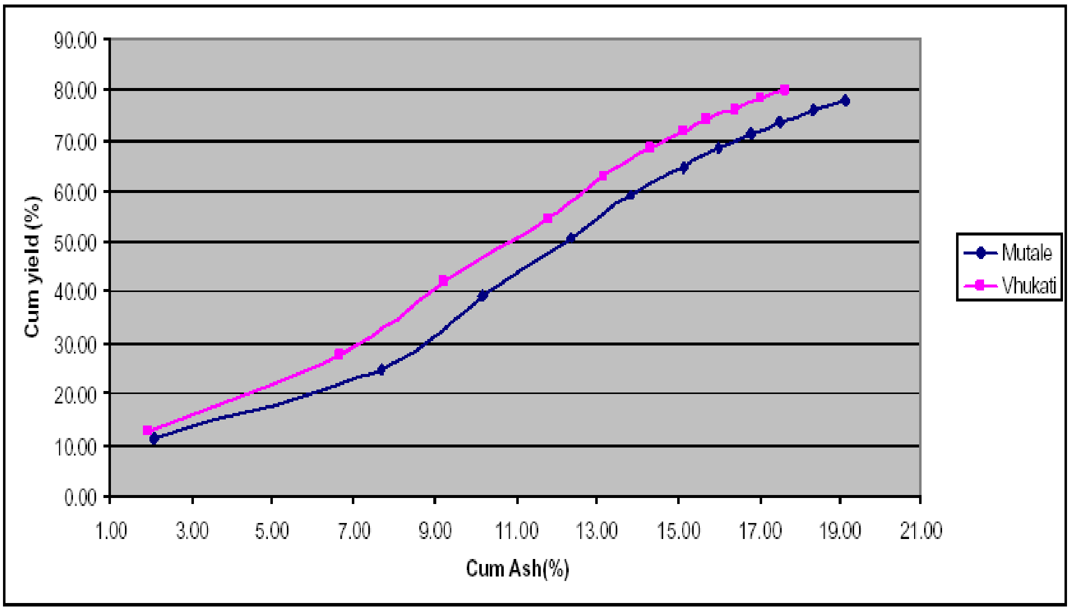

Figure 3.2: Vhukati and Mutale washability graph (Van der Merwe, D. 2007)

Scenario two was opted for feasibility at the time as the technology has already been proven. A similar plant as described in scenario two was at the time being built in the beneficiation plant at Leeuwpan Mine in Mpumalanga. The best option will however be scenario four, even though the capital expenditure (Capex) as well as the operating expenditure (Opex) is the most expensive compared to the other scenarios. The yield obtained from this scenario is however the highest with a difference of 2.3% between scenario two and four. The difference in Opex and Capex is however insignificant when taking into account the high value of Tshikondeni's hard coking coal product.

Pilot scale test work was also conducted at Landau by Coaltech 2020 on Tshikondeni's coal (De Korte and Mcgonigal, 2005). The Ecart Probable (moyen) (EPM), which indicates the sharpness of separation of the unit being used in the beneficiation process, was determined for Tshikondeni's coal utilizing DMS cyclones and is depicted in figure 3.3.

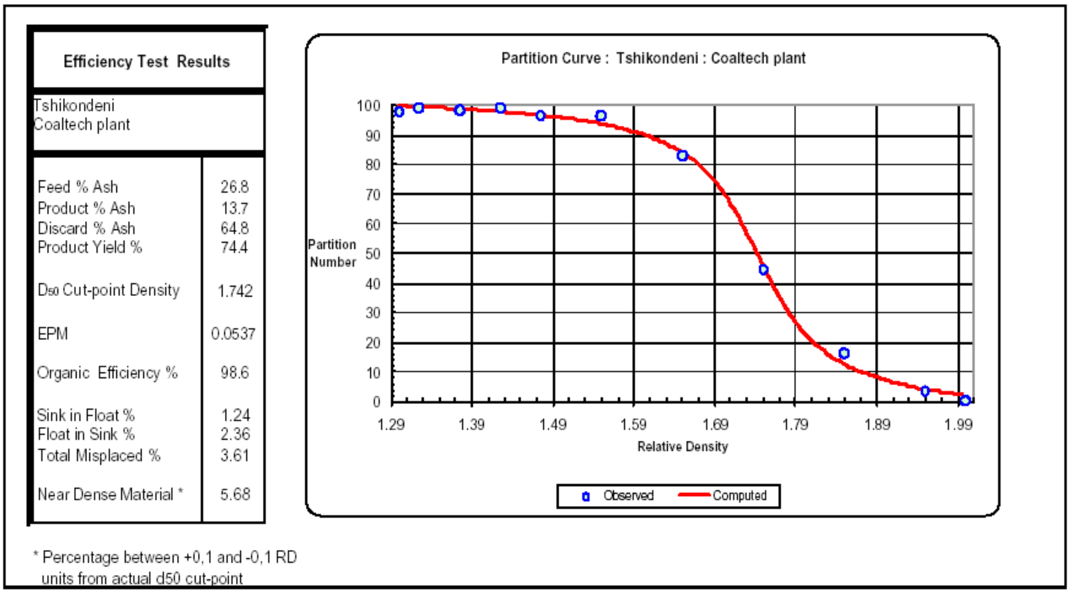

Figure 3.3: Tshikondeni fine coal EPM (De Korte and Mcgonigal, 2005)

The correct medium density at the time of sampling was 1.35 and the reduced EPM obtained was 0.0537. Product samples were taken at various densities after efficiency sampling were completed. An 13.7% ash product was produced at a yield of 74.4%. The organic efficiency was 98.6% (De Korte and Mcgonigal, 2005). These results compared favourably with the simulation results and it was quite clear that Tshikondeni fine coal can be beneficiated utilizing DMS cyclones with success. Figure 3.4 graphically indicate the results obtained from the samples taken of the product at different densities.

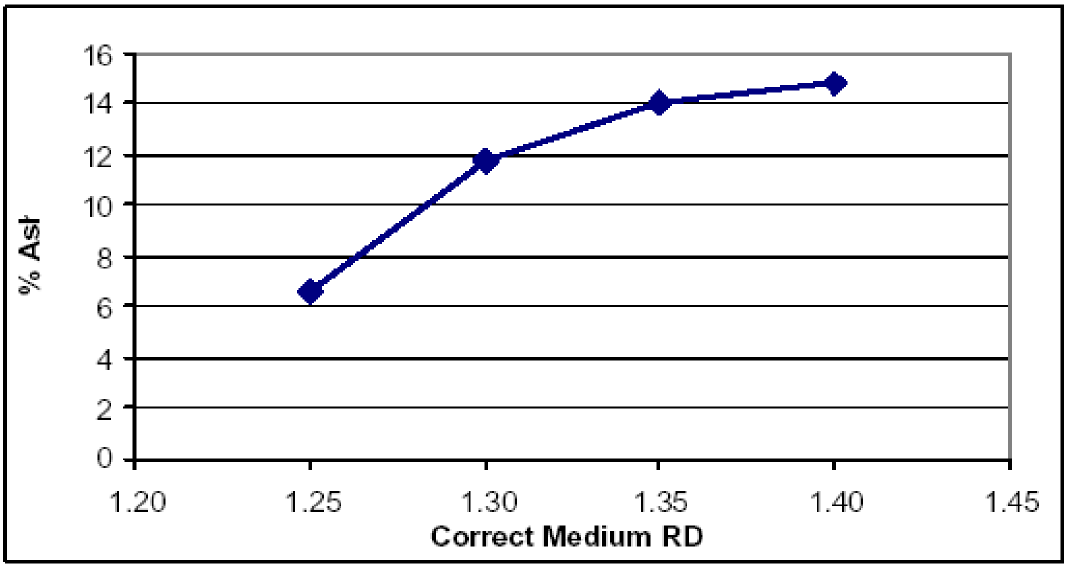

Figure 3.4: Product ash at different densities (De Korte and Mcgonigal, 2005)

A feasibility study was conducted during the course of 2008 and the process flow diagram depicted in figure 3.5 was used for the preliminary plant design. Capital cost to built a fines DMS cyclone plant in each of the two modules came close to R 20 000 000. The project was however put on hold early in 2009. An investment of R20 000 000 was not viable when considering the five years life of mine that was left. The project will not be abandoned and is still a viable option if the life of mine can be extended by finding new minable reserves in the future. DMS is most probably the most efficient way of beneficiating fine coal and a fines DMS plant is being operated successfully at Leeuwpan mine.

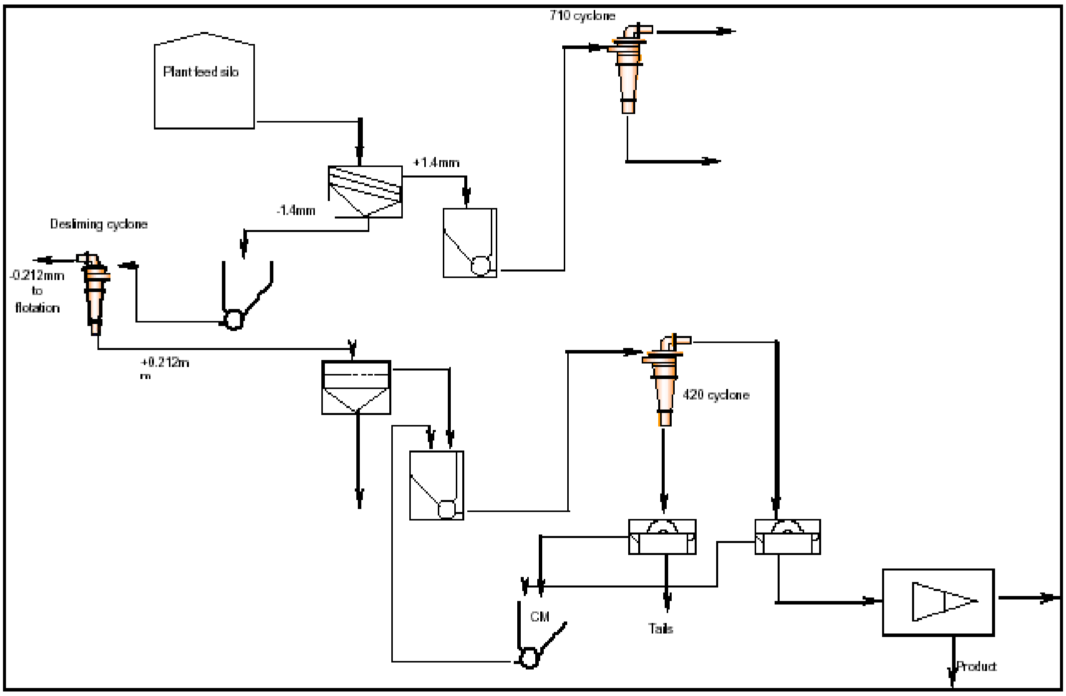

Figure 3.5: Fines DMS plant flow diagram (Van der Merwe, D. 2007)

3.3 <u>Details of the Ultimate Flotation cell</u>

As discussed in chapter one, a rotor and disperser were obtained from Ultimate flotation in 2004 with the main aim to increase the recovery of the + 0.5mm – 1.4mm in the flotation circuit. The rotor and disperser were designed for copper extraction in a flotation cell fed from recovered slimes dams and was only available for a short period of time to be tested. With the first start-up after installation, it became evident that there was something different about the drive assembly. The froth created in the flotation cell was stable, but in such a way that the froth bed continued to flow over the flotation cell lip into the product launder. There was a constant battle since plant commissioning in 1996 to get the froth to flow into the product launder. After installation of the eight blade rotors with the aim to pick up the coarse material in the flotation feed, many hours has been spent to get the froth to flow with no real success. The eight blade rotors created major disruptions in the flotation cells as well as on the water surface. This was due to the extra energy induced as well as the speed at which the rotor needs to turn to create the energy needed. Water was splashed over the flotation cell lip causing gangue minerals to be transferred to the product launder.

In order to mitigate this problem, the level in the flotation cells had to be lowered to prevent water from entering the product launders. The net effect was that the froth bed area in the cells became too deep causing the froth to flow even slower to the product launders. Operators made use of manual paddles to remove the froth from the cells at least once every hour. Water was also sprayed into the froth on a regular basis to encourage fast froth flow. All of these problems disappeared the moment when the ultimate flotation drive was introduced into flotation cell number one in Module A. The self induced airflow increased from 2.4m^3/h to 6 m^3/h assisting in the increased flow rate of the froth from the flotation cell. No surface water disturbances were noticed in the flotation cell due to the slow speed at which the rotor was rotating to induce $6m^3/h$ of air. Froth flow from the cell was rapid and constant which indicated that no resistance to froth flow existed in the flotation cell. The rotor installed in 2004 can be observed in figure 3.6.

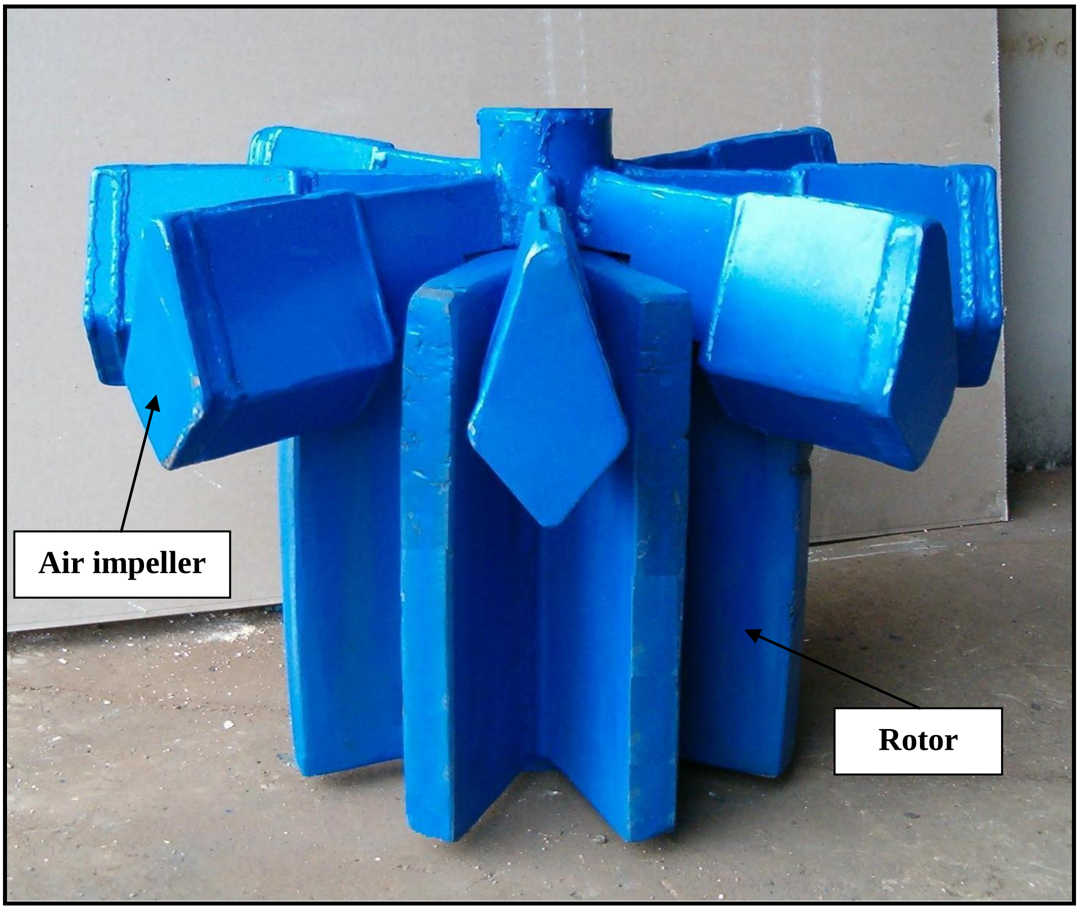

Figure 3.6: Ultimate flotation rotor tested in 2004

It is important to notice that the rotor is basically the same than the original eight blade rotor installed in the existing flotation cells. The air impeller is a new addition to the configuration together with the newly designed disperser. The new disperser does not make use of disperser hoods. The disperser apertures are designed to disperse the particles and air bubbles at an angle as can be seen in figure 3.7.

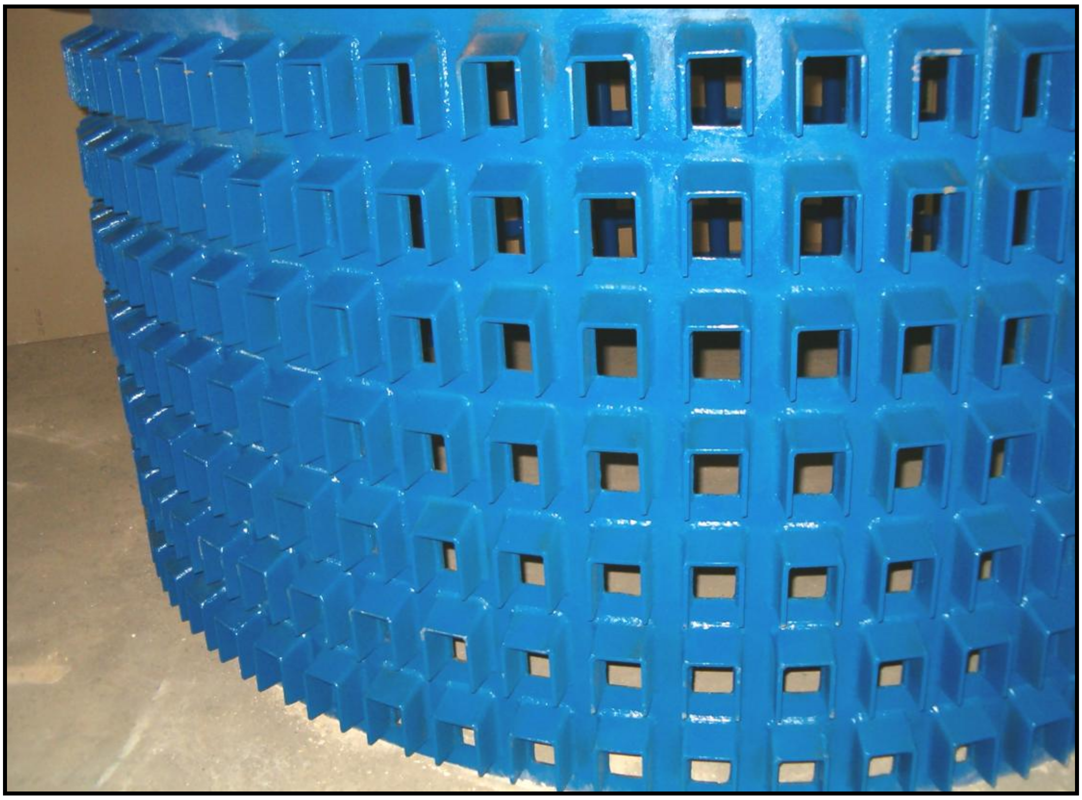

Figure 3.7: Ultimate flotation disperser tested in 2004

The only concern at the time of the first test was that there was hardly any + 0.5mm material noticeable in the froth. The froth was however saturated with the - 0.5mm material which floats much quicker than the coarse material. Another rotor and disperser installed in cell number two would have been the ideal situation. There was however not one available at the time of the test and the decision was made to abandon the test work based on the fact that no coarse material was being floated in the cell fitted with the ultimate flotation mechanisms. Another project to remove the froth from the flotation cells was pursued and mechanical paddles were installed in 2005. The project was successful and froth was removed on a continuous basis. The paddles however created other problems such as added water surface disruptions as well as the breakdown of the froth bed in the flotation cells. Maintenance of the paddles also proved to be a major challenge. Gearboxes with an offset output of 90° had to be installed in order to be able to remove the froth in the cylindrical tank. These gearboxes broke down continuously and procurement proved to be problematic. The paddles had to be removed if replacement gearboxes were not available to prevent them from blocking the natural flow of the froth.

It was realised early in 2008 after the Fines DMS project was put on hold that the flotation cells in Tshikondeni plant had to be replaced. The flotation cells were showing their age and wear and tear was evident upon inspection. Replacement was overdue after the decision was made to postpone the replacement until it became known what the route forward will be relating to the flotation cells and the Fines DMS plant. With the Fines DMS plant taken out of the equation, the opportunity presented itself to redesign the flotation plant with the main aim to mitigate the problems experienced in the flotation plant as discussed in previous chapters. These problems are listed as follow:

- Level control in the flotation cells
- Disruptions to water surface
- Froth flow resistance
- Excessive wear to cell internal mechanisms

One of the most important factors to keep in consideration prior to implementation of any new flotation plant is the availability of an air compressor. The Tshikondeni plant is not equipped with an instrument air system. Excessive water build-up in the air system due to the extreme summer temperatures and humidity caused the plant to be designed without the addition of an air compressor system. Forced air flotation cells could therefore not be considered and Tshikondeni went out on tender for conventional flotation cells with the ability to induce in excess of $6m^3/h$ of air into the cell. Only one tender was received and the decision was made to install Ultimate Flotation cells to beneficiate the – 1.4mm material to the end of mine life.

The ultimate flotation cell was improved since the first test was conducted in 2004. Figure 3.8 indicate the latest rotor and disperser installed in 2009. The difference between the rotor and disperser tested in 2004 and the rotor and disperser installed in 2009 can clearly be seen when the pictures are compared. Figure 3.9 illustrates the false bottom developed to increase the flow of the slurry into the flotation cell. The slurry is guided into the middle of the draught tube by the slurry feed arrangement incorporated in the false bottom.

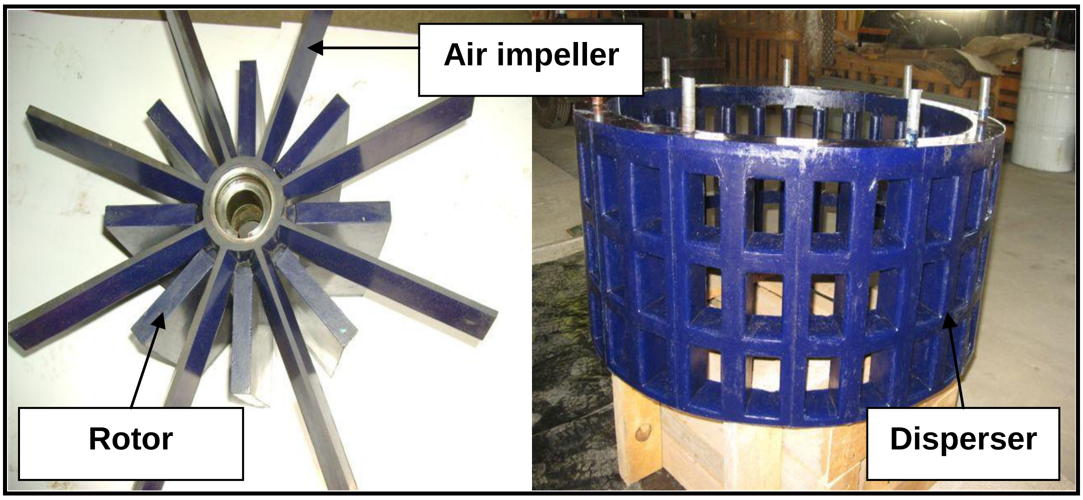

Figure 3.8: Latest Ultimate Flotation rotor and disperser

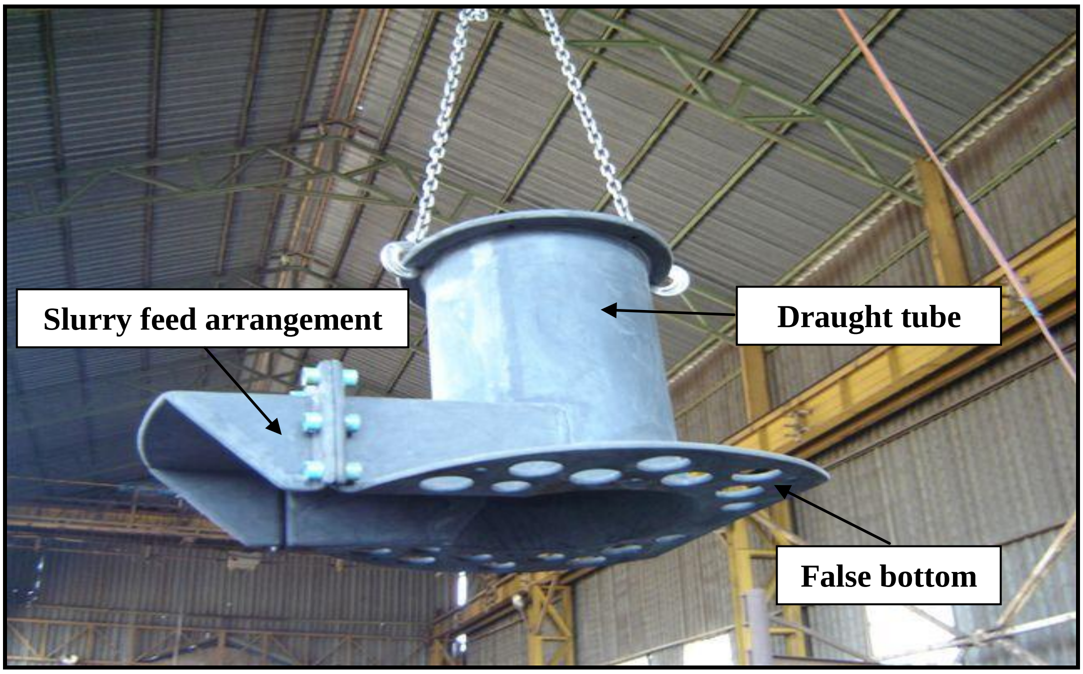

Figure 3.9: Draught tube and false bottom arrangement

Accurate level control was one of the biggest challenges experienced since the plant was commissioned in 1996. As described in chapter one, flotation cells one to three were installed on the same level. Level control was done in cell number three causing some cells to overflow while the level was too low in some of the other cells. Installation of stainless steel rings assisted in rectifying some of the problems experienced with level control. The new flotation cells were installed on different levels with a variation of 440mm from the one cell to the next cell. Level control was introduced to each flotation cell allowing the operator to adjust the levels individually and accurately. The level control system consists of two components. The first component is an adjustable gate which can be adjusted electrically to control the amount of slurry being allowed to flow to the next flotation cell. The slide can be seen in figure 3.10.

Figure 3.10: Level control slide

The slide can also be utilized when the flotation cells needs to be drained. When opened fully, the flotation cells will drain in a very short time allowing the operators to wash the cell when needed. The cells can also be stopped and started under load without experiencing any problems. The slurry in the cell will drop to the bottom of the cell as the slide is opened 15mm only during normal operation. A minimum amount of slurry in the flotation cell is drained to the sumps in case of a power failure or system shutdown. The slurry is being picked up as soon as the water level reaches the rotor once the flotation plant is started under load.

The second component utilized to control the level in the flotation cell works on the same principal as the ring- stock method utilized in slurry dams for level control and the recovery of clean water. This system is totally manual and requires an operator to physically observe the level and add or remove a ring from the system to control the level. Once the desired level is obtained, the flotation cells can be left alone without any problems experienced where level control is concerned. The level will stay stable without any fluctuations assisting with the constant flow of froth into the product launder. The removable rings can be observed in figure 3.11. Note the square tubing divider installed to prevent the froth from entering the centre pipe which is connected to the flotation cell tailings discharge launder. The level control rings can be seen and are easy to be removed or installed while the plant is in operation.

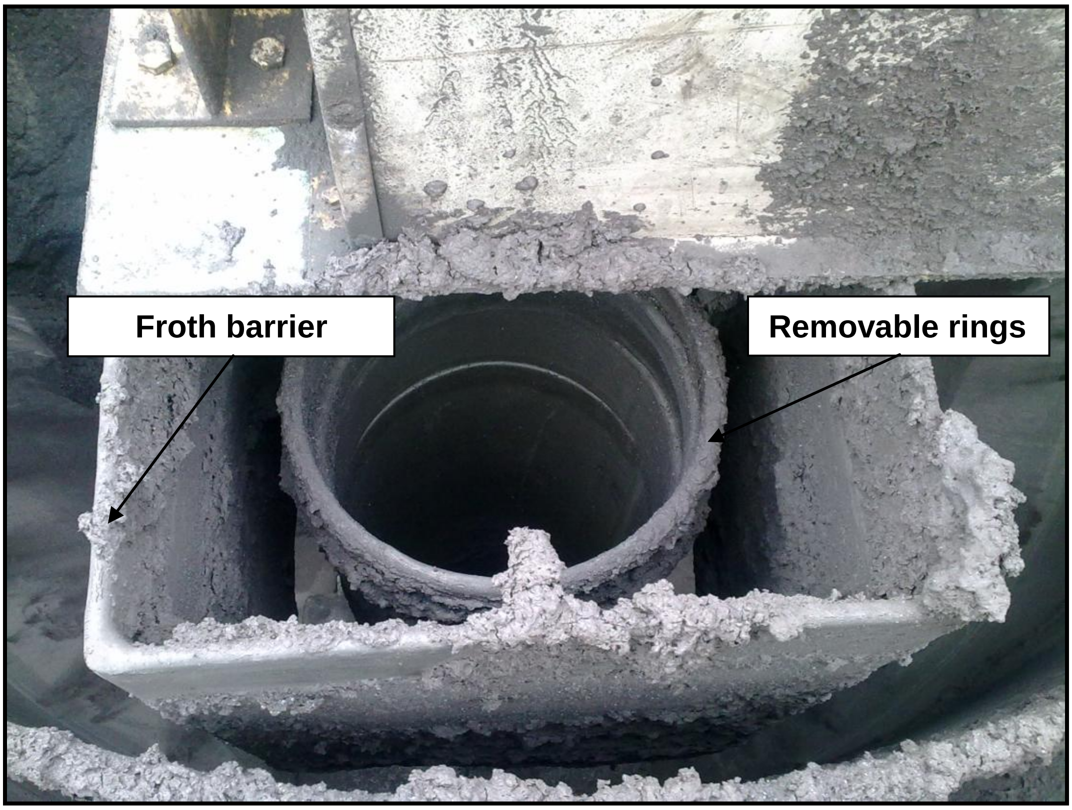

Figure 3.11: Level control ring in the flotation cell

Installation of the first three flotation cells were completed in August 2009 and the second three in October 2009. Start-up of the flotation cells was faultless other than the need to change the direction of some electrical motors. The cells were brought into production immediately without any problems. The levels can be adjusted to ensure that the froth flows freely into the product launders without allowing water to be carried over. Some resistance to flow were experienced in cell number one as the froth was very dense and stable in some areas. Each ultimate flotation rotor drive is fitted with a variable speed drive to allow the speed of rotation to be adjusted. The frequency of rotor number one was increased from 32 Hz to

33 Hz. The airflow into the cell increased and the froth started to flow without problems. Figure 3.12 depicts the Ultimate Flotation cells installed in module A.

Figure 3.12: Flotation cell installation in Module A

The rotor, disperser, draught tube inner and the standpipe inner is cast from polyurethane to ensure durability and extended equipment life. The rest of the flotation cell body and interior equipment is lined with a durable, chemical resistant rubber of 6mm thickness. Each flotation cell is also fitted with a spray system assisting the flow of the froth once discharged into the product launder. It is however very important to use clean water on the spray system as the nozzle aperture is very small and is prone to get blocked when using process water. The last item under discussion is the six froth collecting launders installed in each cell to assist with the froth collection in the flotation cells. These launders increase the lip area available for froth collection to more than double the normal cylinder circumference. These launders are adjustable with the inside tip being able to be adjusted 25mm lower than the flotation cell lip. The launders were adjusted in the beginning to be level with the flotation cell lip. These parameters will be used as a start-off point for the experiment and will be changed to monitor the impact there-off on the flotation circuit.

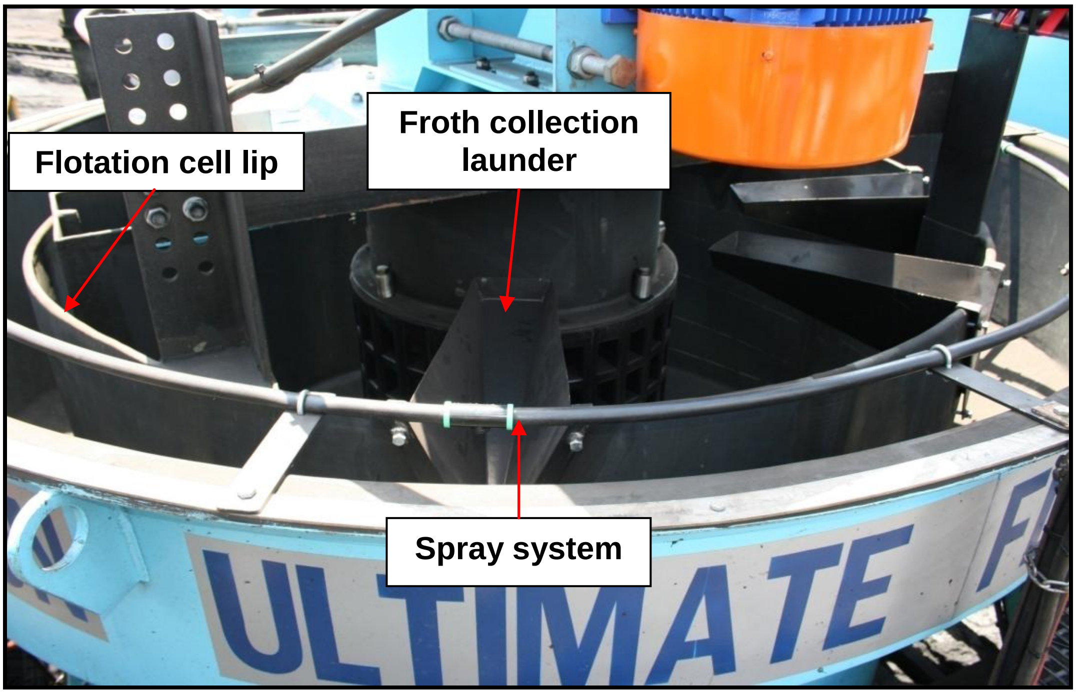

Figure 3.13: Froth collection launders

3.4 Experimental and sampling methodology

3.4.1 Experimental methodology

The main aim of the project is to increase the recovery of the + 0.5mm material in the flotation circuit. The flotation cells were set up with the froth collector launders adjusted level to the lip of the flotation cell and the variable speed were adjusted individually for each flotation cell drive. The initial start-up frequencies were as follow:

Flotation Cell Number	Frequency
Cell Number One	32 Hz
Cell Number Two	30 Hz
Cell Number Three	28 Hz

Table 3.2: Flotation cell frequency configuration

The frequencies of the flotation cell drives were adjusted until the optimum recovery in each flotation cell could be achieved. The adjustable launders could be lowered 25mm at the tip near the centre of the flotation cell increasing the flow of the froth into the froth launders. Care should however be taken as water can run into the launders if the water level in the cell is high. The froth bed can also be depleted when the froth is extracted too rapidly from the flotation cell surface. A froth bed is essential to ensure the consolidation of air bubbles and the entrapment of particles carried to the surface by the bubbles.

Once the optimal operating parameters have been achieved, the energy consumption as well as the actual speed of the rotor was calculated. Tshikondeni and every other mine have a 10 percent energy consumption reduction target. The energy consumption of the Ultimate Flotation cells was compared to the consumption of the old flotation cells that have been removed. The speed of the rotors in the cells plays a major role where the wear of components in the flotation cells is concerned. A slow turning rotor will cause less wear than a rotor turning at a fast speed. The rotor speeds were compared to indicate in which flotation cell one can expect the most wear.

3.4.2 Sampling and analysis methodology

The plant was started and the flotation cells were allowed to fill with water until the desired levels in all the flotation cells were reached. The feed to the plant (ftp) was started and the plant was allowed to run for at least 30 minutes on coal at 100t/h ftp. The reagents dosage was kept the same for all the tests conducted. The dosage rate is indicated in table 3.3.

	Millilitres Per Minute	Millilitres Per Tonne
Collector (Paraffin)	180ml/m	360ml/t
Frother (Betafroth FTN3)	45ml/m	90ml/t

Table 3.3: Flotation reagents dosage rate

After verification of the reagents dosing rates, sampling commenced. A special sampling device designed for the sampling of slurry was utilized to take all the samples needed. Increment sizes were therefore the same volume for all the samples taken at all the sampling points. Table 3.4 indicates the sampling methodology followed during the test period.

Sampling Points	Number of Increments	Test Duration
Flotation feed chute	32	8 hours
Cell one product	32	8 hours
Cell two product	32	8 hours
Cell three product	32	8 hours
Flotation plant product	32	8 hours
Flotation plant tailings	32	8 hours

Table 3.4: Sampling methodology

The samples were taken to the laboratory where they were dewatered in laboratory scale pressure filter and prepared for analysis. Ash analysis was conducted according to the following ISO standard: *Solid mineral fuels – Determination of ash content SABS ISO standard 1171:1997*. Screen analysis was conducted according to the following standard: *Hard coal – Size analysis by sieving. SABS ISO 1953. SABS edition 1 / ISO edition 2 1994.*

Samples were taken in the flotation plant prior to installation of the new flotation cells. The samples were analysed and the results were used as a reference point to measure the performance of the Ultimate Flotation cells. It must also be noted that the original flotation cells did not have any adjustment facilities and that the results obtained from the samples taken could not be bettered with the original Wemco Flotation cells.

CHAPTER 4: Results and discussions

This chapter will look at the results obtained from the analysis conducted on the samples taken during the test periods. Results obtained from the different flotation cells will be compared as well as the results of the Ultimate Flotation cell obtained at different operating parameters. Energy consumption will also be discussed.

4.1 Results and discussion

4.1.1 Wemco flotation plant test results and discussion

Test work started early September 2009 and samples were taken from the old flotation cells for screen and ash analysis. The first three flotation cells on each module were to be replaced with the new technology Ultimate Flotation cells. The comparison of results will therefore be conducted between the samples taken from the first three cells of the existing plant at the time and the newly installed flotation cells. No comparison will be made on the results of the – 0.5mm material as the project was aimed at increasing the recovery of the + 0.5mm material. It is however important to compare the ash of the feed, product and waste to determine if the overall recovery in the flotation cells has improved as well. The ash balance formula was used to calculate the overall yield of the respective flotation plant at each test. The yields were mainly used to determine the average yield of the flotation cells for comparison. The ash balance formula is as follow:

$$\text{Yield} = \frac{\text{Tailings ash} - \text{Feed ash}}{\text{Tailings ash} - \text{Prod ash}}$$

Ten samples were collected from the 1st of September 2009 to the 15th of October 2009 prior to the replacement of the flotation cells. Screen analyses as well as ash analyses were conducted on the samples immediately after the samples were taken. The results were summarised in tables and the Wemco flotation cell results can be observed in Attachment B, table B1 to table B12. The results for the eight tests were averaged in a table which were used for the creation of graphs. The graphs will be used as illustrations for discussions. Figure 4.1 depicts the average + 0.5mm material recovered in the eight test runs.

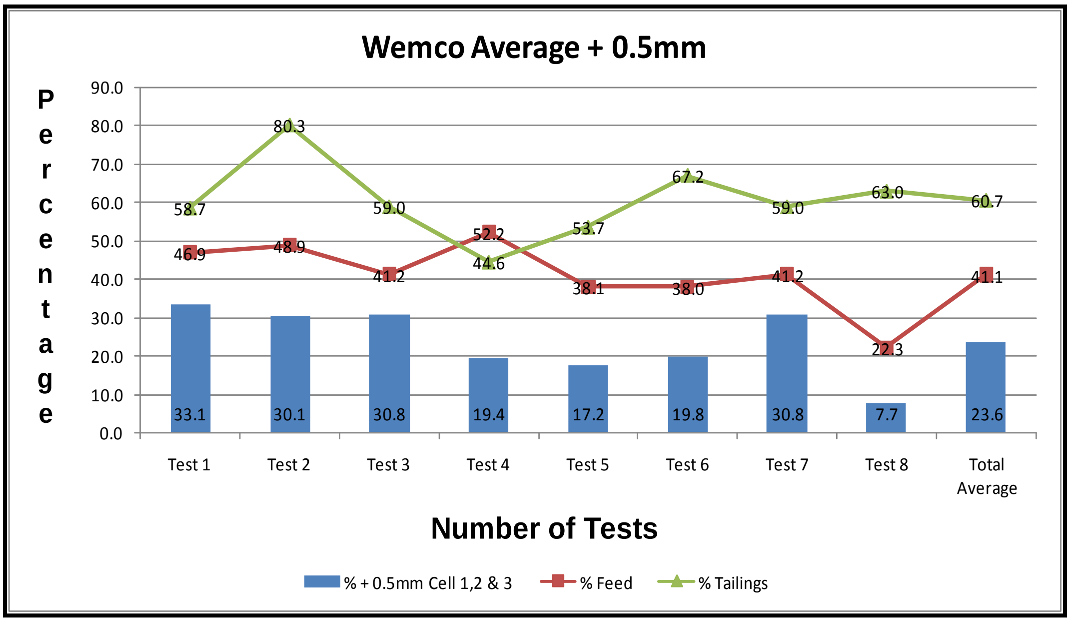

Figure 4.1: Wemco average + 0.5mm recovery

The average amount of + 0.5mm material present in the samples taken from the product launder per test is indicated by the blue bars in figure 4.1. The total average + 0.5mm material present over the eight tests is 23.6

percent (%) which is a fair amount considered the relatively old technology utilized for the recovery of the + 0.5mm material. The overall average + 0.5mm material present in the feed to the flotation circuit is 41.1% where 60.7% + 0.5mm were present in the flotation plant tailings.

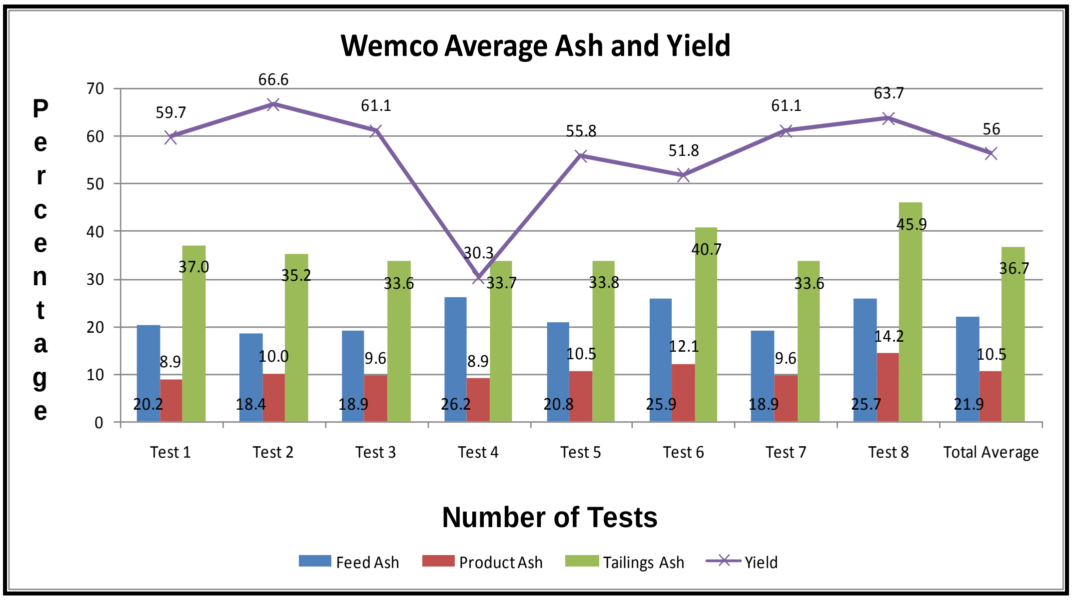

Figure 4.2: Wemco average ash and yield

The graph in figure 4.2 indicates the flotation plant feed, product and tailings ash during the eight test runs. The ash analysis was mainly conducted to compare the results obtained from the test runs with the results obtained with the Ultimate flotation cell. The feed ash will indicate the fairness of the comparison between the different types of flotation cells.

The product ash is a clear indication of the flotation cell's ability to achieve the target product ash of between 12% and 14%. It has been a major challenge during the lifespan of the beneficiation plant to produce a product with an ash in excess of 10.5% in the flotation circuit. This has only been achieved once during the eight tests conducted. The tailings ash percentage indicates the presence of product in the tailings if the ash is low. The tailings ash of Tshikondeni coal when producing a 12% ash product should be higher than 40%. It is therefore clear that some product is still present in the tailings where the ash content is only 36.7%. It also correlates with the results obtained from the laboratory bench flotation tests conducted at the end of every month as can be seen in figure 4.9. These tests indicated that there was between three and five percent product present in the flotation plant tailings monthly composite.

The yield bar is an indication of the yield obtained in the flotation circuit at the time of the tests. The overall yield was quite stable with test four standing out having the lowest percentage recovery. The feed ash percentage was also the highest, rendering it the most difficult feed beneficiated compared to the other seven tests. The product ash is very low at 8.9% and the tailings ash percentage of 33.7% is a clear indication of the presence of product in the tailings. The overall predicted flotation plant yield is between 60% and 70% depending on the quality of the feed. The average yield of 56% is four percent lower than the predicted minimum yield of 60% and support the results obtained in the laboratory indicating that there is between 3% and 5% product present in the flotation plant tailings.

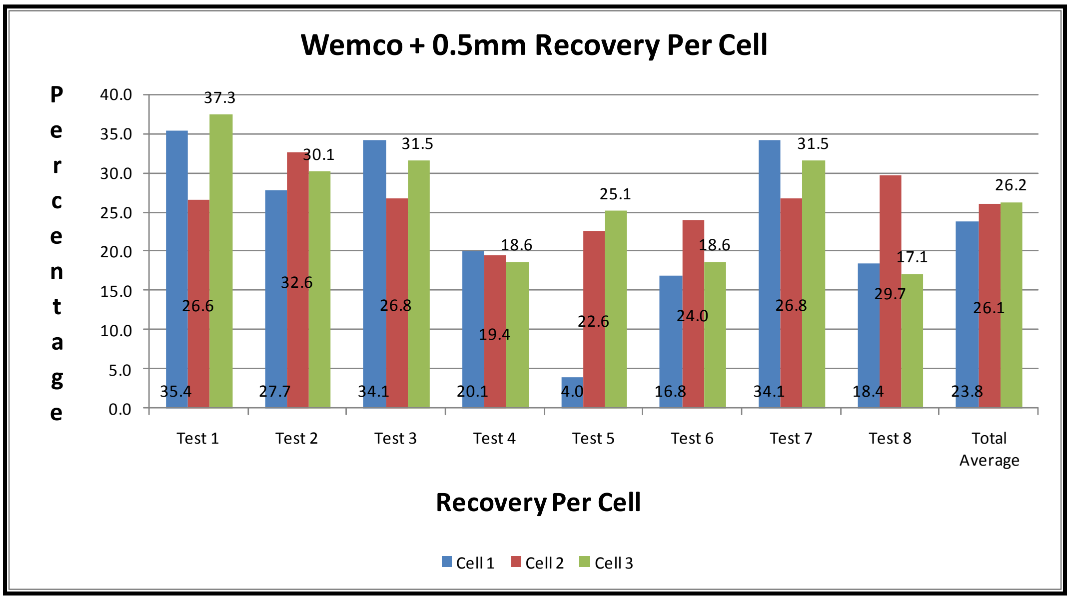

Figure 4.3: Wemco + 0.5mm recovery per cell

The recovery of the + 0.5mm fraction per flotation cell (cell one to cell three) is depicted in figure 4.3. There is a clear variance between the recoveries of each cell throughout all eight tests. The variance is mostly caused by the fluctuation of the feed quality depicted in figure 4.2. The recovery in flotation cell one is mostly affected when the coal is oxidised, influencing the coating of the particles by the collector as could be the case in test five. The overall average recovery of the eight test runs is however within a tolerance of 2.5% between the three cells. These results correlate with the fact that the speed of the rotors in all three cells is the same.

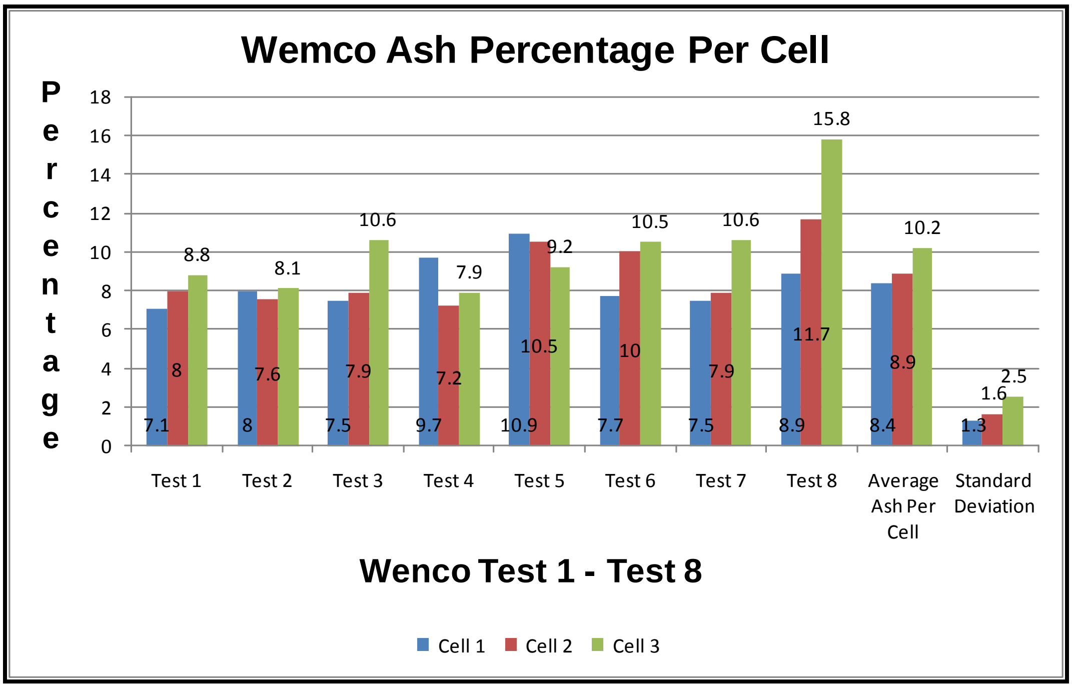

Figure 4.4: Wemco ash percentage per cell

The ash percentage of the product per cell can also be utilized in the comparison between the two different flotation cells. A higher ash percentage per cell will indicate a better recovery compared to the other make flotation cell. The results depicted in figure 4.4 were obtained during the Wemco flotation cell test runs with a 1.8% difference between cell number one and cell number three. This indicates that there is not a large difference in recovery between the three flotation cells which can also be seen in figure 4.3. The results obtained during the Ultimate flotation plant test runs will be compared to the results depicted in figure 4.1, figure 4.2, figure 4.3 and figure 4.4 which represents the Wemco test results.

4.1.2 Ultimate flotation plant test results and discussion

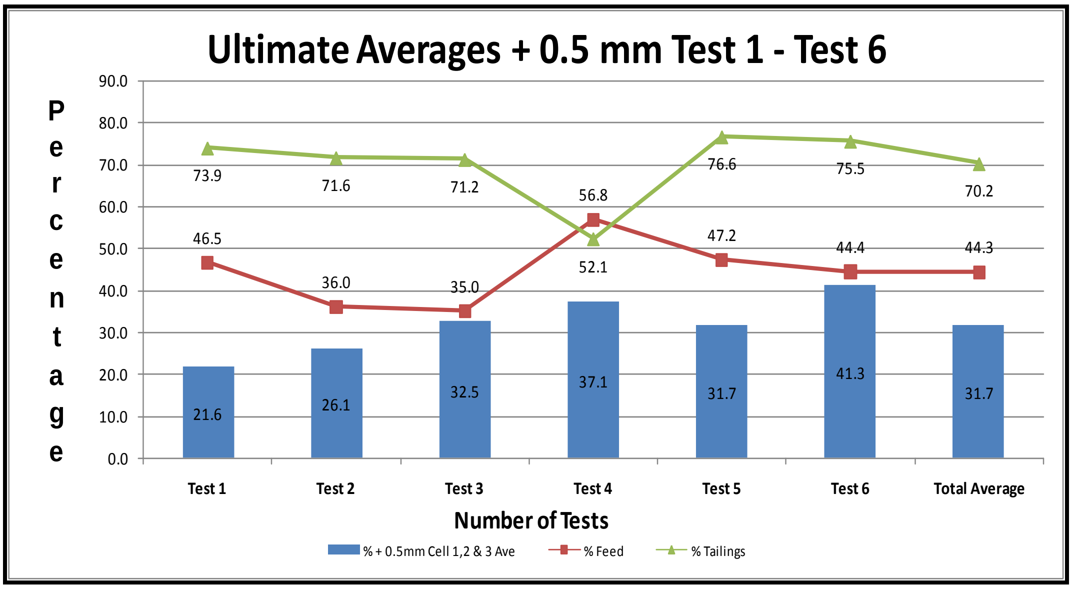

Figure 4.5: Ultimate averages + 0.5mm Test 1 – Test 6

The average + 0.5mm material recovered in the Ultimate flotation plant is illustrated in figure 4.5. The results obtained in test one and test two indicates that the flotation cell set-up during these two test runs was not at an optimal setting. It was the first two sets of samples taken from the Ultimate flotation plant after commissioning was completed. These two results are however close to the 23.6% average recovery obtained during the eight test runs in the old plant. A drastic improvement in recovery is noticeable from run three to run six with an overall average recovery of 31.7%. The average recovery of 31.7% obtained in the Ultimate flotation test is an 8.1% improvement on the average percentage of 23.6% obtained during the test conducted on the original Wemco flotation plant.

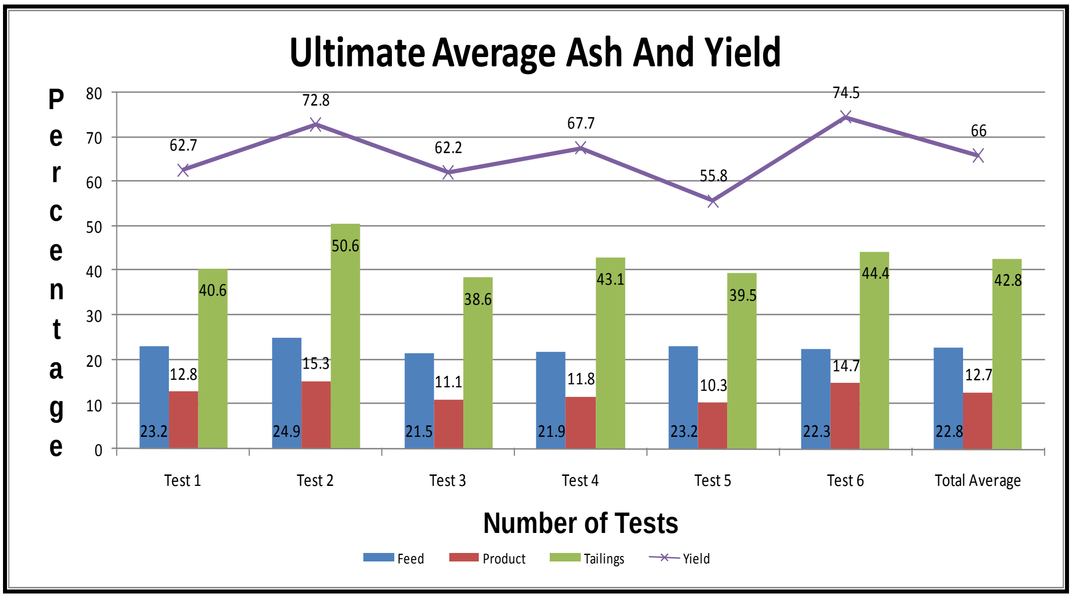

Figure 4.6: Ultimate average ash and yield

The feed ash of 22.8% as depicted in figure 4.6 is 0.9% higher than the 21.9% feed ash in the Wemco flotation test. The difference in the feed ash is quite small and will therefore not have a major effect on the yield comparison between the two flotation plants. The average product ash is however 2.2% higher for the Ultimate flotation test at 12%. This percentage is in line of the required ash of between 12% and 14% percent, indicating a better recovery. The discard ash is in line with the required < 40% at 42.8%. This is also a clear indication of an increase in recovery. The presence of product in the discard will cause the ash content of the discard to decrease to a level lower than 40% as is the case in the Wemco test runs.

The average yield of 66% is 10% higher than the average yield of 56% achieved during the Wemco tests. The higher product and tailings ash support the significant increase in yield achieved during the Ultimate tests. The actual average yield of 66% is 6% higher than the predicted minimum yield of 60% for the flotation plant. The yield was lower than 60% only in test run five at 55%. The reason for this being the frequency settings used during test run five. The settings in test run six proved to be the optimal settings as the highest yield was achieved. The product ash was however 0.7% higher than the highest required ash of 14% from the flotation plant. This deviation can easily be rectified by reducing the collector dosage slightly.

Figure 4.7 illustrates the + 0.5mm recovery in each flotation cell. It is noticeable that there is a variance between the recoveries per cell during the six test runs. The main reason for this being the different rotor frequency adjustments used in each test to determine the optimum frequency at which the individual flotation cells is to be operated at. The + 0.5mm recovery is the highest in cell number three in all the tests except in test three where the recovery in cell number one is more than cell number three.

When comparing the average recoveries achieved between the Wemco flotation cells and the Ultimate flotation cells, it is noticeable that there is an obvious difference between results of the three cells in the Ultimate test. The average recovery achieved per cell in the Wemco test as can be seen in figure 4.3 is very close to each other with the recovery in cell number one 2.4% lower than in cell number three. The average recovery in cell number one in the Ultimate plant is the lowest between the three cells, but almost as much as the highest recovery achieved in the Wemco plant which was 26.2%. The highest recovery in both plants were achieved in cell number three with the recovery in the Ultimate flotation plant 11.9% higher than what was achieved in the Wemco flotation plant. The Ultimate flotation plant recovery in cell number three is 11.3% higher than the recovery in cell number one. The reason for this phenomenon is the low frequency in cell number three as well as the absence of – 0.5mm material that was mostly recovered in cell number one and two.

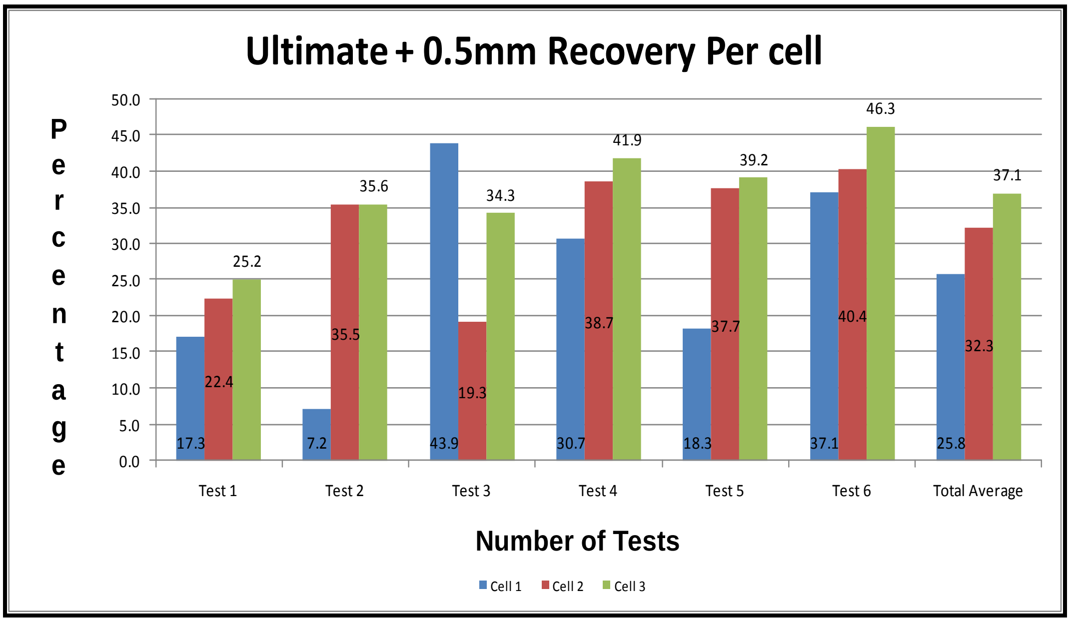

Figure 4.7: Ultimate + 0.5mm recovery per cell

The ash percentage of the product in each of the first three Ultimate flotation cells is depicted in figure 4.8. The average ash in cell number one is the same than the average ash in the Wemco cell number one. The yield in the Ultimate flotation cell one is higher than in the Wemco flotation cell one. The average product ash in the Ultimate cell number two is 0.5% higher than in the Wemco cell indicating a better recovery. The yield difference in the second cell is in favour of the Ultimate flotation cell by 6.2% verifying the difference in product ash. The ash difference in cell number three is the largest with the Ultimate flotation cell ahead by 1.8%. The recovery in the Ultimate flotation cell is 10.9% higher than the 26.2% achieved in the Wemco flotation cell. The standard deviation between the Ultimate flotation results and the Wemco flotation results is very much similar except in cell number one where the Ultimate flotation cell deviation is less than the Wemco flotation deviation. This is an indication that the Ultimate flotation cell is less sensitive for feed quality variance than is the case with the Wemco flotation cells.

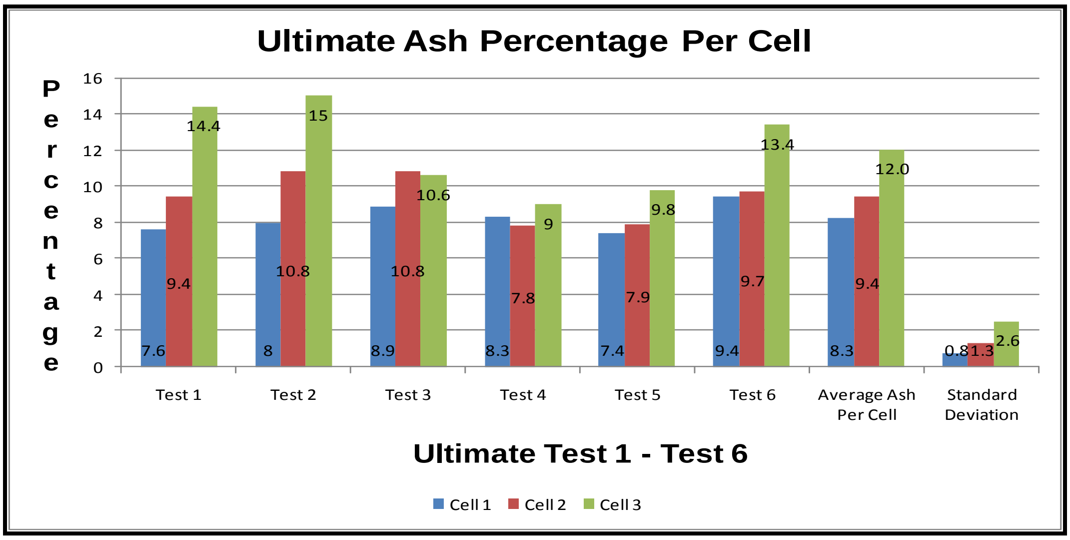

Figure 4.8: Ultimate ash percentage per cell

Ultimate + 0.5mm Recovery per Cell related to Hz						
	Cell 1 + 0.5mm	**Hz Cell 1**	**Cell 2 + 0.5mm**	**Hz Cell2**	**Cell 3 + 0.5mm**	**Hz Cell 3**
Test 1	17.3	35.0	22.4	32	25.2	30
Test 2	7.2	35.0	35.5	33	35.6	31
Test 3	43.9	33.0	19.3	35	34.3	31
Test 4	30.7	33.0	38.7	28	41.9	25
Test 5	18.3	30.0	37.7	26	39.2	22
Test 6	37.1	33.0	40.4	27	46.3	23
Total Average	**25.8**	**33.2**	**32.3**	**30.2**	**37.1**	**27.0**

Table 4.1: Ultimate recovery per cell related to Hertz

The data in table 4.1 indicate the recovery of the +0.5mm material in each Ultimate flotation cell at different frequencies. The frequencies were changed between the tests conducted to determine at which frequency each flotation cell should be operated to achieve maximum recovery. The best recoveries were achieved running cell number one at 33 Hz, cell number two at 27 Hz and cell number three at 23 Hz. The net effect is an increase in recovery of 11.9% which is in line with the predicted flotation plant yield.

The average air intake into the Ultimate flotation cells is 20m^3/h compared to 6.5m^3/h achieved in the Wemco flotation cells. This can be seen as the major contributor to the success of the Ultimate flotation cell as the air intake of 20m^3/h is achieved at much lower rotor speeds as indicated in table 4.2 and table 4.3. No sanding occurs even at very low rotor speeds and water surface disruptions is also absent at all times. The coarse particles are suspended into the bubble attachment zone without being influenced by water streams caused by excessive rotor speeds. The average Wemco rotor speed is 332 revolutions per minute (rpm) and the Ultimate average rotor speed is 174 rpm, which is 47.6% slower than the Wemco rotor speed. Operating at slower rotor speeds will also cause much less wear on internal flotation cell parts which will increase the overall live of the flotation equipment.

Wemco Rotor Speed & Power Consumption					
	Frequency	Motor speed	Rotor speed	Kw	Air intake
Cell 1	50.0	1450	332.0	18.5	6.5
Cell 2	50.0	1450	332.0	18.5	6.5
Cell 3	50.0	1450	332.0	18.5	6.5
Average	**50.0**	**1450.0**	**332.0**	**18.5**	**6.5**

Table 4.2: Wemco rotor speed and power consumption

Ultimate Rotor Speed & Power Consumption					
	Frequency	Motor speed	Rotor speed	Kw	Air intake
Cell 1	33.0	646	207.3	11.6	23.0
Cell 2	27.0	530	170.1	9.5	20.0
Cell 3	23.1	453	145.4	8.2	17.0
Average	**27.7**	**543.0**	**174.3**	**9.8**	**20.0**

Table 4.3: Ultimate rotor speed and power consumption

4.2 Monthly flotation tailings results

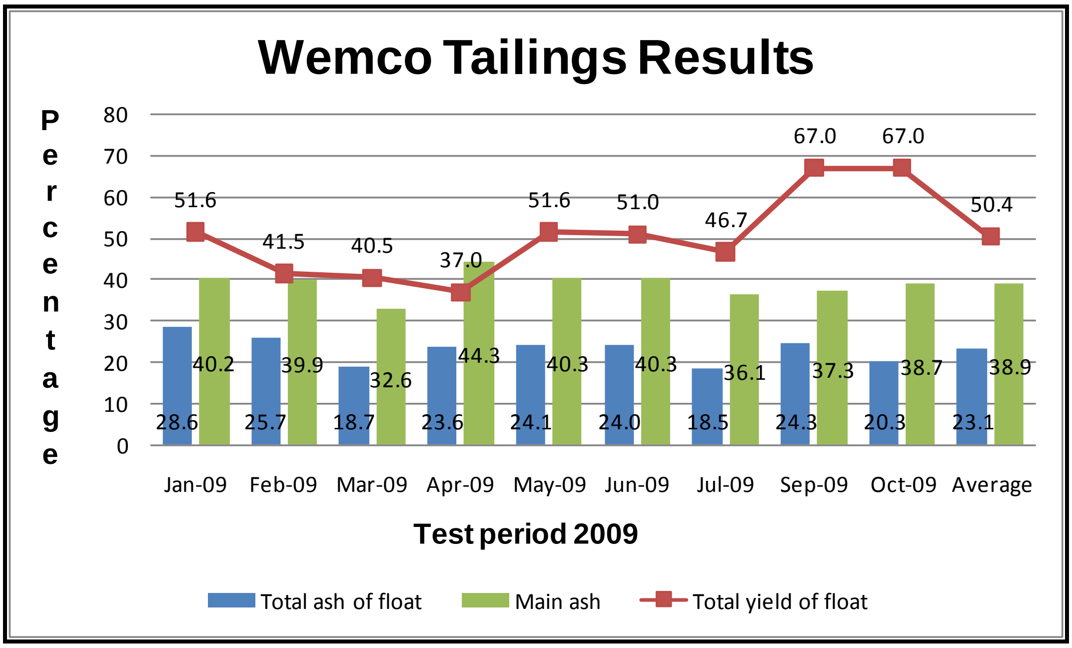

Figure 4.9: Wemco monthly tailings composite results

A monthly composite sample is compiled from the flotation plant tailings and used in a laboratory bench flotation test to determine the amount of product in the tailings. The test is conducted utilizing a Denver laboratory scale flotation cell at the same reagents dosage utilized in the actual flotation plant with the results for the Wemco flotation plant depicted in figure 4.9. The average product yield obtained is 50.4% at an ash content of 23.1%. The average Wemco flotation plant tailings ash (laboratory test feed ash) is 38.9%, which is in line with the 36.7% average tailings ash obtained during the eight Wemco tests conducted.

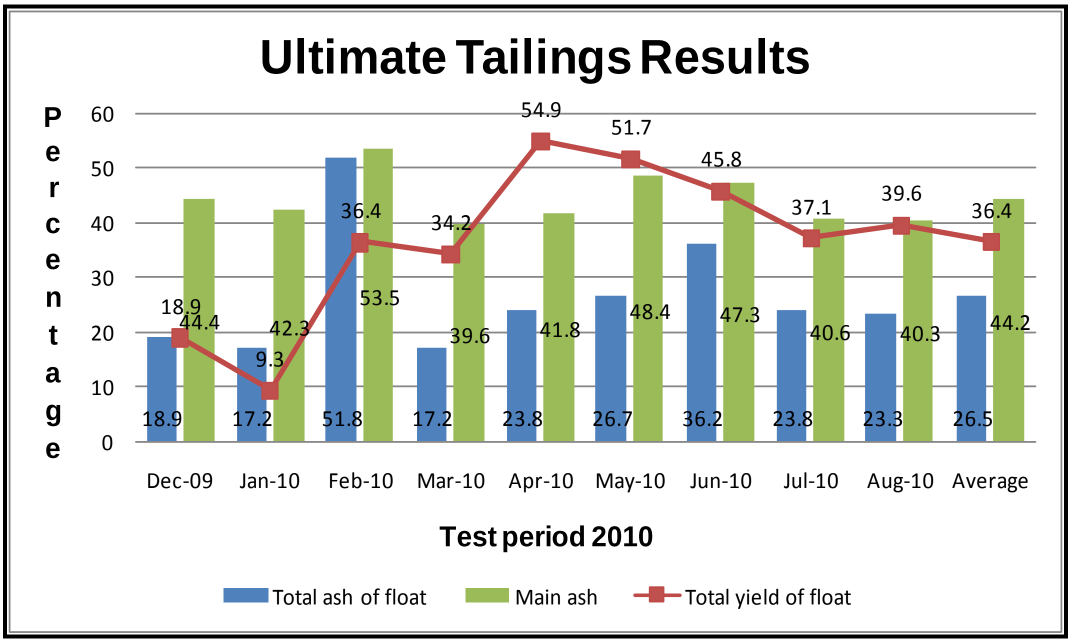

Figure 4.10: Ultimate monthly tailings composite results

The average product yield obtained from the Ultimate flotation plant tailings is 36.4% which is 14% lower than the yield obtained during the Wemco tailings tests indicating that there is 14% less product in the Ultimate flotation plant tailings. The average product ash obtained from the Ultimate flotation plant composites is 3.4% higher than the Wemco flotation plant product ash. This is also an indication that less product is present in the Ultimate flotation plant tailings. The Ultimate flotation plant's average monthly tailings ash (laboratory test feed ash) of 44.2% is 5.3% higher than the Wemco flotation plant's average monthly tailings ash. The tailings ash obtained from the six Ultimate test runs is also similar to the monthly composite average ash. It can therefore be accepted that the test results obtained during the project, are accurate.

4.3 Energy consumption

The reduction of energy consumption is the talk of the decade. Each mining company had to commit to an energy consumption saving of no less than 10%. The target for Tshikondeni mine is also 10% which have to be achieved in order to prevent the mine from paying penalties. This was one of the major reasons for excluding the installation of forced air flotation cells which will need a compressor plant to generate the air needed for injection into the flotation cells. The reduction in rotor speed decreased the average power consumption of the Ultimate flotation plant to a level of 47% less than the Wemco flotation plant. The power consumption per cell as well as the average power consumption for the Wemco and Ultimate flotation cell is illustrated in table 4.2 and table 4.3.

It must be noted that the rotor speed for the Wemco flotation plant cannot be adjusted and that the rotors were changed from a six blade rotor to an eight blade rotor in the attempt to increase the recovery of the + 0.5mm material. Power consumption also increased from an average consumption of 15kw to 18,5kw per cell. With a power consumption reduction of 47% in the Ultimate flotation plant, the 10% target for the beneficiation plant is much easier to achieve.

CHAPTER 5: Conclusions and recommendations

This chapter conclude the project findings and recommends the way forward for flotation as a beneficiation process in the South African coal industry.

5.1 Conclusion

The main aim of this project was to improve the recovery of the – 1.4mm + 0.5mm material in the Tshikondeni flotation plant circuit at the lowest possible cost. It is important to note that the Wemco Smart Cell flotation plant is relatively old technology installed in the mid 1990's. It is therefore also important to note that the results obtained from the Wemco flotation plant were only used as a baseline to determine the recovery improvement achieved utilizing the new technology Ultimate Flotation cells.

The average yield achieved in the Ultimate flotation plant is 10% higher than the average yield achieved in the Wemco flotation plant indicating a tremendous improvement in the recovery of the +0.5mm material. The actual recovery improvement of the +0.5mm material is 8.1% from 23.6% recovered in the Wemco flotation cells to 31.7% recovered in the Ultimate flotation cells.

The flotation product ash improved from an average of 10.5% to an average of 12.7% with the latter achieved in the Ultimate flotation cells. Rotor speeds are much lower than that for the Wemco plant and can be adjusted to ensure optimal recovery. The adjustable product launders were lowered to be 10mm lower than the flotation cell lip improving the flow of the froth into the product launder. The installation of the Ultimate flotation cell in the Tshikondeni plant was successful in many ways. The plant was started during commissioning without any problems and turned out to be a pleasure to operate. Power consumption decreased by 47% and the overall flotation plant yield increased by 10% which is 5% better than the expected yield increase for the project. The main aim of the project was to improve the +0.5mm recovery in the Tshikondeni flotation plant and was achieved by 8.1%. The collector consumption in the flotation plant was also reduced from 200ml/min to 180ml/min resulting in a cost saving spin off which was not initially expected. The Ultimate flotation cell was initially designed to beneficiate a different mineral, but proved to be a valuable asset to the Tshikondeni beneficiation plant.

5.2 Recommendation

The first three flotation cells in module A and module B were replaced in the Tshikondeni beneficiation plant with Ultimate flotation cells. It is currently the only coal beneficiation plant in the world utilizing Ultimate flotation cells to beneficiate the – 1.4mm coal fraction in the feed to plant. The Ultimate flotation cell was originally designed for the copper industry and the rotor mechanism was tested by chance in the Tshikondeni coal flotation plant. The way forward will be to replace the fourth cell in each module and take samples of the tailings. Analysis should be done to determine the presence of product in the tailings. The overall feeling is that the flotation plant can be reduced from six flotation cells per module to four flotation cells per module beneficiating the same amount of coal as in the current six cell plant. The result will be a decrease in maintenance costs as well as a further reduction in energy consumption.

A large scale ($6m^3$) mobile Ultimate flotation cell should be developed for test work in other coal fields. There is currently millions of tons ultra fine and fine coal stored in slimes dams all over South African coal fields. The opportunity is there to produce a low ash product as well as a middling product from these slimes dams. Filtration of fine material has developed in leaps during the last decade and can be used in conjunction with flotation cells or fines DMS plants to produce a handleable and marketable product. Fine coal granulation and briquetting can be utilized to improve the handlability of the ultra fine and fine coal products. Most of these slimes dams are a legal liability as well as an environmental hazard, which can be reduced drastically by removing the coal particles entrapped in them.

References

Bailey, R and Wheler, P.F. 1953. "The influence of pulp temperature on froth flotation of four British fine coals." Journal of the institute of fuel, p 304 – 307, January.

Beafore, F.J., Cawiezel, K.E., and Montgomery, C.T. 1979. "Oxidized coal – What it is and how it affects your preparation plant performance." Paper delivered on symposium of coal preparation and utilization, p 1 – 9.

Briscoe, A and Vander Veen, R. 1990. Factors influencing the choice of coal flotation reagents, *Mine and Quarry*, January 1990, p 65 – 67.

De Korte, J and McGonigal, S. Project 4.11.1 Dense – Medium Beneficiation of fine coal: Coaltech 2020 Pilot Plant at Navigation, Landau Colliery. 2005 CSIR

Falcon, R.M.S. "A brief review of the origin, formulation and distribution of coal in Southern Africa." Mineral Deposits of Southern Africa, p 1879 – 1898, (1986)

Fuerstenau, M.C. 1976. "Flotation" *The American Institute of Mining, Metallurgical and Petroleum Engineers, Inc.* A.M.Gaudin Memorial Volume, 2 p 1235 – 1259.

Fuerstenau, D.W., Yang, G.C.C., and Laskowski, J.S. 1987. "Oxidation phenomena in coal flotation." Part 1. Correlation between oxygen functional group concentration, immersion, wettability and salt flotation response. Coal Preparation, 4: p 161 – 182.

Jenkins, P.R., The Glasdir Experiment, Dragon Wheel Books, 1996.

Kawatra,S.K. Froth flotation. Fundamental Principles. INTERNET. http://www.chem.mtu.edu.

Mining and environmental geology. INTERNET. http://www.geologydata.blogspot.com/.

Nimerick, K.H. and Scott, B.E. 1980. New method of oxidised coal flotation. *Mining Congress Journal,* 66: p 21 – 22.

Powell, D.M. 1999. "A process model for a free jet column cell" (M.Sc – dissertation, university of Witwatersrand).

Sarker, G.G., Ghose, S., Chaudhuri, S.G., Sakha, S., and Daschowdhury, K.P. 1984. "Selectivity of coal macerals during flotation and oil agglomeration: A case study." Coal Preparation, 1: p 35 – 52.

Sun, S.C. 1954. “Effect of oxidation of coals on their flotation properties.” *Trans AIME*, Vol 199: p 396 – 401.

Sun, S.C., and Zimmerman, R.F. 1950. “The mechanism of coarse coal and mineral froth flotation.” *Trans AIME.,* Vol 187: p 516 – 522.

Van der Merwe, D. Tshikondeni Fine Coal Dense Medium Cyclone evaluation. 2007.

Velela. Abercwmboi (1976). INTERNET. http:// en.wikipedia.org/wiki/file: coke_ovens_Abercwmboi.

Viljoen, H. 2009. Personal communication.

Wheeler, T.A. and Keys, R.D. 1986. Factors influencing the choice of coal flotation reagents. INTERNET. http://www.onemine.org/

Wilkes, K.D. 2008. Development and recent applications of Wemco HMS and Flotation cells. INTERNET. http://www.coalresearchforum.org.

Wills, B.A., Napier – Munn, T.J., 2006. “Froth Flotation.” <u>Mineral Processing Technology. Seventh edition, Oxford, Great Britain, Elsevier.</u>

Attachments

Attachment A: Petrographic report

CHEMICAL AND PETROGRAPHIC PROPERTIES OF THE MONTHLY COMPOSITE SAMPLE FROM TSHIKONDENI, JUNE 2010

PROPERTIES		P_c	2010-06	P_N
Chemical Properties				
Moisture (adb)	%	0.4	0.5	0.5
Ash (db)	%	13.8	13.6	13.8
Volatile Matter (db)	%	22.4	20.5	21.6
Total Sulphur (db)	%	0.76	0.77	0.78
Pyritic Sulphur	%	0.14	0.11	0.13
Sulphatic Sulphur	%	0.02	0.02	0.02
Organic Sulphur	%	0.60	0.64	0.63
Carbon (db)	%	77.65	78.30	77.55
Hydrogen (db)	%	4.28	4.00	4.04
Nitrogen (db)	%	1.92	1.71	1.82
Oxygen (db)	%	1.59	1.66	2.07
Fixed Carbon (calc., adb)	%	63.6	65.6	64.4
Gross Cal. Val. (adb)	M	31.2	31.4	31.31
Free Swelling Index	J	9.0	9.0	9.0
Roga Index	/	92	89	91
	k			
Petrographic Properties:	g			
Maceral Composition:	%			
Vitrinite	%	79.3	80.6	80.0
Liptinite (Exinite)				
Reactive Semifusinite		3.2	2.7	3.0
Inertinite		9.9	9.1	9.4
Mineral Matter	%	7.6	7.6	7.6
	%			
Vitrinite Reflectance Classes:	%			
V9 (0.90 to 0.99)	%			
V10 (1.00 to 1.09	%	2 (1.6)	2 (1.6)	2 (1.6)
V11 (1.10 to 1.19)		17 (13.5)	16 (12.9)	20 (16.0)
V12 (1.20 to 1.29)		28 (22.2)	13 (10.5)	18 (14.4)
V13 (1.30 to 1.39)	%	11 (8.8)	9 (7.4)	7 (5.7)
V14 (1.40 to 1.49)	%	12 (10.0)	26 (21.4)	19 (15.8)
V15 (1.50 to 1.59)	%	17 (14.4)	11 (9.3)	14 (11.7)
V16 (1.60 to 1.69)	%	7 (5.6)	15 (12.1)	12 (9.8)
V17 (1.70 to 1.79)	%	2 (1.6)	2 (1.6)	3 (3.0)
V18 (1.80 to 1.89)	%	2 (1.6)		1 (1.2)
V19+ (1.90 and higher)	%	2	6	4
	%			
Petrographic Parameters:	%			
RoV (max)	%	1.38	1.46	1.42
RoR	%	1.37	1.40	1.39
Total Reactives		79.3	76.8	79.2
Total Inerts		20.7	23.2	20.8
Optimum Inerts	%	15.2	12.4	13.6
Composition Balance Index	%	1.36	1.87	1.53
	%			
Predicted Drum Indices:	%			
M10 Index	%	6.9	7.7	7.2
M40 Index		>75	>75	>75
I10 Index		19.3	20.6	19.8
120 Index		78.0	76.6	77.5

P_c: Current mean values in prediction model (April and May 2010 samples).
P_N: New mean values (May and June 2010 samples).
(): Distribution of reactive in reflectance classes.

RHEOLOGICAL AND COAL ASH PROPERTIES OF THE MONTHLY COMPOSITE SAMPLE FROM TSHIKONDENI, JUNE 2010

PROPERTIES		P_c	2010-06	P_N
Rheological Properties				
Dilatation:				
Softening Temp.	º	385	398	388
Maximum Contract. Temp.	C	420	430	424
Maximum Dilation Temp.	º	491	499	497
Maximum Contraction	C	29	27	29
Maximum Dilatation	º	176	150	173
Amplitude	C	205	177	202
	%			
Gieseler Fluidity:	%			
Initial Softening Temp.	%	402	416	407
Maximum Fluidity Temp.		469	469	467
Resolidification Temp.		507	509	508
Maximum Fluidity	º	4 426	1 928	3 923
	C			
Coal Ash Properties:	º			
Ash Composition:	C			
SiO_2	º	57.42	56.33	56.82
Al_2O_3	C	25.78	25.39	25.66
Fe_2O_3	d	4.19	4.36	4.26
T_iO_2	d	1.51	1.50	1.49
P_2O_5	p	0.46	0.52	0.51
CaO	m	3.80	3.46	3.53
MgO		1.40	1.43	1.29
Na_2O		0.59	0.59	0.62
K_2O		1.64	1.61	1.61
SO_3	%	1.23	2.84	2.29
MnO	%	0.04	0.03	0.04
Ba	%	0.23	0.23	0.23
Sr	%	0.14	0.15	0.14
V_2O_5	%	0.04	0.05	0.05
Cr_2O_3	%	0.02	0.04	0.03
ZrO_2	%	0.10	0.09	0.09
Total	%	98.59	98.62	98.66
	%			
Ash Indices:	%			
Base/ Acid Ratio	%	0.143	0.144	0.141
Slagging Index	%	0.108	0.111	0.110
Fouling Index	%	0.084	0.086	0.088
Total Coal Alkali	%	0.231	0.225	0.231
Silica Ratio	%	85.97	85.89	86.22
	%			
Ash Fusion Temperatures:	%			
Initial Temp.		1 404	1 386	1 407
Softening Temp.		1 419	1 392	1 413
Hemisphere Temp.		1 447	1 478	1 475
Flow Temp.		1 486	1 502	1 500
	º			
	C			
	º			
	C			
	º			
	C			
	º			
	C			

P_c: Current mean values in prediction model (April and May 2010 samples).

P_N: New mean values (May and June 2010 samples).

Attachment B: Wemco Test 1 to Test 10 results

Table B1

DATE: 01/09/2009			
Wemco Test 1			
Sample ID	**WET MASSES (g)**	**DRY MASSES(g)**	**ASH %**
Flotation Feed	7557	687	20.2
Flotation Cell 1	8760	2975	7.1
Flotation Cell 2	10333	2448	8
Flotation Cell 3	12237	3455	8.8
Flotation Cell 4	10987	3489	9.9
Flotation Cell 5	11028	2528	9.7
Flotation Cell 6	11001	1830	9.9
Tailings	12112	365	37
SCREEN ANALYSIS		Product Ash	8.9
Flotation Feed			
SIZE (mm)	**FRAC MASS (g)**	**FRAC MASS %**	**CUM MASS %**
2.8	4.0	0.6	0.0
2.0	16.0	2.4	2.4
1.4	75.0	11.3	13.7
1.0	84.0	12.7	26.4
0.85	30.0	4.5	30.9
0.5	106.0	16.0	46.9
0.3	75.0	11.3	58.2
0.212	63.0	9.5	67.7
(-)0.212	210.3	31.7	99.4
INPUT	663.3	100.0	
Tailings			
SIZE (mm)	**FRAC MASS (g)**	**FRAC MASS %**	**CUM MASS %**
2.8	7.0	2.1	0.0
2.0	12.6	3.8	3.8
1.4	58.5	17.5	21.3
1.0	55.0	16.5	37.7
0.85	19.0	5.7	43.4
0.5	51.0	15.3	58.7
0.3	30.0	9.0	67.6
0.212	21.0	6.3	73.9
(-)0.212	80.2	24.0	97.9
INPUT	334.3	100.0	

Flotation Cell 1			
SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	0.0	0.0	0.0
2	0.0	0.0	0.0
1.4	35.0	1.2	1.2
1	174.0	5.9	7.1
0.85	69.0	2.3	9.4
0.5	766.0	26.0	35.4
0.3	508.0	17.2	52.6
0.212	277.0	9.4	62.0
(-)0.212	1119.2	38.0	100.0
INPUT	**2948.2**	**100.0**	

Flotation Cell 2			
SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	0.0	0.0	0.0
2	0.0	0.0	0.0
1.4	15.0	0.6	0.6
1	95.0	3.9	4.5
0.85	56.0	2.3	6.9
0.5	479.0	19.8	26.6
0.3	392.0	16.2	42.8
0.212	348.0	14.4	57.2
(-)0.212	1036.4	42.8	100.0
INPUT	**2421**	**100**	

Flotation Cell 3			
SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	0.0	0.0	0.0
2	0.0	0.0	0.0
1.4	35.0	1.4	1.4
1	167.0	6.6	8.0
0.85	75.0	3.0	10.9
0.5	668.0	26.4	37.3
0.3	455.0	18.0	55.3
0.212	332.0	13.1	68.4
(-)0.212	798.6	31.6	100.0
INPUT	**2530.6**	**100.0**	

Table B2

DATE: 06/09/2009			
Wemco Test 2			
Sample ID	**WET MASSES (g)**	**DRY MASSES(g)**	**ASH %**
Flotation Feed	**17606**	**1640**	**18.4**
Flotation Cell 1	**20036**	**3872**	**8**
Flotation Cell 2	**16140**	**3895**	**7.6**
Flotation Cell 3	**14519**	**4450**	**8.1**
Flotation Cell 4	**13893**	**3842**	**10.2**
Flotation Cell 5	**11739**	**2625**	**14.5**
Flotation Cell 6	**11204**	**2573**	**14.7**
Tailings	**13591**	**974**	**35.2**
SCREEN ANALYSIS		**Product Ash**	**10.0**

Flotation Feed

SIZE (mm)	**FRAC MASS (g)**	**FRAC MASS %**	**CUM MASS %**
2.8	8.3	0.5	0.0
2.0	20.2	1.2	1.2
1.4	134.8	8.3	9.6
1.0	213.4	13.2	22.8
0.85	52.0	3.2	26.0
0.5	369.3	22.8	48.9
0.3	172.3	10.7	59.5
0.212	144.2	8.9	68.4
(-)0.212	501.9	31.1	99.5
INPUT	**1616.4**	**100.0**	

Tailings

SIZE (mm)	**FRAC MASS (g)**	**FRAC MASS %**	**CUM MASS %**
2.8	15.1	1.6	0.0
2.0	42.2	4.5	4.5
1.4	226.4	23.9	28.4
1.0	251.9	26.6	55.0
0.85	61.7	6.5	61.5
0.5	177.3	18.7	80.3
0.3	63.3	6.7	86.9
0.212	28.2	3.0	89.9
(-)0.212	80.2	8.5	98.4
INPUT	**946.3**	**100.0**	

Flotation Cell 1			
SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	0.0	0.0	0.0
2	0.0	0.0	0.0
1.4	112.5	2.9	2.9
1	513.7	13.4	16.3
0.85	262.9	6.8	23.1
0.5	178.0	4.6	27.7
0.3	640.4	16.7	44.4
0.212	485.6	12.6	57.0
(-)0.212	1652.4	43.0	100.0
INPUT	**3845.5**	**100.0**	

Flotation Cell 2			
SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	0.0	0.0	0.0
2	0.0	0.0	0.0
1.4	21.8	0.6	0.6
1	251.8	6.5	7.1
0.85	85.6	2.2	9.3
0.5	901.8	23.3	32.6
0.3	88.0	2.3	34.8
0.212	466.8	12.1	46.9
(-)0.212	2056.9	53.1	100.0
INPUT	**3873**	**100**	

Flotation Cell 3			
SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	0.0	0.0	0.0
2	0.0	0.0	0.0
1.4	30.0	0.7	0.7
1	244.9	5.5	6.2
0.85	127.4	2.9	9.1
0.5	930.4	21.0	30.1
0.3	878.4	19.9	50.0
0.212	446.4	10.1	60.1
(-)0.212	1767.0	39.9	100.0
INPUT	**4424.5**	**100.0**	

Table B3

DATE: 15/09/2009			
Wemco Test 3			
Sample ID	**WET MASSES (g)**	**DRY MASSES(g)**	**ASH %**
Flotation Feed	17938	2140	18.9
Flotation Cell 1	14557	4489	7.5
Flotation Cell 2	14071	4422	7.9
Flotation Cell 3	12669	3708	10.6
Flotation Cell 4	14231	3805	10.4
Flotation Cell 5	11494	2539	11.7
Flotation Cell 6	12000	2737	11
Tailings	15614	1036	33.6
SCREEN ANALYSIS		Product Ash	9.6
Flotation Feed			
SIZE (mm)	**FRAC MASS (g)**	**FRAC MASS %**	**CUM MASS %**
2.8	8.8	0.4	0.0
2.0	19.8	0.9	0.9
1.4	131.8	6.2	7.2
1.0	220.0	10.4	17.6
0.85	71.0	3.4	21.0
0.5	428.0	20.3	41.2
0.3	253.3	12.0	53.2
0.212	189.5	9.0	62.2
(-)0.212	788.7	37.4	99.6
INPUT	**2110.9**	**100.0**	
Tailings			
SIZE (mm)	**FRAC MASS (g)**	**FRAC MASS %**	**CUM MASS %**
2.8	4.9	0.5	0.0
2.0	26.3	2.6	2.6
1.4	115.7	11.4	14.0
1.0	205.6	20.3	34.3
0.85	49.0	4.8	39.2
0.5	200.8	19.8	59.0
0.3	33.9	3.3	62.4
0.212	91.5	9.0	71.4
(-)0.212	284.7	28.1	99.5
INPUT	**1012.4**	**100.0**	

Flotation Cell 1			
SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	0.0	0.0	0.0
2	0.0	0.0	0.0
1.4	30.4	0.7	0.7
1	230.4	5.2	5.9
0.85	139.9	3.1	9.0
0.5	1116.0	25.1	34.1
0.3	790.0	17.7	51.8
0.212	636.8	14.3	66.1
(-)0.212	1509.2	33.9	100.0
INPUT	**4452.7**	**100.0**	

Flotation Cell 2			
SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	0.0	0.0	0.0
2	0.0	0.0	0.0
1.4	15.0	0.3	0.3
1	153.3	3.5	3.8
0.85	133.5	3.0	6.9
0.5	872.2	19.9	26.8
0.3	1052.7	24.0	50.7
0.212	438.7	10.0	60.7
(-)0.212	1723.1	39.3	100.0
INPUT	**4389**	**100**	

Flotation Cell 3			
SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	0.0	0.0	0.0
2	0.0	0.0	0.0
1.4	29.5	0.8	0.8
1	186.0	5.1	5.9
0.85	142.9	3.9	9.7
0.5	800.0	21.7	31.5
0.3	723.6	19.7	51.1
0.212	250.2	6.8	57.9
(-)0.212	1547.3	42.1	100.0
INPUT	**3679.5**	**100.0**	

Table B4

DATE: 21/09/2009			
Wemco Test 4			
Sample ID	**WET MASSES (g)**	**DRY MASSES(g)**	**ASH %**
Flotation Feed	**18389**	**1970**	**26.2**
Flotation Cell 1	**18728**	**3366**	**9.7**
Flotation Cell 2	**15466**	**4554**	**7.2**
Flotation Cell 3	**12480**	**3587**	**7.9**
Flotation Cell 4	**18549**	**4194**	**9.6**
Flotation Cell 5	**12473**	**2712**	**9.6**
Flotation Cell 6	**14530**	**3028**	**10.5**
Tailings	**15656**	**2335**	**33.7**
SCREEN ANALYSIS		**Product Ash**	**8.9**

Flotation Feed

SIZE (mm)	**FRAC MASS (g)**	**FRAC MASS %**	**CUM MASS %**
2.8	10.4	0.5	0.0
2.0	26.6	1.4	1.4
1.4	115.9	6.0	7.3
1.0	214.2	11.0	18.4
0.85	62.3	3.2	21.6
0.5	594.5	30.6	52.2
0.3	225.0	11.6	63.7
0.212	298.6	15.4	79.1
(-)0.212	395.5	20.4	99.5
INPUT	**1943.0**	**100.0**	

Tailings

SIZE (mm)	**FRAC MASS (g)**	**FRAC MASS %**	**CUM MASS %**
2.8	11.2	0.5	0.0
2.0	39.0	1.7	1.7
1.4	173.0	7.5	9.2
1.0	289.9	12.5	21.7
0.85	118.8	5.1	26.9
0.5	410.0	17.7	44.6
0.3	230.9	10.0	54.6
0.212	269.8	11.7	66.3
(-)0.212	768.0	33.2	99.5
INPUT	**2310.6**	**100.0**	

Flotation Cell 1			
SIZE (mm)	**FRAC MASS (g)**	**FRAC MASS %**	**CUM MASS %**
2.8	0.0	0.0	0.0
2	0.0	0.0	0.0
1.4	19.2	0.6	0.6
1	129.4	3.9	4.5
0.85	80.8	2.4	6.9
0.5	440.3	13.2	20.1
0.3	391.4	11.7	31.8
0.212	264.9	7.9	39.8
(-)0.212	2008.3	60.2	100.0
INPUT	**3334.3**	**100.0**	

Flotation Cell 2			
SIZE (mm)	**FRAC MASS (g)**	**FRAC MASS %**	**CUM MASS %**
2.8	0.0	0.0	0.0
2	0.0	0.0	0.0
1.4	16.0	0.4	0.4
1	135.9	3.0	3.4
0.85	71.8	1.6	4.9
0.5	655.0	14.5	19.4
0.3	348.4	7.7	27.1
0.212	716.0	15.8	42.9
(-)0.212	2582.8	57.1	100.0
INPUT	**4526**	**100**	

Flotation Cell 3			
SIZE (mm)	**FRAC MASS (g)**	**FRAC MASS %**	**CUM MASS %**
2.8	0.0	0.0	0.0
2	0.0	0.0	0.0
1.4	7.5	0.2	0.2
1	132.7	3.7	3.9
0.85	51.9	1.5	5.4
0.5	470.1	13.2	18.6
0.3	564.5	15.9	34.5
0.212	515.4	14.5	49.0
(-)0.212	1811.8	51.0	100.0
INPUT	**3553.9**	**100.0**	

Table B5

DATE: 21/09/2009			
Wemco Test 5			
Sample ID	WET MASSES (g)	DRY MASSES(g)	ASH %
Flotation Feed	16377	2286	20.8
Flotation Cell 1	18135	2680	10.9
Flotation Cell 2	15793	5064	10.5
Flotation Cell 3	15116	4499	9.2
Flotation Cell 4	16563	4060	10.7
Flotation Cell 5	13563	2130	9.5
Flotation Cell 6	13040	6380	11.5
Tailings	18117	998	33.8
SCREEN ANALYSIS		Product Ash	10.5
Flotation Feed			
SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	7.0	0.3	0.0
2.0	20.1	0.9	0.9
1.4	115.3	5.1	6.0
1.0	238.0	10.5	16.5
0.85	61.3	2.7	19.3
0.5	424.6	18.8	38.1
0.3	341.8	15.1	53.2
0.212	267.0	11.8	65.0
(-)0.212	783.0	34.7	99.7
INPUT	2258.1	100.0	
Tailings			
SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	2.2	0.2	0.0
2.0	16.4	1.7	1.7
1.4	93.6	9.6	11.3
1.0	180.7	18.5	29.8
0.85	51.0	5.2	35.1
0.5	181.5	18.6	53.7
0.3	101.6	10.4	64.1
0.212	49.7	5.1	69.2
(-)0.212	298.0	30.6	99.8
INPUT	974.7	100.0	

Flotation Cell 1			
SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	0.0	0.0	0.0
2	0.0	0.0	0.0
1.4	0.0	0.0	0.0
1	18.7	0.7	0.7
0.85	10.4	0.4	1.1
0.5	75.4	2.9	4.0
0.3	223.0	8.4	12.4
0.212	371.9	14.1	26.4
(-)0.212	1946.7	73.6	100.0
INPUT	**2646.1**	**100.0**	

Flotation Cell 2			
SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	0.0	0.0	0.0
2	0.0	0.0	0.0
1.4	12.4	0.2	0.2
1	154.8	3.1	3.3
0.85	111.7	2.2	5.5
0.5	860.4	17.1	22.6
0.3	1096.2	21.8	44.4
0.212	516.5	10.3	54.7
(-)0.212	2283.4	45.3	100.0
INPUT	**5035**	**100**	

Flotation Cell 3			
SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	0.0	0.0	0.0
2	0.0	0.0	0.0
1.4	12.8	0.3	0.3
1	133.8	3.0	3.3
0.85	89.4	2.0	5.3
0.5	884.7	19.8	25.1
0.3	698.8	15.6	40.7
0.212	532.0	11.9	52.7
(-)0.212	2114.5	47.3	100.0
INPUT	**4466.0**	**100.0**	

Table B6

DATE: 01/10/2009			
wemco Test 6			
Sample ID	**WET MASSES (g)**	**DRY MASSES(g)**	**ASH %**
Flotation Feed	**19100**	**2135**	**25.9**
Flotation Cell 1	**18960**	**4289**	**7.7**
Flotation Cell 2	**19140**	**5229**	**10**
Flotation Cell 3	**19940**	**5245**	**10.5**
Flotation Cell 4	**15060**	**4556**	**15**
Flotation Cell 5	**11980**	**3116**	**14.2**
Flotation Cell 6	**12460**	**2876**	**18.6**
Tailings	**19240**	**1599**	**40.7**

SCREEN ANALYSIS		Product Ash	12.1

Flotation Feed

SIZE (mm)	**FRAC MASS (g)**	**FRAC MASS %**	**CUM MASS %**
2.8	10.3	0.5	0.0
2.0	34.4	1.6	1.6
1.4	122.6	5.8	7.5
1.0	209.9	10.0	17.4
0.85	99.3	4.7	22.2
0.5	333.7	15.9	38.0
0.3	309.5	14.7	52.7
0.212	151.9	7.2	60.0
(-)0.212	831.7	39.5	99.5
INPUT	**2103.3**	**100.0**	

Tailings

SIZE (mm)	**FRAC MASS (g)**	**FRAC MASS %**	**CUM MASS %**
2.8	17.4	1.1	0.0
2.0	54.8	3.5	3.5
1.4	257.1	16.4	19.9
1.0	312.6	20.0	39.9
0.85	80.5	5.1	45.0
0.5	346.2	22.1	67.2
0.3	162.0	10.4	77.5
0.212	62.4	4.0	81.5
(-)0.212	272.1	17.4	98.9
INPUT	**1565.1**	**100.0**	

Flotation Cell 1			
SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	0.0	0.0	0.0
2	0.0	0.0	0.0
1.4	5.8	0.1	0.1
1	97.3	2.3	2.4
0.85	44.3	1.0	3.5
0.5	567.9	13.3	16.8
0.3	1001.3	23.5	40.3
0.212	358.1	8.4	48.8
(-)0.212	2179.7	51.2	100.0
INPUT	**4254.4**	**100.0**	
Flotation Cell 2			
SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	0.0	0.0	0.0
2	0.0	0.0	0.0
1.4	41.4	0.8	0.8
1	239.2	4.6	5.4
0.85	153.1	2.9	8.3
0.5	814.8	15.7	24.0
0.3	1339.1	25.7	49.8
0.212	322.9	6.2	56.0
(-)0.212	2290.5	44.0	100.0
INPUT	**5201**	**100**	
Flotation Cell 3			
SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	0.0	0.0	0.0
2	0.0	0.0	0.0
1.4	72.5	1.4	1.4
1	200.2	3.8	5.2
0.85	71.4	1.4	6.6
0.5	625.3	12.0	18.6
0.3	1132.9	21.7	40.3
0.212	533.7	10.2	50.5
(-)0.212	2582.2	49.5	100.0
INPUT	**5218.2**	**100.0**	

Table B7

DATE: 13/10/2009			
Wemco Test 7			
Sample ID	**WET MASSES (g)**	**DRY MASSES(g)**	**ASH %**
Flotation Feed	**17938**	**2140**	**18.9**
Flotation Cell 1	**14557**	**4489**	**7.5**
Flotation Cell 2	**14071**	**4422**	**7.9**
Flotation Cell 3	**12669**	**3708**	**10.6**
Flotation Cell 4	**14231**	**3805**	**10.4**
Flotation Cell 5	**11494**	**2539**	**11.7**
Flotation Cell 6	**12000**	**2737**	**11**
Tailings	**15614**	**1036**	**33.6**
SCREEN ANALYSIS		**Product Ash**	**9.6**

Flotation Feed			
SIZE (mm)	**FRAC MASS (g)**	**FRAC MASS %**	**CUM MASS %**
2.8	8.8	0.4	0.0
2.0	19.8	0.9	0.9
1.4	131.8	6.2	7.2
1.0	220.0	10.4	17.6
0.85	71.0	3.4	21.0
0.5	428.0	20.3	41.2
0.3	253.3	12.0	53.2
0.212	189.5	9.0	62.2
(-)0.212	788.7	37.4	99.6
INPUT	**2110.9**	**100.0**	

Tailings			
SIZE (mm)	**FRAC MASS (g)**	**FRAC MASS %**	**CUM MASS %**
2.8	4.9	0.5	0.0
2.0	26.3	2.6	2.6
1.4	115.7	11.4	14.0
1.0	205.6	20.3	34.3
0.85	49.0	4.8	39.2
0.5	200.8	19.8	59.0
0.3	33.9	3.3	62.4
0.212	91.5	9.0	71.4
(-)0.212	284.7	28.1	99.5
INPUT	**1012.4**	**100.0**	

Flotation Cell 1			
SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	0.0	0.0	0.0
2	0.0	0.0	0.0
1.4	30.4	0.7	0.7
1	230.4	5.2	5.9
0.85	139.9	3.1	9.0
0.5	1116.0	25.1	34.1
0.3	790.0	17.7	51.8
0.212	636.8	14.3	66.1
(-)0.212	1509.2	33.9	100.0
INPUT	**4452.7**	**100.0**	

Flotation Cell 2			
SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	0.0	0.0	0.0
2	0.0	0.0	0.0
1.4	15.0	0.3	0.3
1	153.3	3.5	3.8
0.85	133.5	3.0	6.9
0.5	872.2	19.9	26.8
0.3	1052.7	24.0	50.7
0.212	438.7	10.0	60.7
(-)0.212	1723.1	39.3	100.0
INPUT	**4389**	**100**	

Flotation Cell 3			
SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	0.0	0.0	0.0
2	0.0	0.0	0.0
1.4	29.5	0.8	0.8
1	186.0	5.1	5.9
0.85	142.9	3.9	9.7
0.5	800.0	21.7	31.5
0.3	723.6	19.7	51.1
0.212	250.2	6.8	57.9
(-)0.212	1547.3	42.1	100.0
INPUT	**3679.5**	**100.0**	

Table B8

DATE: 15/10/2009			
Wemco Test 8			
Sample ID	**WET MASSES (g)**	**DRY MASSES(g)**	**ASH %**
Flotation Feed	**18340**	**2108**	**25.7**
Flotation Cell 1	**18780**	**4978**	**8.9**
Flotation Cell 2	**18800**	**5774**	**11.7**
Flotation Cell 3	**18340**	**4590**	**15.8**
Flotation Cell 4	**15320**	**3342**	**17.5**
Flotation Cell 5	**10240**	**2266**	**16**
Flotation Cell 6	**13380**	**2739**	**21**
Tailings	**21500**	**2636**	**45.9**
SCREEN ANALYSIS		**Product Ash**	**14.2**

Flotation Feed

SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	5.5	0.3	0.0
2.0	29.7	1.4	1.4
1.4	136.2	6.6	8.0
1.0	216.1	10.4	18.4
0.85	79.3	3.8	22.3
0.5	341.9	16.5	38.8
0.3	346.2	16.7	55.5
0.212	100.7	4.9	60.3
(-)0.212	817.0	39.4	99.7
INPUT	**2072.6**	**100.0**	

Tailings

SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	19.4	0.7	0.0
2.0	69.3	2.7	2.7
1.4	280.6	10.8	13.5
1.0	401.8	15.5	29.0
0.85	884.0	34.1	63.0
0.5	339.8	13.1	76.1
0.3	145.2	5.6	81.7
0.212	60.1	2.3	84.0
(-)0.212	394.8	15.2	99.3
INPUT	**2595.0**	**100.0**	

Flotation Cell 1			
SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	0.0	0.0	0.0
2	0.0	0.0	0.0
1.4	19.4	0.4	0.4
1	127.6	2.6	3.0
0.85	68.7	1.4	4.4
0.5	691.2	14.0	18.4
0.3	1046.5	21.2	39.6
0.212	467.2	9.5	49.0
(-)0.212	2518.2	51.0	100.0
INPUT	**4938.8**	**100.0**	

Flotation Cell 2			
SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	0.0	0.0	0.0
2	0.0	0.0	0.0
1.4	90.9	1.6	1.6
1	335.2	5.8	7.4
0.85	270.6	4.7	12.1
0.5	1008.2	17.6	29.7
0.3	1008.1	17.6	47.2
0.212	661.5	11.5	58.8
(-)0.212	2367.5	41.2	100.0
INPUT	**5742**	**100**	

Flotation Cell 3			
SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	0.0	0.0	0.0
2	0.0	0.0	0.0
1.4	50.1	1.1	1.1
1	180.0	4.0	5.1
0.85	72.6	1.6	6.7
0.5	473.9	10.4	17.1
0.3	728.1	16.0	33.1
0.212	527.2	11.6	44.7
(-)0.212	2513.1	55.3	100.0
INPUT	**4545.0**	**100.0**	

Table B9: Wemco average + 0.5mm recovery

Wemco Average + 0.5mm			
	% + 0.5mm Cell 1,2 & 3	% Feed	% Tailings
Test 1	33.1	46.9	58.7
Test 2	30.1	48.9	80.3
Test 3	30.8	41.2	59.0
Test 4	19.4	52.2	44.6
Test 5	17.2	38.1	53.7
Test 6	19.8	38.0	67.2
Test 7	30.8	41.2	59.0
Test 8	7.7	22.3	63.0
Total Average	**23.6**	**41.1**	**60.7**
Standard deviation	**9.0**	**9.2**	**10.3**

Table B10: Wemco average ash and yield

Wemco Average Ash and Yield				
	Feed Ash	Product Ash	Tailings Ash	Yield
Test 1	20.2	8.9	37.0	59.7
Test 2	18.4	10.0	35.2	66.6
Test 3	18.9	9.6	33.6	61.1
Test 4	26.2	8.9	33.7	30.3
Test 5	20.8	10.5	33.8	55.8
Test 6	25.9	12.1	40.7	51.8
Test 7	18.9	9.6	33.6	61.1
Test 8	25.7	14.2	45.9	63.7
Total Average	21.9	10.5	36.7	56
Standard Deviation	3.4	1.8	4.5	11.4

Table B11: Wemco + 0.5mm recovery per cell

Wemco + 0.5mm Recovery Per Cell			
	Cell 1	Cell 2	Cell 3
Test 1	35.4	26.6	37.3
Test 2	27.7	32.6	30.1
Test 3	34.1	26.8	31.5
Test 4	20.1	19.4	18.6
Test 5	4.0	22.6	25.1
Test 6	16.8	24.0	18.6
Test 7	34.1	26.8	31.5
Test 8	18.4	29.7	17.1
Total Average	**23.8**	**26.1**	**26.2**
Standard Deviation	**11.0**	**4.1**	**7.5**

Table B12: Wemco ash percentage per cell

Average Ash Per Cell			
	Cell 1	Cell 2	Cell 3
Test 1	7.1	8	8.8
Test 2	8	7.6	8.1
Test 3	7.5	7.9	10.6
Test 4	9.7	7.2	7.9
Test 5	10.9	10.5	9.2
Test 6	7.7	10	10.5
Test 7	7.5	7.9	10.6
Test 8	8.9	11.7	15.8
Average Ash Per Cell	8.4	8.9	10.2
Standard Deviation	1.3	1.6	2.5

Attachment C: Ultimate Results Test 1 to Test 6

Table C1

DATE: 15/09/2010			
Ultimate Test 1			
Sample ID	**WET MASSES (g)**	**DRY MASSES(g)**	**ASH %**
Flotation Feed	20560	2749	23.2
Flotation Cell 1	18980	5159	7.6
Flotation Cell 2	19180	5060	9.4
Flotation Cell 3	19820	6097	14.4
Flotation Cell 4	18080	5077	14.7
Flotation Cell 5	15420	3356	15.5
Flotation Cell 6	15600	2740	18.8
Tailings	23380	1875	40.6
Product Ash	12.8	Frequency	
		Cell 1	35hz
SCREEN ANALYSIS		Cell 2	32hz
		Cell 3	30hz

Flotation Feed

SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	0.0	0.0	0.0
2.0	45.6	1.7	1.7
1.4	171.7	6.3	8.0
1.0	285.1	10.5	18.6
0.85	151.7	5.6	24.2
0.5	605.1	22.4	46.5
0.3	433.2	16.0	62.5
0.212	114.7	4.2	66.8
(-)0.212	898.9	33.2	100.0
INPUT	**2706.0**	**100.0**	

Tailings

SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	22.1	1.2	0.0
2.0	67.9	3.7	3.7
1.4	337.4	18.2	21.8
1.0	528.4	28.5	50.3
0.85	87.5	4.7	55.0
0.5	351.3	18.9	73.9
0.3	137.1	7.4	81.3
0.212	53.3	2.9	84.2
(-)0.212	272.1	14.7	98.8
INPUT	**1857.1**	**100.0**	

Flotation Cell 1			
SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	0.0	0.0	0.0
2	0.0	0.0	0.0
1.4	4.2	0.1	0.1
1	95.1	1.9	1.9
0.85	86.1	1.7	3.6
0.5	698.1	13.6	17.3
0.3	1006.1	19.7	36.9
0.212	205.1	4.0	40.9
(-)0.212	3023.8	59.1	100.0
INPUT	**5118.5**	**100.0**	

Flotation Cell 2			
SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	0.0	0.0	0.0
2	0.0	0.0	0.0
1.4	23.5	0.5	0.5
1	172.7	3.4	3.9
0.85	138.2	2.7	6.7
0.5	789.4	15.7	22.4
0.3	1216.0	24.2	46.6
0.212	359.2	7.1	53.7
(-)0.212	2326.9	46.3	100.0
INPUT	**5026**	**100**	

Flotation Cell 3			
SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	0.0	0.0	0.0
2	0.0	0.0	0.0
1.4	79.8	1.3	1.3
1	319.4	5.3	6.6
0.85	216.5	3.6	10.2
0.5	909.8	15.0	25.2
0.3	1373.9	22.7	47.9
0.212	489.0	8.1	55.9
(-)0.212	2668.9	44.1	100.0
INPUT	**6057.3**	**100.0**	

Table C2

DATE: 22/09/2009			
Ultimate Test 2			
Sample ID	**WET MASSES (g)**	**DRY MASSES(g)**	**ASH %**
Flotation Feed	20200	2490	24.9
Flotation Cell 1	19020	4423	8
Flotation Cell 2	19520	5559	10.8
Flotation Cell 3	19950	6492	15
Flotation Cell 4	18250	4196	20.5
Flotation Cell 5	14330	3484	19.9
Flotation Cell 6	12540	2124	25.4
Tailings	19160	1544	50.6
Product Ash	**15.3**	**Frequency**	
		Cell 1	35hz
SCREEN ANALYSIS		Cell 2	33hz
		Cell 3	31hz

Flotation Feed

SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	11.3	0.5	0.0
2.0	32.6	1.3	1.3
1.4	154.3	6.3	7.6
1.0	222.1	9.0	16.6
0.85	100.1	4.1	20.7
0.5	378.5	15.4	36.0
0.3	360.0	14.6	50.6
0.212	192.7	7.8	58.4
(-)0.212	1013.5	41.1	99.5
INPUT	**2465.1**	**100.0**	

Tailings

SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	30.0	2.0	0.0
2.0	87.3	5.7	5.7
1.4	280.7	18.4	24.2
1.0	356.8	23.4	47.6
0.85	99.0	6.5	54.1
0.5	266.8	17.5	71.6
0.3	107.5	7.1	78.6
0.212	36.5	2.4	81.0
(-)0.212	259.0	17.0	98.0
INPUT	**1523.6**	**100.0**	

Flotation Cell 1			
SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	0.0	0.0	0.0
2	0.0	0.0	0.0
1.4	11.4	0.3	0.3
1	124.8	2.8	3.1
0.85	82.1	1.9	5.0
0.5	100.2	2.3	7.2
0.3	907.5	20.6	27.9
0.212	396.9	9.0	36.9
(-)0.212	2773.8	63.1	100.0
INPUT	**4396.7**	**100.0**	

Flotation Cell 2			
SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	0.0	0.0	0.0
2	0.0	0.0	0.0
1.4	85.6	1.5	1.5
1	249.8	4.5	6.1
0.85	195.7	3.5	9.6
0.5	1430.3	25.9	35.5
0.3	940.7	17.0	52.5
0.212	441.9	8.0	60.5
(-)0.212	2186.6	39.5	100.0
INPUT	**5531**	**100**	

Flotation Cell 3			
SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	0.0	0.0	0.0
2	26.9	0.4	0.4
1.4	260.5	4.0	4.4
1	656.8	10.2	14.6
0.85	274.3	4.2	18.8
0.5	1081.0	16.7	35.6
0.3	1075.2	16.6	52.2
0.212	446.6	6.9	59.1
(-)0.212	2644.6	40.9	100.0
INPUT	**6465.9**	**100.0**	

Table C3

DATE: 01/10/2009			
Ultimate test 3			
Sample ID	**WET MASSES (g)**	**DRY MASSES(g)**	**ASH %**
Flotation Feed	15980	1602	21.5
Flotation Cell 1	13278	3844	8.9
Flotation Cell 2	10641	1986	10.8
Flotation Cell 3	6902	1930	10.6
Flotation Cell 4	6141	2598	9.79
Flotation Cell 5	4816	1254	16.7
Flotation Cell 6	4193	973	17.6
Tailings	13822	725	38.6
Product Ash	11.1	Frequency	
		Cell 1	33hz
SCREEN ANALYSIS		Cell 2	35hz
		Cell 3	31hz

Flotation Feed			
SIZE (mm)	**FRAC MASS (g)**	**FRAC MASS %**	**CUM MASS %**
2.8	0.0	0.0	0.0
2.0	15.3	1.0	1.0
1.4	91.6	5.8	6.8
1.0	161.3	10.2	17.0
0.85	50.5	3.2	20.2
0.5	234.5	14.9	35.0
0.3	180.4	11.4	46.5
0.212	343.4	21.7	68.2
(-)0.212	502.0	31.8	100.0
INPUT	**1579.0**	**100.0**	

Tailings			
SIZE (mm)	**FRAC MASS (g)**	**FRAC MASS %**	**CUM MASS %**
2.8	0.0	0.0	0.0
2.0	23.8	3.4	3.4
1.4	134.6	19.2	22.6
1.0	165.0	23.6	46.2
0.85	50.8	7.3	53.5
0.5	123.6	17.7	71.2
0.3	61.7	8.8	80.0
0.212	54.4	7.8	87.8
(-)0.212	85.6	12.2	100.0
INPUT	**699.5**	**100.0**	

Flotation Cell 1			
SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	0.0	0.0	0.0
2	0.0	0.0	0.0
1.4	40.4	1.1	1.1
1	241.0	6.3	7.4
0.85	108.0	2.8	10.2
0.5	1291.8	33.8	43.9
0.3	547.4	14.3	58.2
0.212	361.3	9.4	67.7
(-)0.212	1236.4	32.3	100.0
INPUT	**3826.3**	**100.0**	

Flotation Cell 2			
SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	0.0	0.0	0.0
2	0.0	0.0	0.0
1.4	11.6	0.6	0.6
1	76.8	3.9	4.5
0.85	42.7	2.2	6.7
0.5	248.0	12.6	19.3
0.3	228.4	11.6	30.9
0.212	181.3	9.2	40.2
(-)0.212	1174.1	59.8	100.0
INPUT	**1963**	**100**	

Flotation Cell 3			
SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	0.0	0.0	0.0
2	0.0	0.0	0.0
1.4	10.7	0.6	0.6
1	103.8	5.5	6.0
0.85	74.5	3.9	10.0
0.5	459.8	24.3	34.3
0.3	242.7	12.8	47.1
0.212	202.4	10.7	57.8
(-)0.212	800.2	42.2	100.0
INPUT	**1894.1**	**100.0**	

Table C4

DATE: 07/10/2009			
Ultimate test 4			
Sample ID	WET MASSES (g)	DRY MASSES(g)	ASH %
Flotation Feed	18067	1150	21.9
Flotation Cell 1	18295	5195	8.3
Flotation Cell 2	15708	3459	7.8
Flotation Cell 3	13193	3898	9
Flotation Cell 4	14922	4278	11.7
Flotation Cell 5	12761	2682	17.1
Flotation Cell 6	15687	2345	23.9
Tailings	11729	1024	43.1
Product Ash	11.8	Frequency	
		Cell 1	33hz
SCREEN ANALYSIS		Cell 2	28hz
		Cell 3	25hz

Flotation Feed

SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	11.5	1.0	0.0
2.0	28.1	2.5	2.5
1.4	118.9	10.5	13.0
1.0	182.0	16.1	29.2
0.85	52.4	4.6	33.8
0.5	259.4	23.0	56.8
0.3	202.0	17.9	74.7
0.212	175.3	15.5	90.2
(-)0.212	98.6	8.7	99.0
INPUT	1128.2	100.0	

Tailings

SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	19.1	1.9	0.0
2.0	47.8	4.8	4.8
1.4	147.1	14.7	19.4
1.0	185.7	18.5	37.9
0.85	38.8	3.9	41.8
0.5	104.0	10.4	52.1
0.3	38.2	3.8	56.0
0.212	21.9	2.2	58.1
(-)0.212	401.1	40.0	98.1
INPUT	1003.7	100.0	

Flotation Cell 1			
SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	0.0	0.0	0.0
2	0.0	0.0	0.0
1.4	85.0	1.6	1.6
1	246.3	4.8	6.4
0.85	197.6	3.8	10.2
0.5	1058.4	20.5	30.7
0.3	1016.5	19.6	50.3
0.212	541.3	10.5	60.8
(-)0.212	2028.0	39.2	100.0
INPUT	**5173.1**	**100.0**	

Flotation Cell 2			
SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	0.0	0.0	0.0
2	0.0	0.0	0.0
1.4	20.7	0.6	0.6
1	192.2	5.6	6.2
0.85	95.3	2.8	9.0
0.5	1020.0	29.7	38.7
0.3	602.8	17.6	56.3
0.212	358.6	10.5	66.7
(-)0.212	1141.2	33.3	100.0
INPUT	**3431**	**100**	

Flotation Cell 3			
SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	0.0	0.0	0.0
2	0.0	0.0	0.0
1.4	32.8	0.8	0.8
1	367.7	9.5	10.3
0.85	86.9	2.2	12.6
0.5	1136.4	29.3	41.9
0.3	400.7	10.3	52.2
0.212	223.4	5.8	58.0
(-)0.212	1627.4	42.0	100.0
INPUT	**3875.3**	**100.0**	

Table C5

DATE: 04/02/2010			
Ultimate Test 5			
Sample ID	**WET MASSES (g)**	**DRY MASSES(g)**	**ASH %**
Flotation Feed	17680	2109	23.2
Flotation Cell 1	13900	4604	7.4
Flotation Cell 2	13240	4110	7.9
Flotation Cell 3	14680	4645	9.8
Flotation Cell 4	13280	3081	9.9
Flotation Cell 5	11120	2422	14.2
Flotation Cell 6	12080	2495	17.3
Tailings	14180	667	39.5
Product Ash	**10.3**	**Frequency**	
		Cell 1	30hz
SCREEN ANALYSIS		Cell 2	26hz
		Cell 3	22hz

Flotation Feed

SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	20.0	0.9	0.0
2.0	45.0	2.1	2.1
1.4	184.0	8.7	10.9
1.0	315.0	15.0	25.8
0.85	94.0	4.5	30.3
0.5	357.0	17.0	47.2
0.3	294.0	14.0	61.2
0.212	146.0	6.9	68.1
(-)0.212	651.0	30.9	99.1
INPUT	**2106.0**	**100.0**	

Tailings

SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	12.0	1.8	0.0
2.0	43.0	6.5	6.5
1.4	139.0	21.0	27.5
1.0	168.0	25.4	52.9
0.85	52.0	7.9	60.7
0.5	105.0	15.9	76.6
0.3	43.0	6.5	83.1
0.212	13.0	2.0	85.0
	87.0	13.1	98.2
INPUT	**662.0**	**100.0**	

Flotation Cell 1			
SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	0.0	0.0	0.0
2.0	0.0	0.0	0.0
1.4	25.0	0.5	0.5
1.0	139.0	3.0	3.6
0.85	98.0	2.1	5.7
0.5	582.0	12.6	18.3
0.3	760.0	16.5	34.9
0.212	420.0	9.1	44.0
(-)0.212	2577.0	56.0	100.0
INPUT	**4601.0**	**100.0**	

Flotation Cell 2			
SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	0.0	0.0	0.0
2.0	0.0	0.0	0.0
1.4	73.0	1.8	1.8
1.0	299.0	7.3	9.1
0.85	144.0	3.5	12.6
0.5	1030.0	25.1	37.7
0.3	794.0	19.4	57.1
0.212	329.0	8.0	65.1
(-)0.212	1428.0	34.9	100.0
INPUT	**4097**	**100**	

Flotation Cell 3			
SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	0.0	0.0	0.0
2.0	0.0	0.0	0.0
1.4	209.0	4.5	4.5
1.0	481.0	10.4	14.9
0.85	277.0	6.0	20.8
0.5	850.0	18.3	39.2
0.3	743.0	16.0	55.2
0.212	261.0	5.6	60.8
(-)0.212	1819.0	39.2	100.0
INPUT	**4640.0**	**100.0**	

Table C6

DATE: 30/03/2010			
Ultimate Test 6			
Sample ID	**WET MASSES (g)**	**DRY MASSES(g)**	**ASH %**
Flotation Feed	18820	1644	22.3
Flotation Cell 1	17880	4713	9.4
Flotation Cell 2	17240	5204	9.7
Flotation Cell 3	17920	6650	13.4
Flotation Cell 4	14360	3380	18.6
Flotation Cell 5	12020	1713	22.2
Flotation Cell 6	10720	2323	29.5
Tailings	18820	779	44.4
Product Ash	**14.7**	**Frequency**	
		Cell 1	33hz
SCREEN ANALYSIS		Cell 2	27hz
		Cell 3	23hz

Flotation Feed

SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	9.7	0.6	0.0
2.0	43.5	2.7	2.7
1.4	147.7	9.0	11.7
1.0	197.2	12.1	23.8
0.85	78.8	4.8	28.6
0.5	259.0	15.8	44.4
0.3	207.9	12.7	57.1
0.212	106.5	6.5	63.6
(-)0.212	584.7	35.8	99.4
INPUT	**1635.0**	**100.0**	

Tailings

SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	13.9	1.8	0.0
2.0	63.1	8.2	8.2
1.4	189.2	24.6	32.8
1.0	185.0	24.1	56.9
0.85	43.2	5.6	62.5
0.5	100.2	13.0	75.5
0.3	38.5	5.0	80.6
0.212	13.4	1.7	82.3
(-)0.212	122.2	15.9	98.2
INPUT	**768.7**	**100.0**	

Flotation Cell 1			
SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	0.0	0.0	0.0
2	17.4	0.4	0.4
1.4	50.2	1.1	1.5
1	403.8	8.7	10.1
0.85	120.3	2.6	12.7
0.5	1137.1	24.4	37.1
0.3	818.8	17.6	54.7
0.212	386.7	8.3	63.0
(-)0.212	1722.1	37.0	100.0
INPUT	**4656.4**	**100.0**	

Flotation Cell 2			
SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	0.0	0.0	0.0
2	12.3	0.2	0.2
1.4	155.1	3.0	3.2
1	532.4	10.3	13.6
0.85	285.0	5.5	19.1
0.5	1100.7	21.3	40.4
0.3	832.4	16.1	56.6
0.212	407.5	7.9	64.5
(-)0.212	1832.3	35.5	100.0
INPUT	**5158**	**100**	

Flotation Cell 3			
SIZE (mm)	FRAC MASS (g)	FRAC MASS %	CUM MASS %
2.8	0.0	0.0	0.0
2	87.5	1.3	1.3
1.4	566.0	8.6	9.9
1	1002.8	15.2	25.1
0.85	221.6	3.4	28.5
0.5	1176.0	17.8	46.3
0.3	945.1	14.3	60.6
0.212	415.7	6.3	66.9
(-)0.212	2182.0	33.1	100.0
INPUT	**6596.7**	**100.0**	

Table C7: Ultimate average + 0.5mm recovery

Ultimate Average + 0.5mm			
Average	**% + 0.5mm Cell 1,2 & 3 Ave**	**% Feed**	**% Tailings**
Test 1	21.6	46.5	73.9
Test 2	26.1	36.0	71.6
Test 3	32.5	35.0	71.2
Test 4	37.1	56.8	52.1
Test 5	31.7	47.2	76.6
Test 6	41.3	44.4	75.5
Total Average	**31.7**	**44.3**	**70.2**
Standard Deviation	**7.1**	**8.1**	**9.1**

Table C8: Ultimate average ash and yield

Ultimate Average Ash and Yield				
	Feed	**Product**	**Tailings**	**Yield**
Test 1	23.2	12.8	40.6	62.7
Test 2	24.9	15.3	50.6	72.8
Test 3	21.5	11.1	38.6	62.2
Test 4	21.9	11.8	43.1	67.7
Test 5	23.2	10.3	39.5	55.8
Test 6	22.3	14.7	44.4	74.5
Total Average	**22.8**	**12.7**	**42.8**	**66**
Standard deviation	**1.2**	**2.0**	**4.4**	**7.1**

Table C9: Ultimate + 0.5mm recovery per cell

Ultimate + 0.5mm Recovery Per Cell			
	Cell 1	Cell 2	Cell 3
Test 1	17.3	22.4	25.2
Test 2	7.2	35.5	35.6
Test 3	43.9	19.3	34.3
Test 4	30.7	38.7	41.9
Test 5	18.3	37.7	39.2
Test 6	37.1	40.4	46.3
Total Average	**25.8**	**32.3**	**37.1**
Standard deviation	**13.8**	**9.1**	**7.3**

Table C10: Ultimate average ash per cell

Average Ash Per Cell			
	Cell 1	Cell 2	Cell 3
Test 1	7.6	9.4	14.4
Test 2	8	10.8	15
Test 3	8.9	10.8	10.6
Test 4	8.3	7.8	9
Test 5	7.4	7.9	9.8
Test 6	9.4	9.7	13.4
Average Ash Per Cell	8.3	9.4	12.0
Standard Deviation	0.8	1.3	2.6

Attachment D: Flotation plant flow diagram