Investigation of Flash Flotation Technology Utilizing Centrifugal Forces and Novel Sparging Methods

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ABSTRACT

A new processing technique, centrifugal flotation, has been developed in recent research projects to overcome the large residence times and fine particle limitations of traditional flotation technologies. The major innovation in the area of centrifugal flotation is the Air Sparged Hydrocyclone (ASH), which has proven capabilities in achieving quality products at specific capacities greater than traditional flotation methods. However, the ASH technology ultimately suffers from sparger plugging problems. Therefore, three unique flotation cyclone designs were developed utilizing external sparging systems and control features to float fine coal. The objective of each design was to create a system that mimics the behavior of the ASH technology, while providing advantages in bubble generation and retention time requirements.

The evaluation of the three designs provided evidence towards the development of an efficient centrifugal flotation technique. Evaluation of a flotation cyclone with an external Cavitation Tube yielded a single-stage product with an ash content of 4.41% and a 45% recovery rate in a retention time of 0.66 seconds. However, the system required 16 minutes to meet comparable flotation yields and recoveries. The third design achieved a multiple-stage product of 11.32% ash at a 55% recovery in 20 minutes. These two designs provided low yield, high grade products, but rejected a high percentage of hydrophobic particles and required high retention times to meet typical flotation standards. In addition, these designs suffered by requiring high frother concentrations and recovery could not be increased through increased aeration due to design limitations.

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TABLE OF CONTENTS

LIST OF FIGURES

LIST OF TABLES

1.0 INTRODUCTION

1.1 Background

1.1.1 Flotation Fundamentals

The processing of fine particles in mining and minerals industry has been a dynamic challenge since the development of higher production extraction technologies and the continual decrease of ore grades. Regarded as the most widely used separation technique in mineral processing applications, flotation has developed into an efficient process to upgrade the fine size fractions (minus 100 mesh) of raw ore at high capacities. Flotation exploits differences in surface properties of particles through wettability. Particles that show an affinity for water are classified as hydrophilic while particles that tend to repel water are considered hydrophobic. This fundamental surface property is exploited to create an adequate bubble particle contact through the use of conventional and column cells, chemical reagent addition, and novel bubble generation designs to increase the flotation kinetics of hydrophobic particles and yield a quality product.

Conventional and column flotation machines achieve the same objective of recovering fine hydrophobic particles but varying in their methods and efficiency of separation (Figure 1.1). Typically, a conventional machine consists of a large cell where the flotation feed enters the lower portion and is mixed axially by a rotating impeller. The rotating stator draws both air and slurry to disperse bubbles into the cell. In a column cell, the flotation feed enters near the top of the cell and bubbles are generated through an external or internal sparging system (Kawatra, 2011). Other than the physical design differences, the column cell and conventional mechanical cell contrast in their respective effectiveness of bubble and particle collection, particle and bubble contact, and entrainment of fine gangue particles. In conventional cells, the bubble and

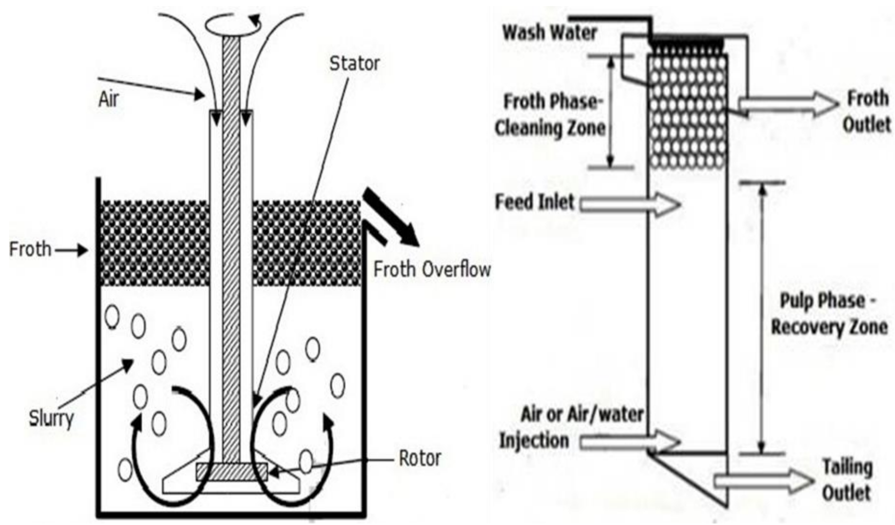


Figure 1.1. Comparison of Conventional and Column Cell (Kawatra, 2011)

particle collection zone is essentially the area surrounding the mechanical impeller where the bubbles are originally dispersed. Column cells however use the entire volume of the cell to allow for bubble and particle contact. The large collection area of the column cell also allows for the separate entry points for the slurry and generated bubbles. The location of the bubble generation point allows the bubbles to rise and directly collide with the suspended particles. In contrast, the particle is dependent on the rotating impeller to provide adequate bubble particle contact, but the action of the impeller often creates turbulence which can lead to particle detachment (Laskowski, 2001). Finally, column flotation is superior to conventional machines through the practice of washing the froth with water to provide a counter-current flow in the froth phase and reduce entrainment of ultrafine gangue particles reporting with the concentrate (Laskowski, 2001).

In either case, these bubbles ascend towards the froth phase colliding with both hydrophobic and hydrophilic particles. Due to the surface properties of these particles, the hydrophobic particles attach to the air bubble and travel towards the froth product while the hydrophilic particles do not attach and report to the underflow. The ability of each flotation machine to recover the hydrophobic particles is dependent of bubble particle attachment, flotation reagents, and bubble generation which will be discussed in the proceeding sections.

1.1.2 Bubble Particle Attachment

The attachment between the bubble and particle in the case of flotation is the most critical principal in the separation of hydrophobic minerals from hydrophilic particles. The thermodynamics of a particle attaching to a gas bubble is shown in Figure 1.2. The thermodynamic relationship between the particle and bubble can be described by Young's equation. The Young's equation describes the interfacial energy equilibrium required for bubble particle adhesion and relates the following interfaces: interfacial tension between liquid and vapor (γ_{LV}), interfacial tension between solid and liquid (γ_{SL}), and interfacial tension between solid and vapor (γ_{SV}). Using these energy relationships, particle bubble attachment will occur as the difference between the interfacial energies results in a negative value (Yoon, 2011).

$$\gamma_{sv} - \gamma_{lv} - \gamma_{sl} < 0 \qquad [1.1]$$

Continuing the analysis and using the diagram shown in Figure 1.2, the contact angle θ between the tension at the slurry vapor interface and the other interfacial tensions will be the

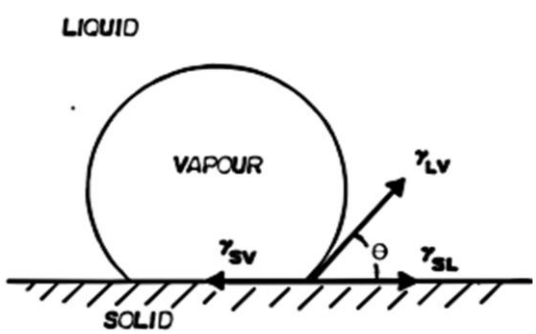


Figure 1.2. Thermodynamic Description of Particle Bubble Attachment (Yoon, 2011). Yoon, R. (2011). Froth Flotation: Thermodynamics of Flotation. Blacksburg, Virginia.
Used under fair use. Form attached.

main parameter in the particle adhering to the bubble.

$$\gamma_{sv} = \gamma_{lv} cos\theta + \gamma_{sl} \quad [1.2]$$

Thus, in order to obtain particle adhesion, a contact angle of 90 degrees will be necessary to reduce the difference between interfacial energies below zero (Yoon, 2011).

Since the development of flotation in the mineral processing industry, several models have been studied to predict particle bubble attachment. In Sutherland's model of particle collection probability in "Kinetics of Flotation Process", Sutherland concluded the recovery of the particle is dependent on the probability of collision between a bubble and particle, the probability for the particle to attach to the bubble, and the associated probability of the bubble detaching from the bubble through the flotation process (Sutherland, 1948). Based on Sutherland's framework, works done by others have concluded the probability of particle collision is a function of the ratio of particle diameter to bubble diameter (Yoon & Luttrell, The Effect of Bubble Size on Fine Particle Flotation, 1989), and the effects of particle size on recovery can be seen in Figure 1.3 (Gaudin, Grob, & Henderson, 1931).

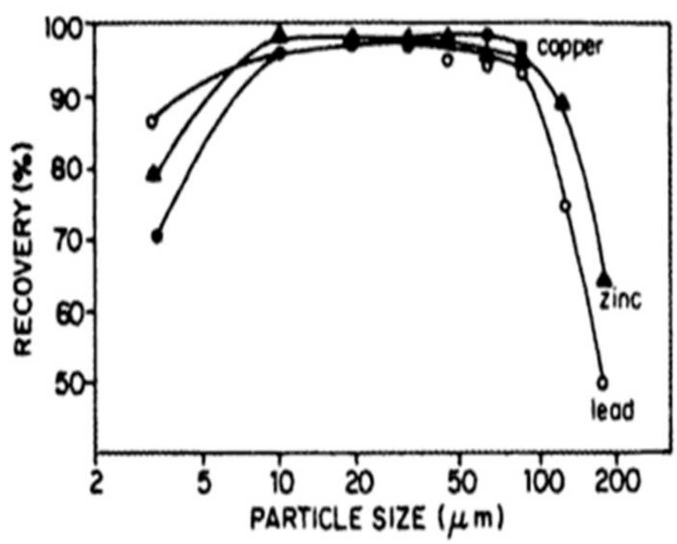


Figure 1.3. Flotation Recovery versus Particle Size (Gaudin, Grob, & Henderson, 1931). The Effect of Particle size on Flotation – AIME, Volume 414. Gaudin A., Grob J., Henderson H. Used with Permission from Steve Kral, Editor of Mining Engineer Magazine at Society of Mining, Metallurgy, and Exploration.

Once the particle collides with the bubble, the particle has the opportunity to attach itself to the bubble to allow for recovery and is known as the probability of attachment. After this collision, the particle remains on the bubble surface for a period of time as the particles slide a specific distance. The period of time that particle slides over the bubble is the sliding time and is related to the velocity of the surrounding liquid as the bubble ascends towards the froth product (Yoon & Luttrell, The Effect of Bubble Size on Fine Particle Flotation, 1989). Sutherland detailed the necessary time for a particle to adhere to a bubble as the induction period (Sutherland, 1948) thus defining a specific time period for the particle to slide along the bubble, rupture the surface, and become attached. Yoon and Luttrell derived probability of attachment equations for various flow conditions and concluded the critical parameters of attachment are bubble radius, particle radius, bubble velocity, and induction time (Yoon & Luttrell, The Effect of Bubble Size on Fine Particle Flotation, 1989). In validation of their conclusions, as induction time decreases for a given bubble and particle size, probability of attachment increases and the

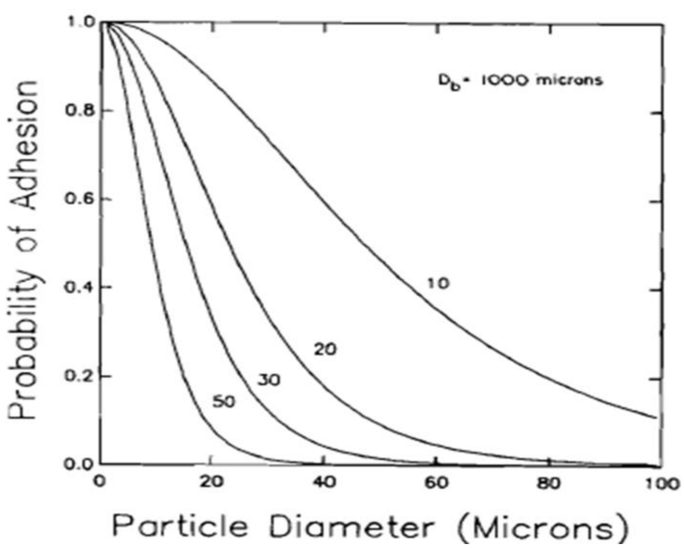


Figure 1.4. Analysis of Adhesion Probability in Flotation at Varying Induction Times (Yoon & Luttrell, 1989). The Effect of Bubble size on Fine Particle Flotation - Mineral Processing and Extractive Review, Volume 5 Issue 1-4 PP 101-122, R. H. Yoon and G. H. Luttrell, Used with permission of Deborah East, www.tandfonline.com, 2014.

same result occurs as particle diameter is decreased (Figure 1.4).

1.1.3 Flotation Reagents

The flotation process can be efficiently modified with the addition of chemicals which alter the chemistry of the slurry or the mineral surface as most feed ores are not ideally suited for the separation of valuable particles. The categories of flotation reagents include collectors, frothers, and modifiers with each type serving a specific purpose to allow for optimum separation. Collectors selectively increase the hydrophobicity of minerals by providing a thin film of hydrocarbons over the selected surface through chemisorption or adsorption. This coating of hydrocarbons essentially increases the contact angle of the particle and bubble aggregate thus improving flotation. Collectors are typically added upstream of the flotation machines to allow for proper conditioning. Although collectors do provide an added advantage of coating non hydrophobic particles with a hydrocarbon film, some ores, like coal, require little to no collector addition thus saving flotation costs (Kawatra, 2011).

Often utilized in junction with collectors are frothers which act as bubble and froth stabilizers. These alcohol based or synthetic compounds reduce the surface tension of the liquid and provide the necessary stability for air bubbles to remain in slurry, capture particles, and ascend to the froth phase with the attached particles. Additionally, frothers create a stable froth phrase allowing for efficient collection of the recovered minerals and ensuring particles do not detached and descend back into the pulp phase of the flotation process (Kawatra, 2011). With these characteristics of frothers, the ultimate effect of the addition of frother is an increase in flotation rate thus increasing recovery. However, the increase in recovery leads to the increased recovery of gangue particles (Klimpel, 1995).

Other flotation chemicals such as modifiers are added to the slurry in order to control the selectively of the flotation process. These modifiers can serve multiple purposes as mineral activators and depressants through controlling the way a collector adheres to each mineral surface or regulating the pH of the pulp in order to induce flotation of different minerals. Specifically, activators allow the adhesion of a specific collector to a mineral that would not normally attach whereas depressants prevent the adhesion. The pH modifiers use acids and alkalis to either lower and raise the pH of the pulp and create an effective environment for the flotation of specific minerals (Kawatra, 2011).

1.1.4 Bubble Generation

Considering the critical necessity for bubble generation throughout the flotation process, the principles of bubble generation in conventional and column cells have been constantly analyzed. As described, slurry enters the impeller of a conventional cell, and air is either drawn in by the vacuum created by the movement of slurry or provided by an installed blower. The

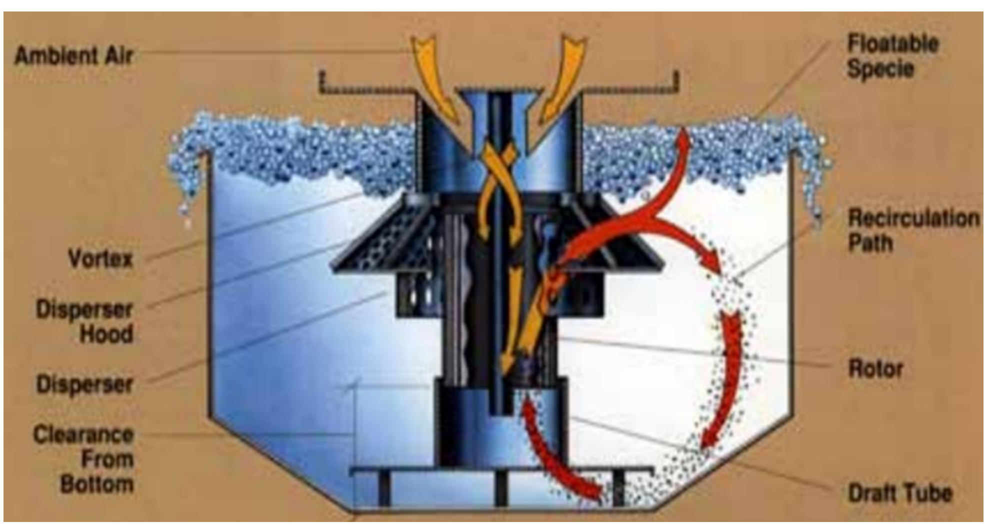
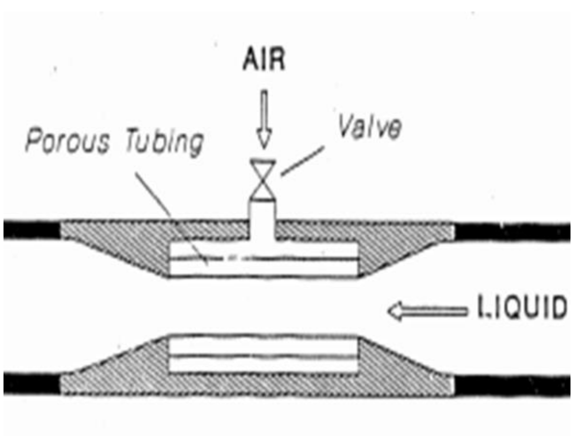


Figure 1.5. WEMCO 1+1 Flotation Cell (WEMCO 1+1 Flotation Cell, 2010). WEMCO 1+1 Flotation Cell. (2010). Retrieved from FL Smidth: http://www.flsmidth.com/~/media/PDF%20Files/Liquid-Solid%20Separation/Flotation/Wemco11brochure.ashx. Used with permission from Andrew Cuthbert, Director of Global Marketing at FL Smidth.

rotating impeller disperses the slurry air mixture and generates bubbles for particle bubble attachment. Shown is a cross sectional view of the fluid and air motion in a mechanically agitated cell (Figure 1.5).

Due to the turbulence created of the rotating impeller and the overall length, flotation columns adopted sparging systems as the primary bubble generation system. Several sparging systems have been developed since the air diffusers made from ceramic material in the early applications of column flotation (Kawatra, 2011). Such systems include static or inline mixers, porous tubes, and devices that utilize venturi principle (Figure 1.6). In the early developments of the Microcel, Yoon *et al.* used a porous venturi-tube sparger where air was drawn into porous tubing and bubbles were generated by the shearing force of the passing slurry (Yoon, Luttrell, & Adel, 1990).

Canadian Process Technologies Inc. developed a sparging system that utilizes the advantages of cavitation. An air and liquid mixture is subjected to a rapid decrease in pressure



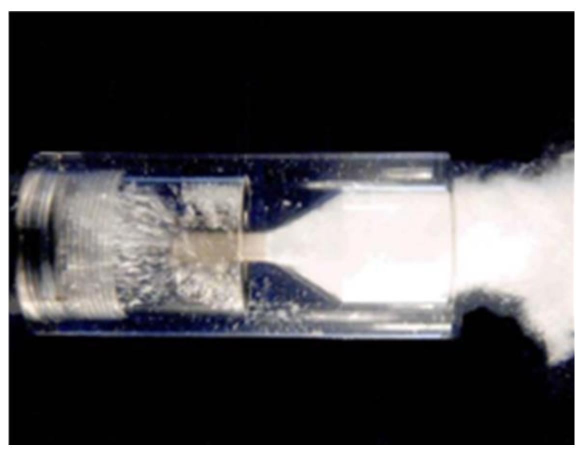


Figure 1.6. Porous Venturi Sparging System (Yoon, Luttrell, & Adel, 1990) and CPT Cavitation Tube System (Column Flotation Systems Cavitation Tube, 2009). *Yoon, R., Luttrell, G., & Adel, G. (1990). Advanced Systems for Producing Superclean Coal. Blacksburg: U.S. Department of Energy. Fair use as government publication.Column Flotation Systems Cavitation Tube. (2009). Retrieved from Canadian Process Technologies Inc.: http://efd.eriez.com/Products/Index/Cavitationtube Used with permission from Dr. Jaisen Kohmuench, Deputy Managing Director at Eriez Flotation Division.*

through a small orifice and cavitation occurs. With the presence of frother and the subsequent increase in area, picobubbles are created for the flotation process (Column Flotation Systems Cavitation Tube, 2009).

1.1.5 Flotation Kinetics

The flotation of particles can be described as a simple model which takes into consideration the floatability of the particle and the time it is exposed to the flotation process. The model shown below represents the recovery of a specific particle and is described as the ratio of recovered mineral mass to the total recoverable mineral mass.

$$R = \frac{kt}{1+kt} \tag{1.3}$$

Where *k* is the flotation rate constant of the particle and *t* is the residence time (Adel, 2014).

The flotation constant of a mineral is dependent on physical particle parameters such as particle size, hydrophobicity, mineral composition, and operating parameters such as aeration rate and reagent addition. As previously described, the recovery has an optimum particle size range and drastically lowers below 400 mesh and above 60 mesh. Thus, the flotation constant can be controlled through efficient comminution and size classification of particles. In addition, the recovery of a particle will be affected by the natural or chemically altered hydrophobicity and consequently increase or decrease the flotation constant. The flotation rate of a particle is also dependent on the mineral composition and whether the particle is a free pure mineral or consists of a gangue mineral. This factor is least controllable as much of composition is dependent on the ore particle size. Creating free particles can be achieved through regrind circuits but these circuits decrease the particle size and can be cost intensive. An increase in operating parameters such as aeration rate or frother concentration will lead to an increase recovery as the probability of bubble particle attachment increases with these adjustments.

1.2 Objectives

This project seeks to design, develop, and analyze new flotation techniques which take advantage of centrifugal forces and novel sparging methods to achieve acceptable yield and recovery rates while maximizing capacity per unit volume and shifting the particle recovery curve towards finer size fractions. The basis for this design work is the introduction of centrifugal forces to increase the flotation rate constant and the inertia for fine particles. For this study, three separation designs are modified from Jan Miller's Air Sparged Hydrocyclone. Studies of the Air Sparged Hydrocyclone (ASH) have shown the ability to achieve comparable yields and grades to typical conventional and column flotation units, but fundamentally have the potential for plugging of the sparging system.

Three centrifugal flotation techniques were evaluated: flotation cyclone with fixed pedestal dimensions and cavitation sparging system, flotation cyclone with variable pedestal dimensions and cavitation sparging system, and flotation cyclone with variable pedestal dimensions and tangential aeration sparging system. These designs potentially provide adequate upgrades over not only traditional flotation units, but also improve innovative centrifugal flotation techniques. Evaluating the three designs were based on determining equilibrium operating and design parameters which created a satisfactory froth product. This research evaluates the performance of the designs by concentrating coal from raw flotation feed.

1.3 Organization

This thesis is comprised of five major components describing the purpose of the research and how it was performed. The previous introductory section details the fundamentals of traditional flotation systems, the associated limitations, and how this research will potentially overcome those limitations.

The literature review section presents the status of current technology in both the conventional and column flotation and also the developments of centrifugal separation technologies. The subsections of the literature review include the limitations of conventional and column flotation, the advantages of centrifugal forces, and current centrifugal flotation technologies. The conventional and column flotation section presents the current trends of flotation cell size, required residence times, and fine particle flotation. The centrifugal forces subsection details the mechanics behind liquid in centrifugal motion and how particles in this field are subjected to specific forces which will aid in the flotation process. The centrifugal flotation technology sections discuss the innovative developments that attempted to fill the gaps of the traditional flotation practices with the aid of centrifugal motion.

The experimental section provides a detailed description of the samples, apparatus, and testing procedures used in this research. The section contains relevant information about operating and analysis equipment that was specifically used to evaluate the performance of the flotation cyclone designs.

The fourth section provides results from the experimental testing of the flotation cyclone designs and provides a discussion of the results. The discussion of the flotation cyclone considers the comparability of the three designs to current flotation technologies and any advantages that were discovered during the design testing.

The fifth and final section is an overview of the research project while providing recommendations for future work of this project and the future research in the area of centrifugal flotation technologies.

2.0 LITERATURE REVIEW

2.1 Scope

The literature review covers three topics relevant to this project: flotation technologies in current practices, centrifugal forces, and centrifugal flotation technologies. The first section presents the limitations of the traditional flotation technologies used in today's mineral processing applications. The second section covers the fundamental mechanics behind centrifugal forces since this is the main basis for the research. The third and final section describes the development of current centrifugal flotation technologies and identifies the limitations of those technologies.

2.2 Flotation Limitations

Although conventional and column flotation has developed into the most widely utilized separation technology in the mineral processing industry, like other methods there are operational limitations which can hinder efficiency. With respect to this project, one of the limiting factors of flotation is the continual increase of cell volume and associated slurry residence time. Described in the early flotation kinetics model, the recovery of a mineral with a given flotation constant is reliant on the particle residence time. The longer a particle resides in the active flotation collection zone, the higher probability the particle will be recovered. This realization in flotation kinetics paired with the continual decrease in ore grades and particle size has led to the cooperation between plant operators and manufacturers to install large volume flotation cells.

Flotation feeds typically are between two and five percent solids by weight which results in significantly large volumetric feed rates (Honaker, Kohmuench, & Luttrell, 2013). As the production of minerals, metals, and coal continues to increase, an increase in cell volume will be a likely consequence in order to satisfy the flotation feed standards thus maintaining efficiencies. A plot created by Noble details the trend in flotation cell size over the past century (Figure 2.1). As flotation cells started at volumes below one cubic meter, developments in the efficiency of the flotation process has led to the exponential growth to volumes surpassing 100 cubic meters and nearing 1000 cubic meters (Noble, 2013).

Several manufacturers offer large volume conventional and column flotation cells in order to meet required slurry residence times and increased capacities. FL Smidth, a mineral processing solutions company based in Utah, boasts the installation of 66 of their 250 cubic meter Wemco Flotation Machines in Mexico and developed 350 cubic meter SuperCells for a

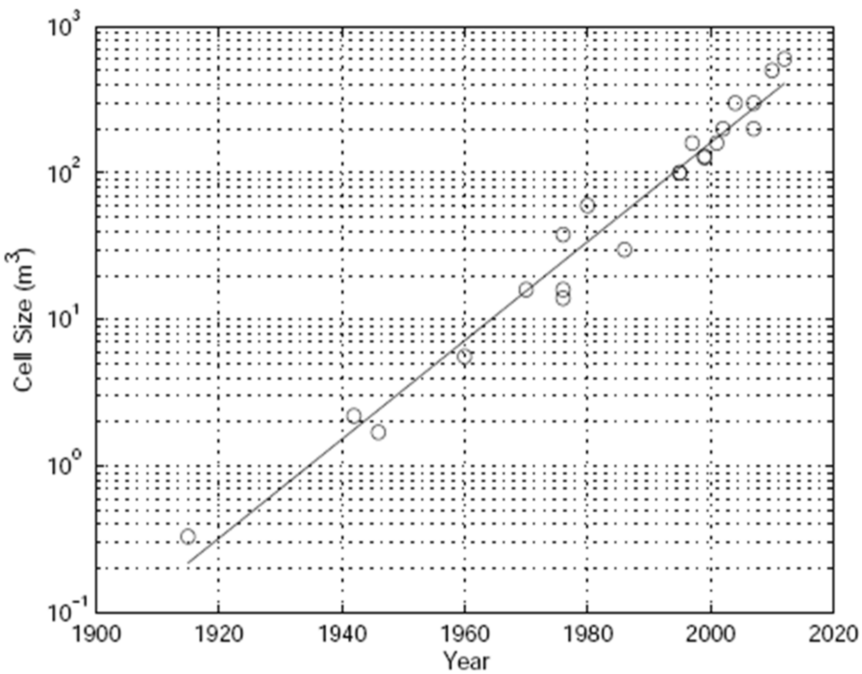


Figure 2.1. Trend of Flotation Size in Past Century (Noble, 2013). Noble, C. (2013). Analytical and Numerical Techiques for the Optimal Design of Mineral Separation Circuits. Blacksburg: Virginia Tech. Used with permission from author. Letter attached.

copper operation in the United States (Flotation Technology, 2010). Metso Minerals offers flotation columns whose height is designed to give a slurry residence time between 10 and 20 minutes (High Recovery Flotation Column, 2014). In addition, Outotec's TankCell flotation machines allow 500 cubic meters of effective volume for the high recovery of fine particles (Outotec, 2014). These flotation technologies offer benefits of high capacities, high efficiencies, and low costs but ultimately suffer from the required long residence times to achieve quality products.

The flotation process can be configured, as shown, as a time rate process in which particle will be recovered at a specific rate based on size, composition, and hydrophobicity. In correlation with the larger scale flotation cells, the retention times of the slurry is a critical factor in the allowance of adequate time for the desirable particles to collide, attach, and rise with the produced bubbles and is shown by the following equation:

$$t = V/Q \tag{2.1}$$

Where t is residence time, V is flotation cell volume, and Q is the volumetric flow rate to the cell. Luttrell *et al.* showed the effects of reducing retention times on recovery in a coal column flotation cell (Figure 2.2). These three different feeds achieved approximately 80 percent recoveries when the retention time was adequate, but when feed rate was increased, the recovery fell significantly in comparison to its theoretical values (Luttrell, Kohmuench, Stanley, & Davis, 1999).

Additional studies have been conducted recently to determine the efficiency of multiple stage column flotation and varying circuit configurations. Dennis Kennedy performed tests on single stage and two stage column circuits in addition to comparing the recovery results of changing a plant flotation circuit from a parallel circuit to a series circuit. Initial tests of the

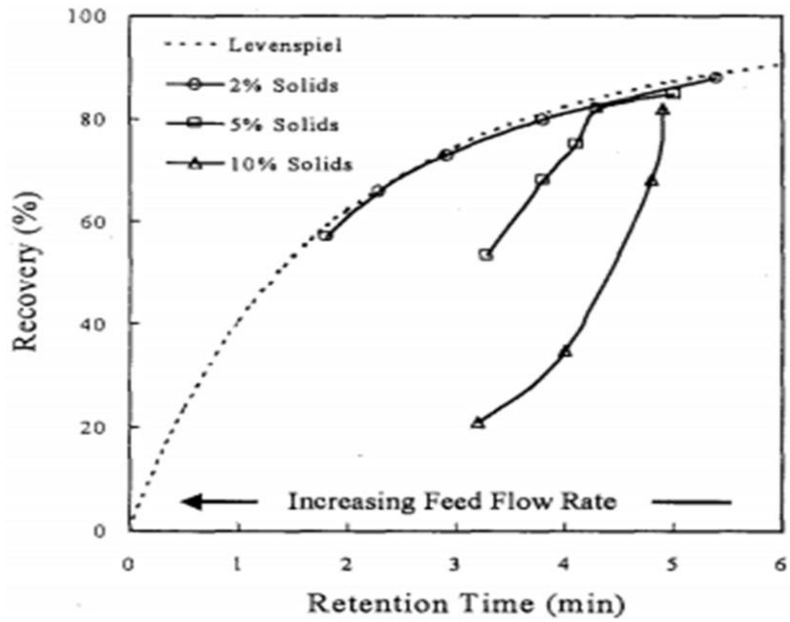


Figure 2.2. Effects of Retention Time on Recovery (Luttrell, Kohmuench, Stanley, & Davis, 1999). Luttrell, G., Kohmuench, J., Stanley, F., & Davis, V. (1999). Technical and Economic Considerations in the Design of Column Flotation Circuits for the Coal Industry. SME Annual Meeting. Denver: Society for Mining, Metallurgy, and Exploration. Used with permission Steve Kral, Editor, from Society of Mining, Metallurgy, and Exploration. Letter attached.

differences between a single stage and two stage circuit, with respective retention times of 11.9 and 12.9 minutes, showed an increase from 74 percent recovery to nearly 80 percent when a two stage circuit was implemented (Kennedy, 2008). These initial results led to the reconfiguring of an in plant columns cell from a parallel to a series circuit. Instead of distributing the feed among five columns, the feed was split to two columns and the tailings were reprocessed in the remaining three columns. The change resulted in a recovery increase from 77 percent to approximately 82% (Kennedy, 2008). Although these column circuit variations result in higher recoveries and the same retention time as Kennedy shown, the increase recovery required an increase volume. The increase in volume presents the overall issue with both column and conventional flotation where high residence times and capacities are required to achieve desired products.

Although not completely dependent flotation variables, the increase in flotation volume can be attributed to the presence of finer size fractions in flotation feeds. Sutherland derived an equation to adequately represent the rate of flotation that is dependent on particle size:

$$k = 3\pi\theta \operatorname{sech}^2\left(\frac{3V\lambda}{4R}\right) RrVN' \qquad [2.2]$$

where R is bubble size, r is particle size, λ is induction time, V is bubble velocity relative to particles, N' is the number of bubbles per unit volume of pulp, and θ is portion of particles retained in the froth after fruitful collision (Sutherland, 1948). This characterization indicates that a decrease in particle size will result in a decrease in flotation rate, which has been shown in several experiments through the development of flotation technologies.

Figure 2.3 provides a representation of flotation rates that Fuerstenau collected from several authors as the particle diameter varies (Fuerstenau, 1980). One significant reason for this is the small mass of particles that lack the overall momentum to deviate from the fluid stream lines surrounding the rising bubble and result in a collision needed for attachment.

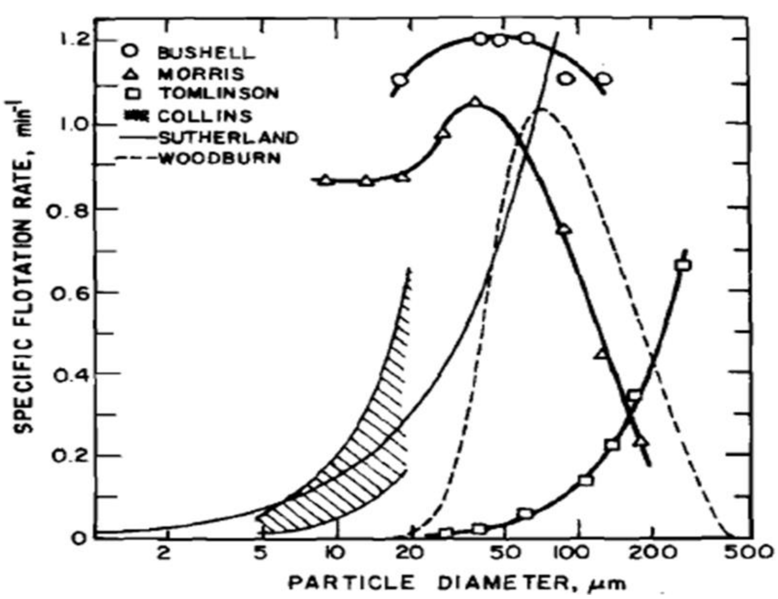


Figure 2.3. The Effects of Varying Particle Diameter on Flotation Rates (Fuerstenau, 1980). Fuerstenau, D. (1980). Fine Particle Flotation. In P. Somasundara, Fine Particles Processing (p. 671). SME-AIME. Used with permission from Jane Oliver, Manager of Book Publishing for Society of Mining, Metallurgy, and Exploration. Letter attached.

2.3 Centrifugal Flotation Separations

2.3.1 Potential Benefits of Centrifugal Flotation

Centrifugal separation is utilized in various unit operations of mineral processing applications including size separation, density separation, and thickening. Although the objective of centrifugal separation in each operation is different, each operation uses centrifugal forces to achieve the desired objective of accelerated or hindered settling. Mineral particles suspended in an aqueous solution are affected by the centrifugal forces developed during equipment operation. For the purpose of this research, the theory of sedimentation and centrifugal forces applied to particles will be discussed.

Hydrocyclones are high capacity and continuous operating units that induce centrifugal forces to accelerate the separation of particles of various sizes or densities (Figure 2.4). In most

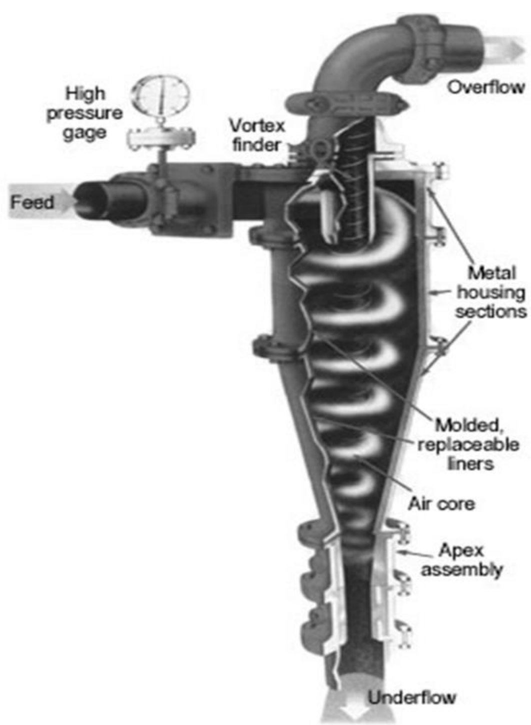


Figure 2.4. Krebs Hydrocyclone Cutaway (FL Smidth). Flotation Technology. (2010). Retrieved from FL Smidth: http://www.flsmidth.com. Used with permission from Andrew Cuthbert, Director of Global Marketing at FL Smidth. Form attached.

applications, slurry is fed tangentially into what is known as a feed chamber or “header”. The feed chamber consists of a vortex finder which serves as an outlet for overflow particles and a mechanism to induce the centrifugal forces. This centrifugal force developed by the entering slurry and assisted by the vortex finder is the driving factor behind accelerated sedimentation resulting in the size or density separation of particles. An apex assembly located at the bottom of the hydrocyclone provides an atmospheric connection and flow restriction to aid the swirling slurry in constructing an air core and transporting material through the vortex finder (Wills & Napier-Munn, 2006). These characteristics are the appealing factors of a flotation unit that induces centrifugal forces and creates a zero pressure zone like a hydrocyclone.

To understand how these forces are developed and their associated impacts in the flotation cyclone, an analysis can be performed by observing a simple particle moving with liquid in a circular orbit (Figure 2.5). As shown a particle moving in a circular motion experiences two forces exerted on it: a drag force by the liquid acting as resistance pulling the particle towards the center of its motion and the centrifugal force which is moving the particle away from the center of its motion. The force due to gravity will be neglected for simplicity of

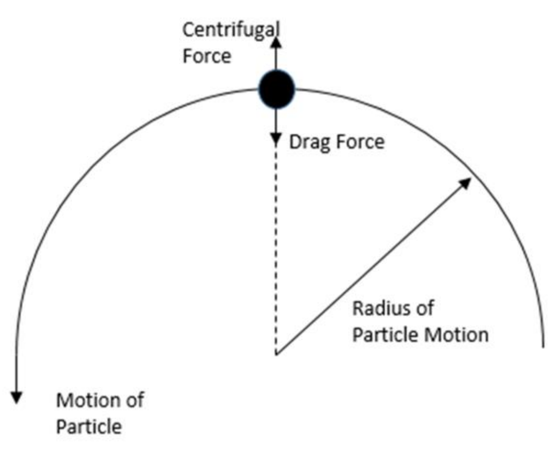


Figure 2.5. Forces Developed in Particle in Circular Motion.

this explanation. In 1981, George Stokes developed an equation for settling velocity of a spherical particle in fluid which can be seen below (Stokes, 1981):

$$v = {gd^2(D_s - D_f)}\Big/{18\eta} \qquad [2.3]$$

where v is terminal velocity, g is acceleration, d is particle diameter, D_s and D_f are particle and fluid densities, and η is fluid viscosity. In the scenario of circular motion, the acceleration term can be substituted by the centrifugal acceleration term given by the product of the particle tangential velocity squared and the inverse of its radius of motion (Holdich, 2002)., i.e.:

$$a_{centrifugal} = {v_t}^2\big/r \qquad [2.4]$$

Thus giving,

$$v = {v_t}^2 d^2(D_s - D_f)\Big/18\eta r. \qquad [2.5]$$

The application of Stokes' settling equation to a centrifugal motion scenario can have added benefits to the design of a flotation cyclone. Stokes' equation for a given application where acceleration, particle diameter, and fluid viscosity is constant, the settling velocity relies on the difference between the particle and fluid densities.

In addition to the application of Stokes' settling equation to centrifugal motion, previous work has been done specifically in the area of centrifugal separation in coal processing. Sokaski, Sands, and McMorris applied the sedimentation forces of dense medium baths to the centrifugal forces in cyclones (Sokaski, Sands, & McMorris, 1991). Sokaski *et al.* used the gravitational force found in typical dense medium vessels which is used in the processing of coarse coal particles shown in the following equation:

$$F_g = \frac{\pi d^3}{6}(\delta - \rho)g \qquad [2.6]$$

where d is the particle diameter, δ is the particle density, ρ is the liquid density, and g is the gravitational acceleration. Substituting the centrifugal acceleration equation in for the gravitational acceleration, g, the centrifugal force on a particle in a cyclone can be found (Sokaski, Sands, & McMorris, 1991):

$$F_c = \frac{\pi d^3}{6}(\delta - \rho)\frac{v^2}{r} \quad [2.7]$$

where v is the tangential velocity in the cyclone and r is the radius of the cyclone.

As previously described, flotation utilizes the ability of hydrophobic particles to attach to air bubbles and ascend to the top of a flotation column due to the lower density than the surrounding fluid. This fundamental principle of traditional flotation is the same principal used in centrifugal flotation designs. However, an added advantage of centrifugal flotation over the traditional flotation process is the presence of the centrifugal forces acting on the particles and increasing the settling velocity. As shown by Stokes and Sokaski *et al.*, an air bubble will be subjected to a force pulling the bubble with a specific velocity towards the air core at the center of the rotating motion. This is due to the bubble's density relative to the surrounding liquid which will potentially give the bubble particle aggregate a velocity towards the developed air core.

2.3.2 Current Centrifugal Flotation Technologies

As the need for smaller yet equally efficient flotation increases in the mineral processing industry, innovative technologies will continue to develop. Centrifugal flotation has been one developing technology in order to overcome the shortcomings of traditional flotation by reducing retention time and increasing flotation rate without sacrificing recovery. Each section provides both improvements of the Air Sparged Hydrocyclone, Imhoflot G Cell, and TurboFlotation over traditional flotation the associated limitations. In addition to the technology descriptions, any lab scale or pilot plant tests performed using the mentioned technologies are provided with procedures and results.

2.3.3 Air Sparged Hydrocyclone

Research towards developing the Air Sparged Hydrocyclone (ASH) began in the 1980's by Jan Miller and the Metallurgical Engineering Department at the University of Utah. The ASH unit was designed considering both traditional hydrocyclone features while using a novel sparging method in efforts to develop a high capacity, low volume fine particle flotation method (Figure 2.6). A tangential feed and vortex finder configuration is located at the top of the unit where slurry enters and develops a swirl flow. The swirl flow enters a porous cylinder which acts as the sparging system where the swirl flow shears the entering air to create bubbles for the

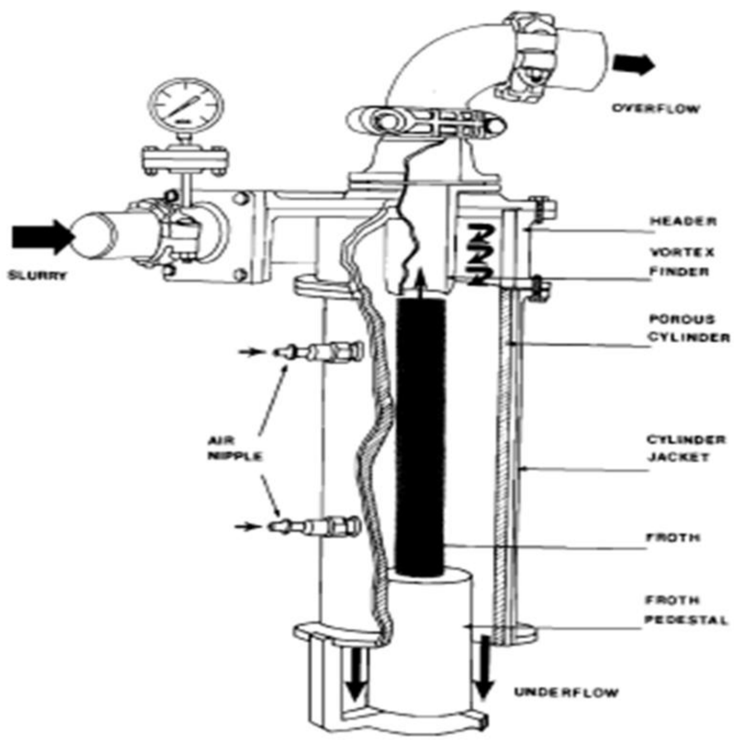


Figure 2.6. Air Sparged Hydrocyclone Assembly (Ye, Gopalakrishnan, Pacquet, & Miller, 1988). Ye, Y., Gopalakrishnan, S., Pacquet, E., & Miller, J. (1988). Development of the Air Sparged Hydrocyclone - A Swirl-Flow Flotation Column. Column Flotation '88 - Proceedings of an International Symposium (p. 9). Denver: SME. Used with permission from Jane Oliver, Manager of Book Publishing for Society of Mining, Metallurgy and Exploration. Letter attached.

attachment of hydrophobic bubbles. The swirl flow continues to travel until exiting the unit through an annular discharge at the bottom of the hydrocyclone with the rejected hydrophilic particles (Ye, Gopalakrishnan, Pacquet, & Miller, 1988). The mechanics and features behind the ASH unit will be discussed in length in subsequent sections.

The utilization of centrifugal forces aids in the flotation of ultrafine particles that are characterized by low flotation rates. In the early developments of the ASH unit, Van Camp studied the effects of force fields surrounding the flotation of a particle while Miller *et al.* derived expressions to determine the critical particle size that exhibits insignificant inertia to attach to a bubble. Van Camp concluded the relationship between flotation rates and force fields are directly proportional. In traditional flotation columns, the force acting on bubbles and particles is due to gravitational acceleration thus increasing their respective flotation rates by a unity factor (Van Camp, 1981). Miller tested the effects of increasing force fields and the resulting effects lowered the critical particle diameter with the required inertial momentum to collide with rising bubbles (Miller, Kinneberg, & Van Camp, 1982). The work done by Miller *et al.* and Van Camp provided the base theory for the Air Sparged Hydrocyclone unit by providing evidence that centrifugal forces will increase the flotation rate constant and shift the effective size range of the flotation process towards the finer size fractions.

To help induce centrifugal forces, the Air Sparged Hydrocyclone clearly resembles the typical hydrocyclone in mineral processing applications and utilizes a similar header unit. The traditional hydrocyclone header consists of two features: a tangential inlet and a vortex finder. Both features help create the swirl flow needed in the ASH unit while the vortex finder serves a dual purpose as the overflow discharge. The tangential inlet allows the slurry feed to enter the Air Sparged Hydrocyclone and induce rotational flow at a radius typically equal to the hydrocyclone radius. The vortex finder acts as a fixed point for the slurry to rotate about and extends down into the unit to continue the rotational flow and prevent short circuiting of the feed (Miller & Kinneberg, Fast Flotation with an Air Sparged Hydrocyclone, 1984).

As stated, the Air Sparged Hydrocyclone develops a swirl flow that shears air from a porous cylinder in efforts to collide with hydrophobic particles and be recovered (Figure 2.7). These particles that attach to the generated bubbles move radially towards to the center axis of the hydrocyclone and develop a froth phase. The froth phase moves axially through the vortex finder and exits the unit as product (Ye, Gopalakrishnan, Pacquet, & Miller, 1988).

The generation of the froth core is assisted to the presence of a zero pressure zone similar to that of an air core in a typical hydrocyclone that is created through the restriction or stabilization of the swirling slurry. Through the experiments of Ye *et al.*, controlling the characteristics of the froth core depended on the annular opening, overflow diameter, and the presence of new hydrophobic minerals (Ye, Gopalakrishnan, Pacquet, & Miller, 1988). The underflow area restricts the slurry flow inside the hydrocyclone thus forcing the layer towards the zero pressure zone and out the overflow. In addition, emulating a traditional hydrocyclone,

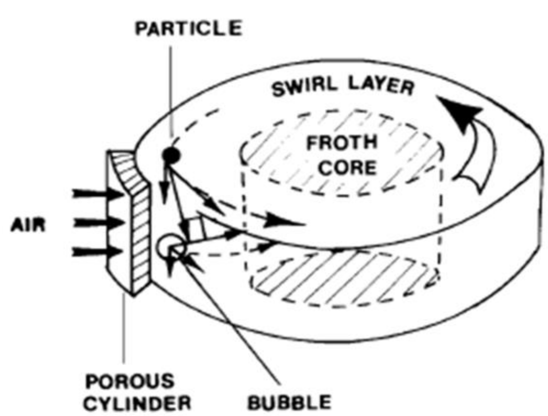


Figure 2.7. Varying Phases of Air Sparged Hydrocyclone (Ye, Gopalakrishnan, Pacquet, & Miller, 1988). Ye, Y., Gopalakrishnan, S., Pacquet, E., & Miller, J. (1988). Development of the Air Sparged Hydrocyclone - A Swirl-Flow Flotation Column. Column Flotation '88 - Proceedings of an International Symposium (p. 9). Denver: SME. Used with permission from Jane Oliver, Manager of Book Publishing for Society of Mining, Metallurgy and Exploration. Letter attached.

an increase in overflow diameter also increases the size of the froth core and consequently increasing the overflow rate. Lastly, in order to increase the amount of new hydrophobic particles to the froth layer, increased reagent concentration or gas flow rate was necessary (Ye, Gopalakrishnan, Pacquet, & Miller, 1988). The froth phase experiments concluded the benefits of both flotation and centrifugal separation units as the presence of a zero pressure zone allows the froth to build and stabilize while controlling such factors which are often found in typical hydrocyclone units.

The sparging method of the Air Sparged Hydrocyclone is the stand out feature which enables centrifugal flotation for the unit. The sparging cylinder is comprised of micrometer sized pores to create air capillaries and allow the air to enter the hydrocyclone (Miller & Kinneberg, Fast Flotation with an Air Sparged Hydrocyclone, 1984). These pores can vary in size as Miller and Kinneberg performed tests using 340 and 630 micron pore sizes while Lelinski, Bokotko, Hupka, and Miller performed tests with pore sizes ranging between 20 and 90 microns (Lelinski, Bokotko, Hupka, & Miller, 1996). The air traveling through these capillaries is sheared by the swirl flow of the unit and a distribution of bubbles varying in size is generated. These bubbles generated travel towards the zero pressure zone in the unit due to their low density relative to the slurry (Miller & Kinneberg, Fast Flotation with an Air Sparged Hydrocyclone, 1984). This travel distance is the main probability for attachment of hydrophobic particles to the bubbles, and varying certain operating and design parameter of the ASH unit can increase the attachment probability by bubble concentration, stability, and size.

The critical parameters which make bubble generation possible have been significantly studied throughout the development of the Air Sparged Hydrocyclone. The study of critical parameters done by Lelinksi *et al*. in 1996 aimed to numerically analyze the effects of frother concentration, slurry flow rate, and pore size on average bubble size and bubble size distribution (Lelinski, Bokotko, Hupka, & Miller, 1996). A fine (20-40 micron), medium (40-60 micron), and coarse (70-90 micron) porous tube were used in the experiment to study the effects of pore size on bubble generation while shear force effects were examined at flow rates between 35 and 70 liters per minute. For a study of the surfactant effects, sodium dodecyl sulfate at concentrations between 0 to 10^{-3} moles were tested. At a fine pore size and constant flow rate, the average bubble diameter reduced from 1028 microns to 286 microns. At constant porous tube size and surfactant concentration, increasing the flow rate thus increasing the shear force reduced the average bubble diameter from 1028 microns to 755 microns. Lastly, by varying the porous tube size while holding other factors constant, it can be seen the average bubble size reduces from 1028 to 838 microns (Lelinski, Bokotko, Hupka, & Miller, 1996). In addition, at high flow rates and surfactant concentrations, the porous tubes were able to produce bubbles distributions with an average size of 161 microns. The work done by Lelinski *et al.* showed the bubble generation method in the Air Sparged Hydrocyclone unit was controlled by the same factors as the traditional flotation unit and the additional control of the flow rate. In conclusion, the ASH unit was able to achieve small bubble sizes which are necessary for the flotation of fine particles.

To compare the Air Sparged Hydrocyclone with traditional flotation technologies, a considerable amount of pilot scale units with various ores have been tested to analyze the feasibility of the unit. One of the earliest evaluations of the Air Sparged Hydrocyclone was done by Miller and Van Camp in 1982 to separate a traditional water only cyclone coal feed (-28 mesh) with 22-25 percent ash content from Cerro Marmon Coal Group in Pennsylvania (Miller & Van Camp, 1982). This coal feed was primarily comprised of the finer size fractions with 50

Table 2.1. Parameters for ASH Unit in Cerro Marmon Coal (Miller & Van Camp, 1982)

Air Sparged Hydrocyclone Parameters		
Air Flow Rate	6.6	L/s
Frother Concentration	20	ppm
Feed Rate	4	L/s
Percent Solids	3	%
Cyclone Diameter	6	in
Porous Cylinder Length	29	in

to 70 percent of the particles finer than 400 mesh. The operating conditions and design parameters of the ASH unit used in the coal testing is shown in Table 2.1.

Miller and Van Camp compared the results from testing the Air Sparged Hydrocyclone with a batch flotation test using a conventional flotation cell. The conventional bench cell was treated with 0.25 kg/ton of frother and product was collected for two minutes. In comparison of product ash contents, the Air Sparged Hydrocyclone separated a product with 16 percent ash at a 75 percent yield rate at a 3.35 second retention time while the batch scale flotation produced a 15.5 percent ash product at a 67 percent yield rate (Miller & Van Camp, 1982). The comparison proves the improvement of the ASH unit over the traditional conventional flotation cell as the hydrocyclone obtained the same product quality at a higher yield and a lower retention time.

In addition to the comparison of the ASH unit and batch flotation, the effects of air flow rate and porous cylinder length were evaluated. The separation efficiency was evaluated at cylinder lengths of 16 and 29 inches and air flow rates of 200 and 400 lpm using 20 ppm frother, 3 percent solids, and 240 lpm feed rate. Increasing the air flow rate from 200 to 400 lpm produced the same quality product at 16 percent ash while increasing the yield from 52 to 75 percent (Miller & Van Camp, 1982). The increased air flow rate acted in the fashion a typical flotation unit would as the yield should typically increase. However, an increase in the product ash is usually associated with this change in air flow rate but the ash remained unchanged as proven. Like the higher flow rate, increasing the cell length from 16 inches to 29 inches improved the product quality from 22.5 percent ash down to 16 percent ash while increasing the yield by 16 percentage points. The higher quality product at an increased yield rate is likely due to the increased retention time (Miller & Van Camp, 1982). Although the increasing the cell length negates the objective of the ASH unit, the retention time in comparison with the two minute conventional batch cell test was still significantly lower. However, the air flow rate for the conventional cell was not listed but as described, a 400 liter per minute air flow rate was required to achieve concentrate values which could be a disadvantage of the system.

The effectiveness of the ASH unit in the separation of the various size fractions was evaluated and compared to other traditional flotation and separation technologies (Table 2.2). It can be seen that the evaluation of the Air Sparged Hydrocyclone significantly held the advantage over the ash removal capabilities throughout the various size fractions of water only cyclones and single stage flotation.

Although testing the Air Sparged Hydrocyclone unit with coal slurry was a simple yet effective evaluation, flotation is a separation technology with superior efficiencies throughout the

Table 2.2. Ash Removal Comparisons between Separation Technologies (Miller & Van Camp, 1982). Miller, J., & Van Camp, M. (1982). Fine Coal Flotation in Centrifugal Field With an Air Sparged Hydrocyclone. SME Mining Engineering, 1575-1580. Used with permission from Steve Kral, Editor for Society of Mining, Metallurgy, and Exploration. Letter attached.

Ash Removal (Percent)					
Mesh Size	WOC	Single Stage Flotation	Air Sparged Hydrocyclone Unit		
			Illinois 6	Beaver Creek	Lower Kittaning
28x100	60-65	50-60	77-86	84.2	70.4
100x200	40-45	40-45	59-68	70.2	39.6
200x325	15-18	40-45	44-62	77	42.5
-325	0-5	50-55	57-67	81	54

various mining industries. Continuous testing of the system has been performed with a variety of ores to evaluate the versatility. In efforts to demonstrate the capabilities of the Air Sparged Hydrocyclone, Miller *et al.* performed an evaluation with fine gold ore in sand obtained from the Colorado River in Utah (Miller, Misra, & Gopalakrishnan, 1985). Miller *et al.* performed several tests with the ASH unit with specific design and operating variables (Table 2.3) and compared the results with a ten minute conventional batch flotation experiment.

The feed used in the testing was obtained from a gravity plant and was considered tailings at minus 28 mesh with a majority of the feed in the minus 400 mesh size fraction. An analysis on the feed proved the ore contained gold concentrations of 0.01 to 0.02 ounces per metric ton. This feed was separated into a concentrate and reject stream using the Air Sparged Hydrocyclone and batch conventional cell. Both the ASH unit and the conventional cell were operated at the same operating parameters and the results were compared (Miller, Misra, & Gopalakrishnan, 1985). It should be noted the air flow rate to the ASH unit was 100 lpm while

Table 2.3. Design and Operating Variables for Gold Flotation (Miller, Misra, & Gopalakrishnan, 1985). Miller, J., Misra, M., & Gopalakrishnan, S. (1985). Fine Gold Flotation From Colorado River Sand with the Air Sparged Hydrocyclone. SME-AIME. Albuquerque: Society of Mining Engineers of AIME. Used with permission from Steve Kral, Editor for Society of Mining, Metallurgy, and Exploration. Letter attached.

ASH Parameters		
Design Variables		
Cyclone Diameter	5	cm
Cyclone Length	52.5	cm
Pore Size	1	micron
Pedestal Diameter	4.25	cm
Operating Parameters		
Promoter	0.05	g/kg
Collector	0.08	g/kg
Frother	0.1	g/kg
Air Flow Rate	100	slpm
Solids Feed Rate	1	tph
Percent Solids	16	%

the bench test was 6 lpm.

The results showed the superiority of the Air Sparged Hydrocyclone over the traditional conventional flotation cell. In the testing of the Colorado River Sand containing fine gold particles, the ASH unit achieved a concentrate grade of 2.17 ounces per metric ton at a recovery of nearly 81 percent while the conventional batch cell only recovered 55 percent of the gold at a 1.075 opt grade (Miller, Misra, & Gopalakrishnan, 1985). Table 2.4 shows the complete comparison between two separation processes abilities on the gold ore. The comparison of the two separation methods proved the ASH unit recovered gold particles at a higher rate and higher quality than the conventional flotation method. The improvement showed a shift in the grade versus recovery curve towards obtaining the highest grade at the highest recovery rates possbile, but the discrepancy between air flow rates may show a flaw of the ASH unit.

As the development of the Air Sparged Hydrocyclone continued through various mineral industries on a laboratory scale, larger scale units were implemented into existing plants to evaluate the true feasibility of the technology. One of the major efforts to install the ASH unit into a processing facility occurred at a Florida phosphate operation where Miller cooperated with the Florida Institute of Phosphate Research. In this partnership, Miller *et al.* analyzed the operational feasibility of producing a high BPL (Bone Phosphate of Lime) by replacing a

Table 2.4. Comparison of Separation Results for Fine Gold Ore (Miller, Misra, & Gopalakrishnan, 1985). Miller, J., Misra, M., & Gopalakrishnan, S. (1985). Fine Gold Flotation From Colorado River Sand with the Air Sparged Hydrocyclone. SME-AIME. Albuquerque: Society of Mining Engineers of AIME. Used with permission from Steve Kral, Editor for Society of Mining, Metallurgy, and Exploration. Letter attached.

	Comparison of Flotation Methods on Gold Ore					
	Air Sparged Hydrocyclone			Conventional Batch Flotation		
Stream	Weight %	Grade	Recovery	Weight %	Grade	Recovery
Concentrate	0.39	2.17	80.98	0.7	1.075	55.81
Tail	99.61	0.002	19.02	99.3	0.006	44.19
Feed	100	0.01	100	100	0.014	100

rougher cleaner circuit installation with a single stage six inch Air Sparged Hydrocyclone. The results from the tests proved the ASH unit was able to achieve capacities 50 times the capacity of traditional flotation techniques and produce a 66 percent BPL concentrate at a recovery of 91 percent (Miller, Wang, Yin, & Yongqiang, 2001).

Although the installation of the single stage ASH unit was efficient in producing a quality BPL concentrate, the major concern of the Air Sparged Hydrocyclone resulted from the phosphate testing. After operating continuously for 10 hours, the porous cylinder, the sparging method for the unit, started plugging as crud formed on the inner wall. This plugging interrupted the flow of air entering the system and ultimately lowered the recovery of the unit (Miller, Wang, Yin, & Yongqiang, 2001). Significant studies were performed after the installation to test various porous cylinders to fix the plugging problem. Plastic, ceramic, stainless steel, stainless steel wire mesh, and hydrophilic plastic porous cylinders were examined and all materials suffered from the plugging issue. The stainless steel tube's permeability decreased to 57% of its original value after 16 hours of operations and was the least impacted material by the crud (Miller, Wang, Yin, & Yongqiang, 2001). This problem leads to the additional motivation of this research in order to develop a sparging method that prevents plugging issues during continuous operation.

2.3.4 Imhoflot G Cell

The Imhoflot G Cell is a novel concept developed by Rainer Imhof at Maelgwyn Mineral Services to recover ultrafine particles using centrifugal forces and minimal retention times (Figure 2.8). The device exposes the flotation processes (aeration, bubble particle contact, and froth pulp separation) into single operations and combines it into one unit. Slurry containing ultrafine particles enters a downcomer unit where the slurry is aerated with multi-jet venturi systems where the slurry shears the air and produces micrometer sized bubbles. The aerated

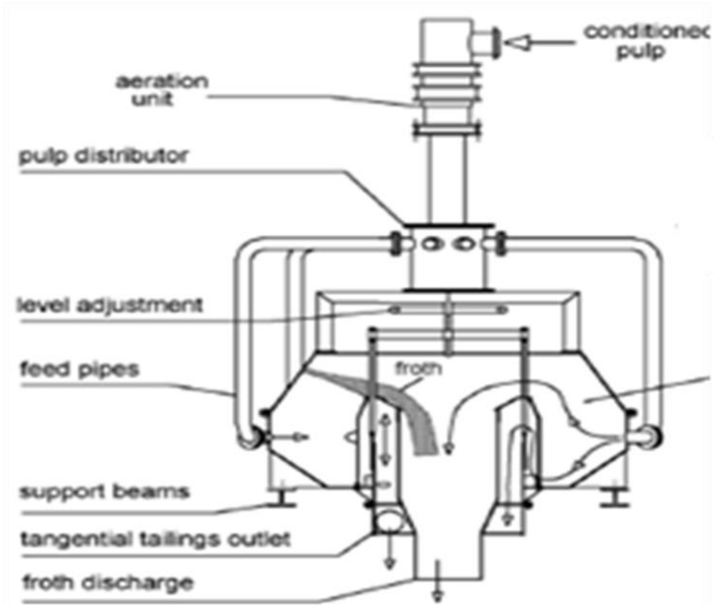


Figure 2.8. Pictorial Description of Imhoflot G Cell (Battersby, Brown, & Imhof, 2003). Battersby, M., Brown, J., & Imhof, R. (2003). The Imhoflot G-Cell - An Advanced Pneumatic Flotation Technology for the Recovery of Coal Slurry From Impoundments. Cincinatti: Society for Mining, Metallurgy, and Exploration. Used with permission from Jane Oliver, Manager of Book Publishing for Society of Mining, Metallurgy, and Exploration. Letter attached.

slurry splits into multiple ports that enter the cell tangentially to create centrifugal accelerations ranging between 10 and 30 m/s^2. The centrifugal motion accelerates the froth and pulp separation as the froth travels towards the center of the cell and exits axially while the primarily hydrophilic pulp exits from the bottom of the cell. With this design, the G Cell is able to achieve retention times between 25 and 30 seconds (Battersby, Brown, & Imhof, 2003).

Several implementations of the Imhoflot G Cell in the mineral processing industry proved its efficiency over the previous flotation process. A three stage G Cell configuration was installed over a conventional rougher cleaner circuit at a kaolin processing operation where particle sizes of a few microns exist. With three 1.8 meter G-Cells at a capacity of 110 cubic meters per hour, the circuit was able to achieve a kaolin concentrate at a 7 percent increase in recovery but a 0.4 percent decrease in grade. The reported retention time for the three cells was 120 seconds versus the 14 minutes required for the previous rougher cleaner circuit (Imhof, Fletcher, Vathavooran, Singh, & Adrian, 2007).

In further efforts to determine effectiveness, a fine coal impoundment was treated with the Imhoflot G Cell. Fine coal impoundments are typically comprised of minus 325 mesh particles due to the inabilities of traditional flotation techniques to recover that size fraction. A two stage G cell flotation circuit yielded a 12 percent ash product from a 32 percent ash feed. The two stage circuit produced a quality grade product at a yield of 70 percent from the original feed of 40 tonnes per hour (Battersby, Brown, & Imhof, 2003). These applications of the Imhoflot G Cell not only prove the technology's performance against conventional flotation techniques but prove the idea of centrifugal flotation can be just as effective. However, the G Cell may suffer from the same flaws of conventional and column cells where multiple stages are needed which increase the required volume of the circuit.

2.3.5 TurboFlotation

The TurboFlotation system, developed by the Commonwealth Scientific and Industrial Research Organization (CSIRO), is a compact flotation technology that isolates the froth separation from the bubble particle contact and bubble generation zones and utilizes centrifugal forces to speed up the pulp froth separation. The system consists of several individual components which helps optimize the unit flotation processes: jet ejector for bubble generation, motionless mixer to induce bubble particle contact, and a centrifugal flotation cell to separate the hydrophobic froth from the hydrophilic pulp (Figure 2.9). The jet ejector creates a low pressure zone by increasing the liquid velocity thus drawing air into the slurry and creating bubbles. The static inline mixer creates turbulent conditions to allow for the bubbles and particle to collide. The slurry enters the separation cell tangentially to induce centrifugal forces and speed up the flotation rate of the bubble particle aggregate (Ofori, Firth, & Howes, 2000).

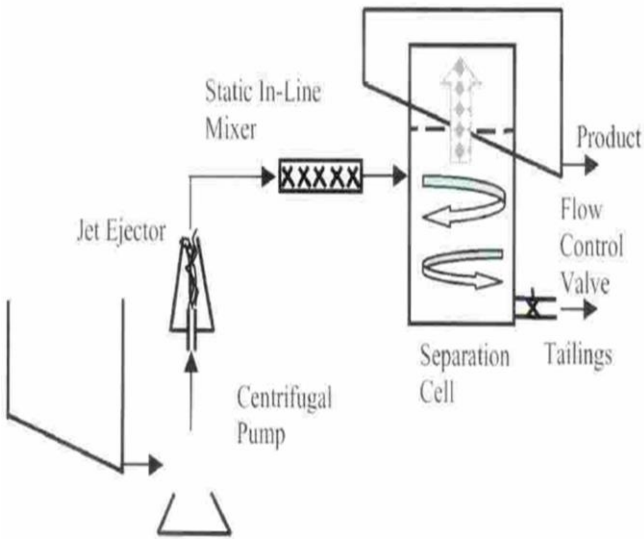


Figure 2.9. TurboFlotation System (Ofori, Firth, & Howes, 2000). Assessment of the Controlling Factors in TurboFlotation by Statistical Analysis – Coal Preparation – Volume 21 Issue 4 PP 355-382 – Authors – P. K. Ofori, B. A. Firth & T. Howes. Used with permission of Deborah East, www.tandfonline.com, 2014.

The developments of the TurboFlotation system have shown the technology's ability to achieve 7-9% ash content products from a coal feed containing 23% ash (Ofori, Firth, & Howes, 2000). Initial efforts of testing the operating parameters in coal flotation, a two liter separation cell was operated with a slurry feed rate from 10 to 16 liters per minute and achieved yield values ranging from 53 to 80 percent. The low volume separation cell achieved separation efficiencies as high as 64 percent at a retention time between 8 and 12 seconds (Ofori, Firth, &

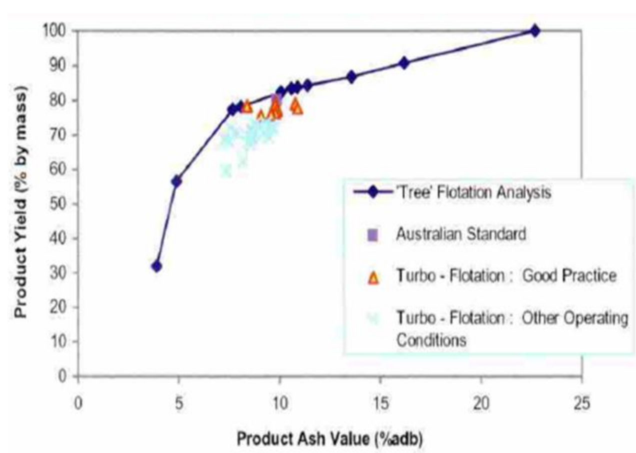


Figure 2.10. Pilot Plant TurboFlotation Yields and Ash Content (Ofori, Firth, & Howes, 2000). Assessment of the Controlling Factors in TurboFlotation by Statistical Analysis – Coal Preparation – Volume 21 Issue 4 PP 355-382 – Authors – P. K. Ofori, B. A. Firth & T. Howes. Used with permission of Deborah East, www.tandfonline.com, 2014.

Howes, 2000). Subsequent pilot plant testing of a TurboFlotation system compared the efficiency of the system with typical flotation results in Australia (Figure 2.10).

3.0 EXPERIMENTAL

3.1 Coal Samples

In order to effectively test the designed prototypes for flotation capabilities, coal samples were obtained from a currently operating coal plant. The natural hydrophobicity of coal makes is an ideal testing material as it requires little modification in the flotation process. The sample used in the current test program was provided from a facility located in southwestern Virginia. The preparation plant produces both thermal and metallurgical coal while serving several different mines in the area. The plant is equipped with two Microcel flotation columns that each treat run-of-mine (ROM) minus 100 mesh feed. The minus 100 mesh coal was 48.30% ash and pre-treated with collector at 40 ml/min. The feed release analysis and column data were provided by the company in Tables 3.1 and 3.2.

Table 3.1. Release Analysis of the Coal Sample.

	Individual		Cumulative Float		
Float Product	**Mass (%)**	**Ash (%)**	**Mass (%)**	**Ash (%)**	**Rec. (%)**
C1	9.01	6.95	9.01	6.95	16.21
C2	15.35	7.69	24.36	7.42	43.62
C3	16.24	9.32	40.6	8.18	72.1
C4	11.35	16.47	51.95	9.99	90.43
T2	2	65.66	53.95	12.06	91.76
T1	46.05	90.75	100	48.3	100

Table 3.2. In-Plant Performance of Column Cells.

	% Solids	% Ash
Feed	5.38	44.87
Conc	12.9	9.98
Tails	3.89	80.56
Yield %	50.57	
Recovery %	82.57	
Frother (ppm)	4.2	

3.2 Prototype Designs

In order to develop a system that incorporated centrifugal forces to aid in the flotation of fine particles, three designs were created following the operating and physical parameters of the Air Sparged Hydrocyclone, classifying cyclone, and flotation column. Although following similar operating and physical parameters, each design consists of a unique sparging method to provide bubbles for particle attachment. The subsequent descriptions will explain each of the design theories with the expected advantages over the Air Sparged Hydrocyclone.

3.2.1 Initial Design: Fixed Pedestal with Tangential U/F

The primary design of the flotation cyclone consists of a tangential inlet and a discharge ball valve with a pedestal of fixed diameter and height. The tangential inlet provides a feed port for the slurry where centrifugal forces can be generated similar to a hydrocyclone. The discharge ball valve provides a control for the slurry level inside the cyclone which can dictate the associated water split. The pedestal provides a source of resistance in the cyclone in order to develop an air core. In addition to acting as a source of resistance, the pedestal supports the air core which is where bubbles containing fine coal particles would report.

According to J.D. Miller, the best separation efficiency for fine coal in the Air Sparged Hydrocyclone occurred at an overflow area to underflow area ratio of 0.9 (Gopalakrishinan, Ye, & Miller, 1991). Therefore, a pedestal diameter of 3.5 inches is calculated based on the determined overflow and cyclone diameter. In addition, according to the Air Sparged Hydrocyclone patent, the pedestal height should be at least fifty percent of the cylinder length which results in a 12 inch pedestal height for the initial design (Miller, 1981). The vortex finder diameter and length were based on design equations:

$$VF_{Diameter} = 0.35 * D \qquad [3.1]$$

Table 3.3. Dimensions of Initial Flotation Cyclone Design.

Primary Design of Flotation Cyclone		
Inlet Dia.	1	in
Cylinder Length	24	in
Cylinder Dia.	3.786	in
Discharge Dia.	1	in
VF Dia.	1.5	in
VF Length	2.2	in
Pedestal Dia.	3.5	in
Pedestal Length	12	in

$$VF_{Length} = 0.55 * D \qquad [3.2]$$

where D is the diameter of the cylinder (Mular, 2003). In addition to the vortex finder dimensions, the 1-inch inlet diameter is based on the equation which relates the cylinder diameter to inlet area (Mular, 2003), i.e.:

$$A_{inlet} = 0.05 * D^2 \qquad [3.2]$$

The dimensions of the primary design are shown in Table 3.3. For reference, a schematic illustration of the test unit is shown in Figures 3.1 and 3.2.

The main difference between the Air Sparged Hydrocyclone and the initial flotation cyclone (Figure 3.1) is the sparging method. As stated, the sparged method in the ASH method is a porous cylinder where air is sheared by the rotating slurry. However, due to plugging of the porous cylinder, a two inch Cavitation Tube is placed before the inlet of the flotation cyclone. The Cavitation Tube is a novel sparging device design by Eriez Manufacturing which creates picobubbles by subjecting slurry to a constricted area. The slurry velocity increases to a point where the decrease in slurry pressure induces cavitation and bubbles are created and stabilized by the presence of frother (Column Flotation Systems Cavitation Tube, 2009). The Cavitation Tube is expected to provide an external sparging source which would provide adequate bubble particle attachment probability by generating picobubbles.

Figure 3.1. Initial Flotation Cyclone (Yan, 2013) Yan, E. (2013). Initial Flotation Cyclone. Eriez Flotation Division. Used with permission from creator. Letter attached.

Figure 3.2. Initial Flotation Cyclone Design in Operation.

3.2.2 Secondary Design: Adjustable Pedestal with Tangential and Axial U/F

The second flotation cyclone design incorporated not only potential advantages over the Air Sparged Hydrocyclone but also over the initial design of a flotation cyclone with a pedestal of fixed height and diameter. The sparging method remained the same with the Cavitation Tube providing picobubbles, but the secondary design changed the discharge method to a two point underflow system where slurry can exit tangentially or axially (Figure 3.3). The addition of the axial underflow makes it potentially possible to control the level of the rotating slurry and to install a pedestal whose height could be varied.

The adjustable pedestal provides an enhanced control factor to aid in the flotation of particles. In the initial design, the volume of the flotation column developed inside the hydrocyclone was limited by the fixed pedestal height and diameter. The fixed pedestal dimensions potentially leads to bubbles bypassing the pedestal thus missing the collection zone. The new design allows the pedestal to be adjusted to recover those potentially bypassing bubbles and also limit water reporting to the overflow. Also, the pedestal diameter can be varied which

Figure 3.3. Flotation Cyclone with Adjustable Pedestal and Two U/F Streams.

presents an advantage over the initial design through the ability to aid in developing an air core in changing situations.

The implementation of the axial underflow creates another source of control by dictating the level of the rotating swirl layer in the flotation cyclone. The axial underflow is a circular opening which acts similarly to an overflow weir of a flotation column. The theory is the level of fluid exiting the axial underflow will not exceed a diameter much smaller than the underflow. The diameter formed is expected to be uniform throughout the length of the cylinder thus providing a source of level control.

Table 3.4. Flotation Cyclone with Adjustable Pedestal and Axial Underflow.

Secondary Design of Flotation Cyclone		
Inlet Dia.	1	in
Cylinder Length	24, 12	in
Cylinder Dia.	3.786	in
A U/F Dia.	varies	in
VF Dia.	varies	in
VF Length	2.2	in
Pedestal Dia.	varies	in
Pedestal Height	varies	in

Other than the addition of the adjustable pedestal and axial underflow, the dimensions of the flotation cyclone remained the same from the original design. An additional shorter cylinder with the same characteristics is constructed to aid with the development of the air core. The dimensions are based from previous works of the Air Sparged Hydrocyclone which demonstrate the ideal parameters for the pedestal and axial underflow. The diameter and lengths for the vortex finder were given from the hydrocyclone equations previously stated. The dimensions for this design can be seen in Table 3.4, and the network configuration for the second flotation cyclone evaluation can be seen in Figure 3.4.

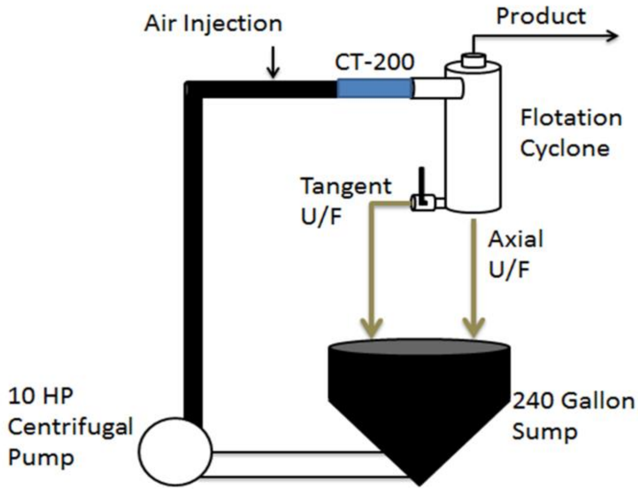


Figure 3.4. System Configuration for Second Flotation Cyclone Evaluation.

3.2.3 Tertiary Design: Tangential Aeration Chamber with Axial U/F

The third design of the flotation cyclone strays from the original two design ideas by following more closely to the operating principles and dimensions of the Air Sparged Hydrocyclone. The two important features that distinguish the third design are a tangential aeration cylinder for the sparging method and an involute feed (Figure 3.5). These new design features were expected to provide significant improvements in the aeration and swirl flow of the slurry.

The implementation of the tangential aeration cylinder aerates the slurry as it rotates along the cylinder wall. The design is similar to the Air Sparged Hydrocyclone sparging method which uses a porous membrane to inject the air into the rotating slurry. In order to avoid the

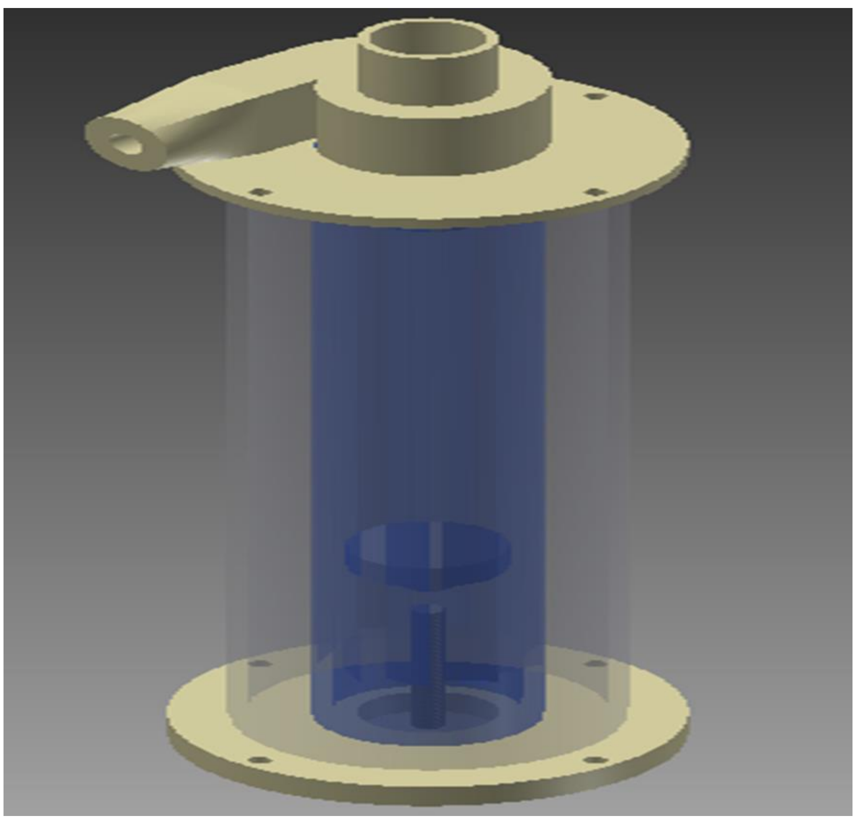


Figure 3.5. Third Flotation Cyclone Design.

previously stated plugging issue with the porous membrane, a series of tangential ports, one millimeter in width, are created in a 2-inch cylinder. The design of tangential ports allows the slurry to travel along the cylinder wall with minimal impedance as it shears through the air exiting the port. In addition, the tangential orientation of the ports will limit the potential for particles flowing into the port and prevent plugging.

Designing the flotation cyclone with an involute feed provides many advantages over the previous tangential feed design. The involute feed subjects the entering slurry to centrifugal forces before the slurry transitions into the aeration cylinder. These forces will exhibit a certain degree of separation between recoverable coal particles and gangue. The involute feed also limits the turbulence caused by inlet junction. The limitation of turbulence will allow the floated product to freely move into the vortex finder and out the flotation cyclone.

Although this design uses a different aeration and feed system, the third design still incorporates aspects from the second design. Included in this design is the adjustable pedestal where the height and diameter can be varied to fit the optimum conditions. The pedestal will serve the purpose of creating resistance in the swirl flow in addition to supporting the froth column generated. The tailings will be discharged through the axial underflow provided by the circular opening.

The dimensions of the third flotation cyclone are based on the Air Sparged Hydrocyclone used during phosphate testing by Jan Miller in partnership with the Florida Institute of Phosphate Research (Table 3.5). The third design was comparably modeled after the cylinder diameter, aeration length, and inlet area of the air sparged hydrocyclone. The pedestal and axial underflow dimensions will vary between tests to create best operation, but will mainly be determined from Miller's design parameters for the Air Sparged Hydrocyclone.

Table 3.5. Flotation Cyclone with Tangential Aeration Cylinder.

Tertiary Design of Flotation Cyclone		
Inlet Area*	0.5	in^2
Aeration Length*	13	in
Cylinder Dia.*	2	in
A U/F Dia.	varies	in
VF Dia.	1.25	in
VF Length	1.5	in
Pedestal Dia.	varies	in
Pedestal Height	varies	in

*Denotes dimensions of ASH in Phosphate Testing

In efforts to experiment with different sparging methods, the third flotation cyclone design with aeration chamber was modified. Instead of utilizing compressed air in the aeration chamber, an idea of filling the chamber with a slurry bubble mixture was developed. The justification is the swirl flow will draw the bubbles through the tangential ports and the swirling particles will collide with the entering bubbles. This opposes the original aeration chamber theory by individualizing the flotation unit processes. Slurry was drawn out of the 240 gallon sump using a ¾ horsepower pump, sent through a static mixer, and entered the aeration chamber (Figure 3.6). The bubbles generated by the static mixer will fill the aeration chamber and be

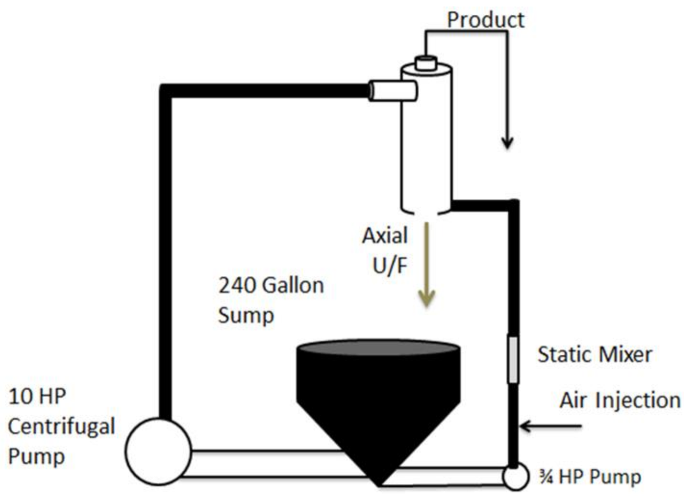


Figure 3.6. Tertiary Flotation Cyclone Design with Static Mixer.

subjected to the rotating slurry inside the smaller two inch cylinder. The rotating hydrophobic particles will attach with the bubbles entering the tangential ports, travel towards the zero pressure air core, and exit through the overflow as product.

3.3 Experimental Apparatus

3.3.1 Slurry Handling

In typical mineral processing applications, hydrocyclones, dense medium cyclones, and water only cyclones are fed with slurry through a network of sumps, centrifugal pumps, and piping. This network configuration helps plant operators have control over the feed rate to the associated hydrocyclones which is of great importance to the separation of particle sizes or densities. For the testing of all three flotation cyclone designs, the same network was constructed in the pilot plant area of the Mining and Minerals Engineer Research Facility. The network consisted of a 240 gallon sump equipped with a 10 HP centrifugal pump and one inch schedule 40 PVC pipe to transport the slurry to the flotation cyclone (Figure 3.7).

Figure 3.7. Slurry Handling Configuration.

3.3.2 Dewatering

Considering the objective of the flotation cyclone is to achieve similar recovery rates in comparison to column flotation cells, dewatering equipment was needed to obtain a dry sample for analyzing of the feed, product, and tailing streams. In order to dewater collected samples, a simple filter and vacuum pump configuration was utilized to produce a filter cake. The configuration consisted of a ¾ HP vacuum pump, a 4500 mL flask, a 1000 mL flask, and a filter (Figure 3.8). The ¾ HP pump pulls air thus creating a vacuum and allowing the water contained in the samples to flow into the empty flasks. In addition, to dewater the slurry completely, an industrial oven was used in the process to remove any residual moisture which was unable to be removed by the vacuum filter. The industrial oven allows the user to control the temperature and provides a heating range up to 250 degrees Celsius.

Figure 3.8. Vacuum Pump Dewatering Configuration

3.3.3 Ash Determination

Determining the ash content of the feed, product, and reject streams was critical in determining the performance of the three flotation cyclone designs. Provided at the research facility was the LECO TGA701 Thermogravimetric Analyzer and it assisted in the determination of the sample ash contents. The analyzer uses a controlled environment to determine composition of the materials of interest. There are multiple software programs that analyze weight loss of a material as temperature varies in the controlled environment. In the experiments of the flotation cyclone designs, the analyzer has an ash analysis program that measures the weight loss of combustible material in the samples as temperature increases (Figure 3.9).

3.3.4 Operational Controls and Measurements

In order to control and monitor the operating conditions for the experimental evaluation of the flotation cyclone designs, several control devices were installed. The controls were used to set desired levels of operating conditions such as slurry feed rate and air flow rate. Controlling the slurry flow rate was performed by a PLC instrument connected to the centrifugal pump motor (Figure 3.10). The controller regulated the speed of the motor which in turn controlled the rotating speed of the pump. A more manual approach was taken to control the air flow rate. An

Figure 3.9. LECO Thermogravimetric Analyzer.

Figure 3.10. PLC Motor Controller.

air regulator and ball valve combination was used to respectively control the pressure and volume of air entering the circuit.

In addition to the control mechanisms, measurement apparatuses assisted by providing an analog output of the operating condition. To measure the volumetric flow rate of slurry passing through the network configuration, a one inch Maglite Flow Meter was installed in the piping network (Figure 3.11). The Maglite Flow Meter sent the measured flow rate to an installed analog output device which displayed the system flow rate (Figure 3.12). The output device was configured to display readings up to 100 gallons per minute of slurry. As for the volumetric flow rate of air being injected in the system, a simple floating flow meter was installed after the regulator but before the ball valve. Multiple air flow meters were available to accommodate both high (6 – 60 cfm) and low (0.4 – 4 cfm) flow rates.

3.3.5 Flotation Chemicals

In order to increase to flotation rate, chemical reagents were added to the slurry before commencing the tests. For collector addition, kerosene, a typically used reagent in coal flotation, was added to the metallurgical coal samples. The amount of collector added depended on the solids content of the slurry but the normalized collector addition was sustained between 0.2 and 2

Figure 3.11. System Slurry Flowmeter.

Figure 3.12. Slurry Flowrate Analog Output.

lbs. of kerosene to one ton of solids. For the frothing reagent, Aerofroth by the American Cyanamid Company was used to generate bubbles and stabilize the created froth. The Aerofroth is a mixture of aliphatic hydroxylated hydrocarbons and was added to slurry in the 240 gallon sump at concentrations between 7 and 60 parts per million.

3.3.6 Miscellaneous Equipment

Throughout the experiments, miscellaneous equipment was used in junction for the analysis of the flotation cyclone designs. The most important tools, electronic scales, were mainly used for weighing wet and dry samples taken from the three designs. The OHAUS Defender scale (Figure 3.13) was used in the measuring of wet samples especially collected samples that weighed over four kilograms. A smaller scale was the primary measurement method for dry samples gathered from the flotation cyclone experiments.

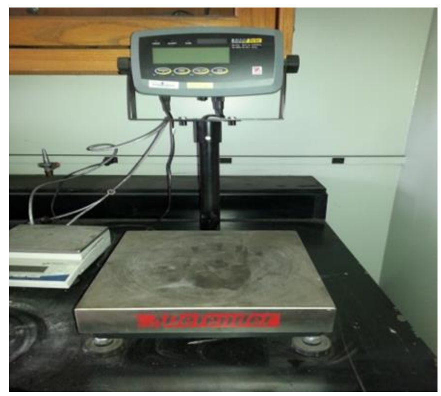


Figure 3.13. OHAUS Defender 5000 Scale.

3.4 Testing Procedures

3.4.1 Water Only

Throughout this research, using water only provided a significant advantage at evaluating the three flotation cyclone designs. The advantages of using water as a testing fluid included visibility and ease of froth generation. Feeding water into the flotation system allowed observation of the fluid mechanics of each design and how those mechanics are affected by varying the operating and design parameters. In addition, the surface tension of water can easily be reduced by the addition of a simple detergent in order to generate bubbles. These simple yet effective characteristics of water allowed the optimization of the three flotation cyclone designs.

To evaluate the designs of the flotation cyclone, each design was constructed with specific pedestal dimensions, discharge characteristics, and operating conditions. The 240 gallon sump was filled with water to the appropriate level to prevent any pump operating problems. Using the pump PLC controller, the speed of the pump motor was adjusted to the desired frequency and consequently the desired water feed rate. Injecting the air required adjusting the regulator and ball valve to the desired pressure and flow rate. At these conditions, the pump was started and the action of the flotation cyclone design was evaluated. If any modifications were needed to achieve the desired output, the operating or design parameters were adjusted. Once satisfied, the simple detergent was added for froth generation.

Testing each cyclone design with water as the media served the purpose of monitoring the effects of varying operating and design conditions. The effects of changing operating conditions were measured by varying one of the parameters and measuring the response of the products. The same procedure was performed for the design parameters. A single dimension such as underflow diameter was varied while the other dimensions were held constant. The masses of the overflow and underflows were collected in a five gallon bucket during a five second interval and weighed. These values were recorded and were further analyzed with Design Expert software to identify the critical operating and design parameters. The Design Expert software uses statistical analysis to help develop prediction models for experiments. In this case, the software was used to determine correlation of input and output parameters. Other input controls and measured results are listed with a brief description (Table 3.6).

Table 3.6. Description of Input and Output Variables in Water Testing Evaluations.

Input Variables	**Symbol**	**Description**
Feed Rate	Q_s	GPM
Underflow Diameter	D_u	GPM
Pedestal Diameter	D_p	inch
Pedestal Height	H_p	% Cyclone Length
Air Flow Rate	Q_g	scfm
Frother Concentration	C_f	ppm
Tangential Flow Rate	Q_t	GPM
Overflow Diameter	D_{of}	inch
Output Variable		
Underflow Flow Rate	Q_u	GPM
Overflow Flow Rate	Q_o	GPM
Pressure Drop	ΔP	psi
Water Height	H_w	inches (air core)

In addition to identifying the appropriate operating and design conditions for the flotation cyclone, an additional goal of testing with water was to verify the design acts in a similar manner to the original Air Sparged Hydrocyclone and traditional flotation techniques. With respect to the Air Sparged Hydrocyclone, each design should mainly be controlled by the feed rate and underflow characteristics. In regards to both the flotation techniques and the ASH unit, the operational performance of the designs should be related to the frother concentration and gas flow rate. Concluding each design mimics the previous flotation work performed by achieving the desired product will potentially prove that the design is an upgrade over existing technologies.

3.4.2 Coal Flotation

Once the major parameters were identified using the water testing, each flotation cyclone design was evaluated at the optimum operating point using the provided coal samples previously described. As stated, using a coal sample as the floatable material decreased the complexity of this procedure. The purpose of testing with the coal will provide yield and recovery rates that will be the ultimate evaluation of the designs.

Prior to performing the evaluations of the designs, the coal slurries were sampled to determine the feed characteristics such as percent solids and ash contents. A small sample was collected using a simple syphon technique and a two liter bucket. The slurry was weighed to obtain an initial slurry mass for future calculations. All the moisture in the sample was removed using the vacuum pump and industrial oven to obtain a dry feed sample and the resulting dry weight was measured and recorded. Using the slurry and dry sample masses, the percent solids by weight was calculated and recorded. The sample was then stored until after all required samples were collected, weighed, and dried for the ash analysis stage.

Once a sample of the collective feed was obtained, the slurry was placed in the 240 gallon sump. The flotation reagents, the collector and frother, were added to sump as well and allowed to mix with slurry in order to properly disperse both chemicals. The amount of reagents added was based on both traditional reagent dosages for flotation and the Air Sparged Hydrocyclone. Using the optimum operating and design conditions determined by the water testing, the feed rate, air flow rate, and design dimensions were set using the various experimental controls, and the evaluation of the design commenced.

As the flotation cyclone came to a steady state, the overflow and underflow streams were sampled during a five second interval using five gallon buckets. Collecting these samples represented a single stage flotation cell which will be the basis of comparison. The product and reject samples were weighed to record the slurry masses for the calculation of percent solids. These samples went through the dewatering and drying process to prepare the samples for the ash analysis step.

In addition to evaluating the flotation cyclone design as a single stage process, the designs were evaluated as a continuous process where the reject streams were recirculated to recover coal not floated during the first stage. This process would act similar in theory to industrial flotation circuits that have multiple stages to recover as much valuable mineral as possible. For this process, PVC pipe was attached to the overflow launder to direct the stream to a separate collection point and the reject streams were allowed to enter the sump for recirculation. To begin the process, the sample coal slurry was placed into the sump and the optimum operating and design parameters were established. The pump was started and the flotation cyclone was allowed to come to a steady state. The first samples of product and reject streams were collected and the process continued until multiple concentrates were collected. These samples were weighed, dewater, and dried in preparation for the ash analysis procedure.

The ash analysis procedures used the dry coal feed, product, and reject samples collected during the design evaluations and found the ash content of each sample using the previously described LECO TGA701 Thermogravimetric Analyzer. A representative sample of each individual sample was placed in the analyzer which raised the temperature of the control environment to remove any organic material. The analyzer reported the mass of inorganic material remaining in the sample which represents the ash content of each sample. These values for each flotation cyclone evaluation are reported in the Results and Discussion section of this project.

4.0 RESULTS AND DISCUSSION

4.1 Primary Design: Fixed Pedestal with Tangential U/F

4.1.1 Water Only

Initial water testing of the first flotation cyclone design intended to show the proof of concept. Figure 4.1 depicts the initial test configuration of the flotation cyclone. As shown, an air core was developed at the top of the pedestal. This air core is the basis of the flotation cyclone to allow the bubble particle attachment to travel toward the developed zero pressure center.

Although the initial design achieved the desired air core result using centrifugal motion, an inherent disadvantage was identified. When frother and air was added to the system, it could be seen that bubbles travelling down the cyclone had the possibility of missing the collection zone and reporting to the underflow. This disadvantage was attributed to the stationary pedestal design of the flotation cyclone. The stationary pedestal design was rigid in the sense that it could not be adapted to changing operating and design conditions. Recognizing this disadvantage led to the secondary design with the adjustable pedestal.

Figure 4.1. Development of Air Core in Initial Flotation Cyclone.

4.2 Secondary Design: Adjustable Pedestal with Tangential and Axial U/F

4.2.1 Water Only

Initial water testing of the secondary flotation cyclone design intended to identify the critical design and operating parameters. The testing did not include the presence of air or frother to assure proper evaluation of the fluid mechanics and system outputs. For the first procedure, varying the design parameters along with cyclone feed rate was performed. The primary goal was to obtain proper design conditions for the secondary design. However, a secondary goal was declared in attempts to verify the previous ASH dimension testing and conclude the design acts in a similar manner. The underflow and overflow flow rates, cyclone pressure drop, and water height were the measured output parameters. The results from testing the input variables are shown in Table 4.1.

Table 4.1. Secondary Flotation Cyclone Operating and Design Evaluation.

Input Variables				**Measured Results**			
D_{uf} (in)	D_p (in)	H_p (%)	Q_s (GPM)	Q_u (GPM)	Q_o (GPM)	ΔP (psi)	H (in)
1.25	-	-	60	28.51	8.03	9.5	1.74
1.25	2	25	60	24.92	13.58	9	1.83
1.25	2	50	60	26.46	12.45	8.5	1.71
1.25	2	75	60	27.70	10.80	8.5	1.74
1.25	2	100	60	28.40	10.42	9	1.74
1.25	3.5	50	60	28.78	9.77	8	1.71
2	3.5	50	60	35.87	1.84	6	1.89
2.5	3.5	50	60	41.70	0.00	5	0.33
3	3.5	50	60	43.17	0.00	4	0.00
1.25	1.25	50	60	25.07	12.62	8	1.80
1.25	1.5	50	60	26.27	12.17	8	1.83
1.25	2	50	60	25.08	12.59	8.5	1.80
1.25	3.5	50	60	28.63	9.07	8	1.71
1.25	2	75	40	23.37	3.62	4	1.64
1.25	2	75	50	26.05	5.59	6	1.67
1.25	2	75	60	27.70	10.80	8.5	1.74
1.25	2	75	70	33.52	11.18	11	1.74

Through correlation analysis, the underflow flow rate, overflow flow rate, and inlet pressure are significantly dependent on the four input variables (Table 4.2). As expected, the underflow opening has a positive effect on the underflow flow rate but negative impacts on the overflow flow rate and inlet pressure. These relationships confirm the underflow's function as a restriction point similar to the ASH unit and typical hydrocyclone operation. The pedestal dimensions have both positive and negative effects on the underflow and overflow streams which indicate the pedestal as another source of restriction and can control the characteristics of the product streams. The system feed rate is typical of hydrocyclone operation as increasing feed rate has a positive effect on all three measured results. These correlations result in the optimum design conditions that will achieve the desired overflow stream where minimal water reported.

Observed during the evaluation was the water level within the flotation cyclone, or better described as the development of the air core. Drawing a conclusion about the air core using the Design Expert software cannot be made due its non-linear behavior. Based on observations and the results provided, an air core would develop twice while adjusting the input parameters. An air core would first develop at lower feed rates or flow restrictions where the centrifugal forces and water reporting to the overflow were minimal. However, as the feed rate or flow restrictions increased, there would be an increase in water reporting to the overflow and the high centrifugal

Table 4.2. The Design Expert Correlation of System Inputs and Outputs.

Correlation of Inputs and Outputs			
		Correlation	
Inputs	Q_u (GPM)	Q_o (GPM)	ΔP (psi)
D_{uf} (in)	0.916	-0.808	-0.662
D_p (in)	0.709	-0.669	-0.443
H_p (%)	0.11	-0.038	0.095
Q_s (GPM)	0.359	0.352	0.663

forces within the flotation cyclone would create another air core. In the case of centrifugal flotation, the air core at lower feed rates and restrictions is desirable over the high centrifugal field air core. These initial efforts of testing the secondary flotation cyclone design parameters provided knowledge about the fluid mechanics and air core development which aids in the design optimization for future coal testing.

Although the conclusions from the initial water evaluations of the flotation cyclone with adjustable pedestal and axial underflow identified the design parameters as critical, not all of the parameters could be considered adjustable during operation. Of the four parameters tested, feed rate and underflow diameter could be considered the two adjustable operating variables. Therefore, further testing to analyze the effects of varying feed rate and underflow diameter was performed. Table 4.3 shows the results from the water evaluation of the feed rate and underflow diameter. No air or frother was added to this evaluation to simplify the analysis.

Inspecting the results from the feed rate and underflow diameter test, similar conclusions can be made about the correlation with product streams, inlet pressure, and water height. Table 4.4 shows the correlation coefficients relating the feed rate or underflow diameter to the measured result. At the various underflow diameters, feed rate positively affects the product streams and the inlet pressure. In comparison, by varying the underflow diameters, the underflow flow rate increases as expected, which decreases the overflow flow rate and inlet pressure due to the lower flow restriction. These conclusions result in the determination that underflow diameter and system feed rate will be the main parameters to change in order to optimize the flotation cyclone for coal evaluation. Once again in this designed experiment, the water level was not a linear trend as two air cores could be developed due to low flow restriction and high centrifugal forces.

Table 4.3. Further Testing of Underflow Diameter and System Feed Rate.

Input Parameters				**Measured Results**			
D_{uf} (in)	Q_s (GPM)	D_p (in)	H_p (%)	Q_u (GPM)	Q_o (GPM)	ΔP (psi)	H (in)
1	30	2	50	15.60	4.13	1.50	1.64
1	40	2	50	18.63	7.17	4.00	1.83
1	50	2	50	20.73	11.75	6.00	1.71
1	60	2	50	24.49	13.37	9.00	1.77
1	70	2	50	28.69	15.28	12.50	1.67
1.25	30	2	50	19.62	0.51	2.00	1.89
1.25	40	2	50	22.87	4.11	3.50	1.86
1.25	50	2	50	23.89	9.84	5.00	1.71
1.25	60	2	50	27.84	11.54	8.00	1.77
1.25	70	2	50	30.92	12.77	11.00	1.71
1.5	30	2	50	20.84	0.00	1.00	0.83
1.5	40	2	50	25.76	1.09	2.00	1.89
1.5	50	2	50	28.54	4.80	5.00	1.83
1.5	60	2	50	31.08	9.22	7.00	1.83
1.5	70	2	50	34.64	10.16	9.50	1.71
2	30	2	50	20.26	0.00	0.00	1.89
2	40	2	50	27.36	0.00	2.00	0.00
2	50	2	50	32.43	0.00	4.00	0.77
2	60	2	50	41.33	0.00	5.00	1.89
2	70	2	50	46.06	1.08	7.50	1.89

Table 4.4. Correlation Coefficients of Underflow Diameter and Feed Rate with System Outputs.

Input	**Correlation**		
	Q_u (GPM)	Q_o (GPM)	ΔP, psi
D_{uf} (in)	0.586	-0.708	-0.319
Q_s (GPM)	0.752	0.608	0.93

Concluding from the preceding tests, the underflow diameter provided a restriction point that aided the development of the required air core for the flotation process and dictates the overflow flow rate. Therefore, the optimum diameter could be found to set the desired air core size and limit the amount of water in the overflow. However, observations during the initial testing established that high inlet feed rates were required to maintain desired air core sizes and overflow quantities. As stated in the design description, two cyclone lengths were constructed, 24 and 15 inches, for the evaluation of the secondary design. The objective of constructing the shorter flotation cyclone was to aid the development of the air core by limiting the required feed rate (Table 4.5).

The analysis of the 15-inch flotation cyclone shows the low pressure requirements and the minimal amount of water reporting to the overflow stream despite the presence of air addition. These results are desirable in a flotation process. The shorter cyclone acts in the same manner where underflow diameter and system feed rate are the crucial parameters and dictate the flotation outputs. However, the advantage of the 15-inch cyclone makes the flotation process more compact thus increasing its potential for capacity per unit volume. Therefore, these

Table 4.5. Evaluation of Pressure Requirements for 15" Flotation Cyclone.

Inputs				**Results**			
D_{uf} (in)	Q_s (GPM)	D_p (in)	Q_g (scfm)	Q_u (GPM)	Q_o (GPM)	ΔP (psi)	H (in)
1	40	2	2	9.07	8.87	6.5	1.58
1.25	40	2	2	10.72	7.65	6	1.71
1.5	40	2	2	13.55	5.84	5.5	1.64
2	40	2	2	20.58	0.85	4.5	1.89
2.5	40	2	2	22.68	0.00	3.5	0.39
1.5	30	2	2	5.79	0.00	1	0.58
1.5	40	2	2	6.53	0.47	4	1.89
1.5	50	2	2	8.12	2.48	6.5	1.89
1.5	60	2	2	8.38	3.83	8.5	1.89

conclusions led to the use of the 15-inch cylinder length in the coal flotation analysis of the secondary design.

In addition, the operational parameters of the secondary flotation cyclone were evaluated using water as the testing media. Table 4.6 represents the testing of the operational parameters feed rate (gpm), frother concentration (ppm), and gas flow rate (scfm) and their associated effects on tangential underflow, axial underflow, and overflow. The purpose of testing the operational parameters was to identify the conditions that controlled the overflow product and achieve a system that mimics a traditional operating flotation process. The goal was to develop a central froth column and limit the amount of water reporting to the overflow, which in flotation

Table 4.6. Operating Parameter Testing of Secondary Flotation Cyclone.

Operating Variables			**Measured Results**						
Slurry Flow Rate (GPM)	Gas Flow Rate (scfm)	Frother (ppm)	Tangent U/F		Axial U/F		Overflow		Q_s
			GPM	% Feed	GPM	% Feed	GPM	% Feed	GPM
40	1	5	17.52	0.60	9.39	0.32	2.40	0.08	29.31
40	1	10	18.51	0.63	8.34	0.28	2.70	0.09	29.56
40	1	15	15.45	0.54	8.48	0.29	4.91	0.17	28.84
40	1.5	5	19.86	0.68	8.30	0.28	1.14	0.04	29.30
40	1.5	10	17.70	0.62	8.06	0.28	2.57	0.09	28.33
40	1.5	15	15.46	0.56	8.13	0.29	4.16	0.15	27.74
50	1	5	22.95	0.66	9.64	0.28	2.33	0.07	34.92
50	1	10	24.65	0.67	8.93	0.24	3.18	0.09	36.76
50	1	15	21.47	0.61	8.37	0.24	5.58	0.16	35.42
50	1.5	5	23.16	0.68	8.52	0.25	2.48	0.07	34.16
50	1.5	10	21.86	0.64	8.99	0.26	3.32	0.10	34.17
50	1.5	15	20.04	0.57	9.11	0.26	5.95	0.17	35.11
60	1	5	30.60	0.73	9.05	0.21	2.44	0.06	42.10
60	1	10	27.68	0.65	9.96	0.23	5.12	0.12	42.76
60	1	15	22.69	0.54	9.86	0.23	9.84	0.23	42.39
60	1.5	5	30.86	0.74	8.01	0.19	2.56	0.06	41.42
60	1.5	10	25.10	0.61	11.07	0.27	5.10	0.12	41.27
60	1.5	15	23.22	0.56	11.04	0.27	7.05	0.17	41.31

can increase the product ash content through entrainment. Achieving these conditions allow optimal conditions for flotation in secondary flotation cyclone design.

Analysis of the operating parameters will indicate the critical parameters that can be changed to achieve the desired product in addition to verifying the system acts like a flotation process. The correlation coefficients signify frother concentration has the largest positive impact on the product overflow (Table 4.7). The feed input flow rate positively impacts all the product streams with the tangential underflow subjected to the largest impact. The increase in gas flow rate has a small negative impact on all three product streams, but significant testing could not be achieved as excessive air quantities "burped" the flotation cyclone. In addition, it should be noted the presence of the Cavitation Tube reduces the inlet feed flow rate due to the decrease in pressure and the generation of bubbles. This phenomenon does not affect the slurry rate measured by the flow meter, but instead may reduce the centrifugal force developed within the cyclone and inhibit the development of the air core.

4.2.2 Coal Flotation

The evaluation of the secondary design using coal as the flotation feed dictated whether

Table 4.7. Correlation of Operating Inputs and System Outputs.

Input	Output	Correlation
Q_s (GPM)	T U/F (GPM)	0.858
	A U/F (GPM)	0.616
	O/F (GPM)	0.464
Q_g (scfm)	T U/F (GPM)	-0.053
	A U/F (GPM)	-0.048
	O/F (GPM)	-0.107
C_f (ppm)	T U/F (GPM)	-0.412
	A U/F (GPM)	0.163
	O/F (GPM)	0.791

the design could be an effective flotation technique. After optimizing the design and operating parameters of the flotation cyclone, a coal flotation analysis on the sample was performed and analyzed. Table 4.8 shows the input operating and design parameters with the feed, product, and tailing stream characteristics.

The initial results of the secondary design's capability of achieving a quality flotation product provide significant information. The design achieved a product with an ash of 20.5 % from a 47.33% ash feed. However, the design did not significantly affect the ash content of the tailings stream as the content was only increased to 47.71%. However, this test was performed incorrectly as the air flow rate was not injected with any pressure. The air flow rate was established before the test began and equilibrium was created within the pipe network where

Table 4.8. Initial Flotation Analysis of Secondary Design.

Cyclone Design Parameters		
Flotation Height	5.25	in
Pedestal Diameter	2	in
O/F Diameter	1	in
U/F Diameter	1	in
Tangential U/F	25	GPM
Operating Parameters		
Feed Rate	41	GPM
Air Flow	7	SCFM
Frother Concentration	7	ppm
Collector Concentration	0.065	lb/ton
Stream Characteristics		
Product	20.46	% Ash
Axial U/F	51.47	% Ash
Tangential U/F	47.71	% Ash
Feed	47.33	% Ash
Flotation Analysis		
Mass Yield	7.35	%
Product Ash Content	20.46	%
Combustible Rec.	11.09	%

minimal air, not 7 SCFM, was injected. Therefore, another test was performed.

The following flotation test was performed with the proper air addition corrections. Table 4.9 shows the design and operating parameters, stream characteristics, and flotation analysis of the experiment. The analysis did not show any improved qualities in the product after correcting the air flow rate and increasing the frother and collection addition. These results are particularly due to a majority of the feed reporting to the underflow streams in a short time period. Therefore, large fractions of the initial coal weight are not recovered in that short time period and leads to a tailings composition similar to the feed. Therefore, it was concluded to reduce the volumetric flow rates of the underflow streams, specifically the tangential underflow, in order to either decrease the required feed rate or increase the overflow flow rate.

Table 4.9. Secondary Flotation Cyclone Performance with Corrected Air Injection.

Cyclone Design Parameters		
Flotation Height	12.5	in
Pedestal Diameter	2	in
O/F Diameter	1.25	in
U/F Diameter	1	in
Tangential U/F	26.4	GPM
Operating Parameters		
Feed Rate	50	GPM
Air Flow	1	SCFM
Air Pressure	25	psi
Frother Concentration	10	ppm
Collector Concentration	0.2	lb/ton
Stream Characteristics		
Product	23.81	% Ash
Axial U/F	52.74	% Ash
Tangential U/F	47.52	% Ash
Feed	47.15	% Ash
Flotation Analysis		
Yield	5.85	%
Ash	23.81	%
Combustible Recovery	8.43	%

In efforts to reduce underflow volumetric flow rates and increase the overflow, new optimization procedures were taken. The tangential underflow was reduced to zero and the frother concentration was increased to 30 ppm. The design and operating parameters were optimized to produce a desirable flotation product using water and the parameters were evaluated. The results from this evaluation are summarized in Table 4.10.

The results from the third test are promising as the product quality was greatly improved over the previous tests by rendering the tangential underflow non-existent and increasing the frother and collector dosages. In order to indicate performance, this point was plotted on the ash recovery curve of the coal release analysis and compared with the flotation column at the plant (Figure 4.2). The concentrate ash content was reduced to 4.41% which signifies a high quality

Table 4.10. Flotation Results with Axial Underflow.

Cyclone Design Parameters		
Flotation Height	11.25	in
Pedestal Diameter	3.5	in
O/F Diameter	1.25	in
U/F Diameter	2.5	in
Operating Parameters		
Feed Rate	50	GPM
Air Flow	0.5	SCFM
Air Pressure	60	psi
Frother Concentration	30	ppm
Collector Concentration	0.25	lb/ton
Stream Characteristics		
Product	4.41	% Ash
Axial U/F	61.29	% Ash
Feed	47.16	% Ash
Flotation Analysis		
Mass Yield	24.84	%
Product Ash Content	4.41	%
Combustible Rec.	44.94	%

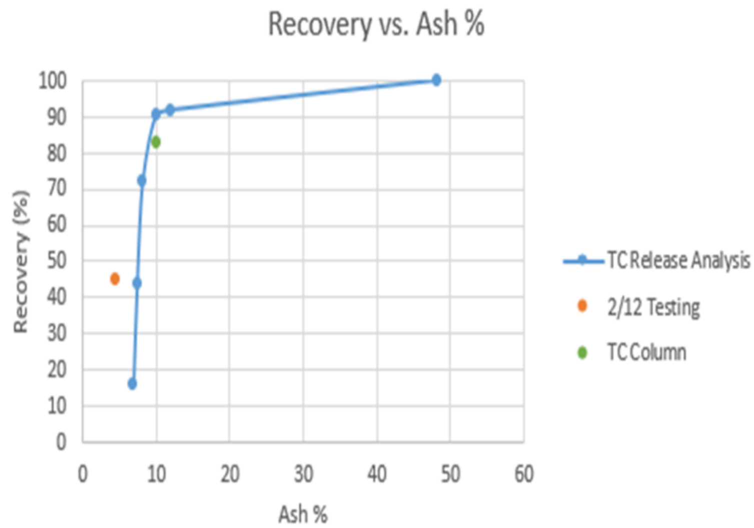


Figure 4.2. Comparison of Flotation Cyclone and Release Analysis.

product, and the design achieved a yield of nearly 25% and recovered nearly 45% of the coal. By implementing a one tailing discharge system, the axial tailings were downgraded to a 61% ash content which show significant improvements over the previous system where the feed and tailings ash were essentially the same. However, a frother concentration of 30 ppm was need to achieve the product which follows Miller's ASH testing but greatly exceeds the frother dosage of the plant column. Although these results show the systems capability to produce a high quality concentrate, the yield and recovery rates for a single cycle are not comparable to traditional flotation techniques and typical frother concentrations.

In efforts to increase the yield and combustible recovery of the secondary flotation cyclone design, a continuous flotation process was utilized. The concentrate was collected at two minute intervals and the tailings returned to the flotation cyclone for further separation. Prior to sample number six and eight, additional frother was added to return the frother concentration to 30 ppm and increase recovery of coal. The results of this test are in Table 4.11 and the comparison to the release analysis in Figure 4.3. The analysis shows the requirement to collect

Table 4.11. Results of Continuous Flotation Process.

Sample	Individual (%)		Cumulative (%)		
	Ash	Yield	Ash	Yield	Recovery
C1	6.32	11.38	6.32	11.38	21.87
C2	5.84	2.50	6.24	13.59	26.14
C3	8.50	1.68	6.46	15.04	28.86
C4	8.93	3.55	6.87	18.06	34.50
C5	7.38	2.03	6.91	19.72	37.65
C6*	7.50	4.54	7.01	23.36	44.57
C7	14.92	14.44	9.55	34.43	63.89
C8*	28.82	14.55	13.73	43.97	77.82
Tailings	80.71	56.03	51.26	100.00	100.00

concentrate while reprocessing the tailings to achieve similar product ash contents, yield and recovery rates to meet the release analysis results. In comparison with the plant flotation column, the continuous flotation cyclone achieved a product of 13.7% ash at a 77% recovery rate while the flotation column achieved 9.9% ash product at 82% recovery. The flotation column at the plant is superior to the flotation cyclone design after the continuous processing of the tailings stream and subsequent additions of frother. In essence, the secondary flotation cyclone design

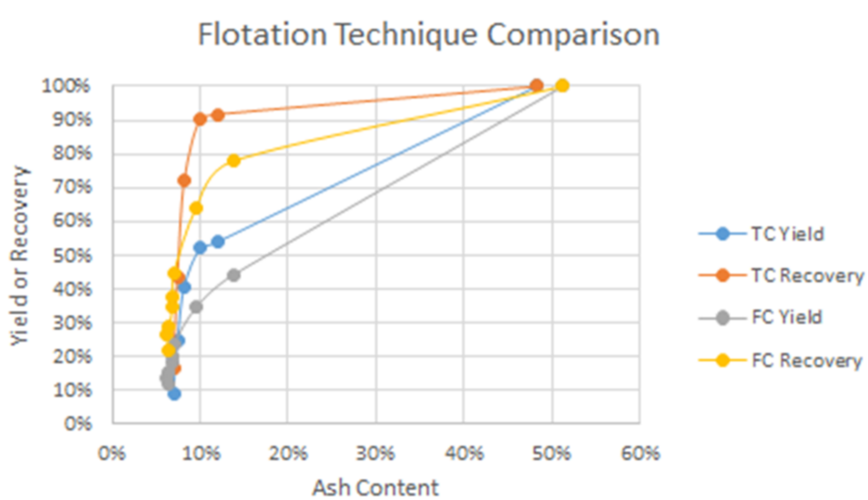


Figure 4.3. Comparison of Continuous Flotation and Release Analysis.

acts in a similar behavior of flotation columns and conventional cells as large retention times are required. However, the unit residence time for the second flotation design is 0.66 seconds.

4.3 Tertiary Design: Tangential Aeration Chamber with Axial U/F

4.3.1 Water Only

Analyzing the third flotation cyclone design through the use of water and frother provided the conclusion that the design was incapable of producing a froth product. Major problems arose from the attempts to optimize the design and operating parameters to mimic flotation behaviors. The aeration cylinder's inability to produce bubbles caused the added air of the system to short circuit to the overflow and water filled up the aeration chamber. In order to solve the problem of water filling the aeration chamber, changes were made to increase the volume of the air without drastically increasing the pressure. However, increasing the volume resulted in the increase of air short circuiting to the overflow. To overcome the shortcoming of the bubble generation problem, frother concentration was added to further reduce the surface tension of the water and the feed rate was increased in efforts to increase the shear rate of the swirling water against the incoming air. Increasing the frother had no effects and increasing the feed rate resulted in the aeration chamber filling up faster. These results show this designs inability to produce the desired froth column mainly due to the aeration cylinder not being an adequate sparging system.

Due to the conclusions of the initial design, the variation to include a static mixer as an external sparging system was made. Through optimization of the design and operating parameters, the third flotation cyclone design was able to achieve a desired flotation behavior. Upon observation with water, bubbles entered the outer aeration chamber, entered the inner cylinder, and agglomerated in the center of the cyclone. Subsequently when air and frother was

Table 4.12. Design and Operating Parameters of Third Flotation Cyclone Design.

Design Parameters		
U/F Diameter	2	in
O/F Diameter	1.25	in
Pedestal Diameter	1.5	in
Flotation Height	13	in
Operating Parameters		
Feed Rate	19	GPM
Air Flow Rate	0.5	SCFM
Frother Concentration	60	ppm
Air Pressure	20	psi
U/F Pump Speed	87	%

added, the concentration of bubbles increased and the air core diameter increased. These parameters are listed in Table 4.12. These parameters achieved a froth column in which an adequate overflow product with minimal water was generated. The high frother concentration increased the diameter of the froth column inside the two inch cylinder and the low air flow rate provided the proper air to water mixture passing through the static mixer to prevent burping and large bubbles.

4.3.2 Coal Flotation

Using the design and operating parameters determined in the water testing of the third flotation cyclone design, a continuous flotation process using the metallurgical coal sample was performed. A feed and concentrate sample was collected for five second periods once the process reached steady state and the tailings were reprocessed. Concentrate samples were collected at five minute intervals and the results are shown in Table 4.13.

The third design resulted in a higher recovery and yield than the previous two designs while producing an acceptable ash content value. The composite ash value is 11.32 % while the rejected tails is approximately 37.8% ash. Using the typical flotation performance factors, the

Table 4.13. Continuous Process of Third Flotation Cyclone.

	Individual (%)		Cumulative (%)			
	Mass	Ash	Mass	Ash	Yield	Recovery
C1	10.57	9.86	10.57	9.86	11.6	14.1
C2	6.2	14.13	16.77	11.44	18.4	21.9
C3	9.03	10.65	25.8	11.16	28.4	33.8
C4	7.9	11.59	33.7	11.26	37.1	44.2
C5	8.25	11.54	41.95	11.32	46.1	54.9
Tails	48.99	37.79	90.94	25.58	100.0	99.9
Feed	128.83	25.54				

cumulative yield and recovery of the flotation cyclone after a 20 minute operation are 46.1% and 54.9% respectively. At a feed rate of 19 gallons per minute and a 60 ppm frother concentration, the flotation cyclone design acts more like a traditional flotation cell rather than a fast flotation process. The required retention time and high frother concentration to achieve these results defeats the objective of this research project. The third flotation design requires such high retention times because the mass flow rate of hydrophobic particles through the overflow is such a small rate on the individual level. In addition, the high frother concentration is necessary to develop a substantial froth column in the center of the cyclone and provide sufficient bubbles for particle attachment.

An adequate comparison between the third flotation cyclone design and the industrial plant data is difficult to conclude as the feed for the third design contained far less ash by weight than the original industrial coal sample. The original samples from the plant were approximately 47% ash while the flotation cyclone feed ash content was approximately 25%. The cause of this discrepancy is uncertain. One possible solution is the through flotation cyclone testing, the sample could not be transferred between the sump and sample containers. This inability to recover the complete sample from the sump could credit the loss of the large majority of the ash fractions. Although the comparison cannot be made, the performance of the third flotation cyclone shows its ability to achieve a decent product but only after a long processing period and high frother dosages.

5.0 CONCLUSIONS AND RECOMMENDATIONS

In the evaluation of the three flotation cyclone designs, the ability to develop a flotation device was a challenging task. The initial flotation cyclone design had an inherent disadvantage of the stationary pedestal. This design flaw created the possibility of bubble particle agglomerates missing the collection zone and reporting to the underflow. Despite this design flaw, water testing validated the theory of centrifugal flotation through the creation of an air core with an external sparging system.

The second flotation design incorporated the design on an adjustable pedestal and improved the design by adding an axial underflow to control the fluid level within the cyclone. Evaluation of the design with water eliminated the need for a tangential underflow but proved the axial underflow did not add a source of fluid control. The coal flotation proved the design achieved a very high quality product but at a low recovery rate. By evaluating the design as a continuous process, the system was able to achieve combustible recoveries nearing 73% at ash contents of 13.7%. However, the design required 16 minutes of reprocessing tailings to achieve these values thus making the design behave more like a typical flotation technology as opposed to a fast flotation technology. In addition to the long retention time, increasing the aeration rate was not a feasible option to increase flotation which adds a limitation to the design.

The third flotation design strayed from the original two designs by incorporating a tangential aeration chamber and an external static mixer as the bubble generator. The system was able to achieve an 11.32% ash concentrate at a recovery of 55% in a 20 minute retention time. Further analysis of the third design as a continuous process didn't yield higher recoveries. Although the recovery rate and ash values portray effectiveness, the majority of the feed was floatable material and this may have led to the increased recovery rate. In addition, the third design required higher frother concentration not typically found in flotation systems and potentially could have contributed to the better performance.

The development of a centrifugal flotation technology proved separation results but not to the standard of traditional flotation techniques. The major limiting factors of the three designs was controlling the level within the cyclone, providing adequate aeration with an external sparging system, and increasing mass flow rate of hydrophobic particles to the overflow in a short time frame. In regards to these three designs, further work could possibly be performed with flotation feed containing a low concentration of hydrophobic particles with the second flotation design. The design proved the ability to effectively recover high quality particles at low yield rates, but no tests were performed to study the size of the recovered particles. In addition, the Cavitation Tube was designed for flow rates that far exceeded the needs of the second design. Possible improvements to recovery could be made by utilizing a Cavitation Tube able to handle those flow rates. In regards to improving the designs and the movement towards fast flotation technologies, the initiatives should be towards developing an appropriate sparging method that can be implemented, withstand plugging, and achieve high recovery rates.

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APPENDIX A

Figure 1.1 Comparison of Conventional and Column Cell
Kawatra, S. K. (2011). Fundamental Principles of Froth Flotation. In P. Darling, *SME Mining Engineering Handbook* (pp. 1517-1531). Society for Mining, Metallurgy, and Exploration. Used with permission from Society for Mining, Metallurgy, and Exploration. Letter attached.

Figure 1.2 Thermodynamic Description of Particle Bubble Attachment
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Used under fair use. Form attached.

Figure 1.3 Flotation Recovery versus Particle Size
Gaudin, A., Grob, J., & Henderson, H. (1931). Effect of Particle Size in Flotation. *AIME Volume: No. 414*. Used with permission from American Institute of Mining, Metallurgical, and Petroleum Engineers, Inc. Letter attached.

Figure 1.4 Analysis of Adhesion Probability in Flotation at Varying Induction Times
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Figure 1.5 WEMCO 1+1 Flotation Cell
WEMCO 1+1 Flotation Cell. (2010). Retrieved from FL Smidth: http://www.flsmidth.com/~/media/PDF%20Files/Liquid-Solid%20Separation/Flotation/Wemco11brochure.ashx. Used with permission from FL Smidth. Letter attached.

Figure 1.6 Porous Venturi Sparging System and CPT Cavitation Tube System
Yoon, R., Luttrell, G., & Adel, G. (1990). *Advanced Systems for Producing Superclean Coal.* Blacksburg: U.S. Department of Energy. Fair use as government publication.
Column Flotation Systems Cavitation Tube. (2009). Retrieved from Canadian Process Technologies Inc.: http://efd.eriez.com/Products/Index/Cavitationtube. Used with permission from Jaisen Kohmuench of Eriez Flotation Division. Letter attached.

Figure 2.1 Trend of Flotation Size in Past Century
Noble, C. (2013). *Analytical and Numerical Techiques for the Optimal Design of Mineral Separation Circuits.* Blacksburg: Virginia Tech. Used with permission from author. Letter attached.

Figure 2.2 Effects of Retention Time on Recovery (Luttrell, 1999)
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Figure 2.3 The Effects of Varying Particle Diameter on Flotation Rates (Fuerstenau, 1980)
Fuerstenau, D. (1980). Fine Particle Flotation. In P. Somasundara, *Fine Particles Processing* (p. 671). SME-AIME. Used with permission from Society of Mining, Metallurgy, and Exploration. Letter attached.

Figure 2.4 Krebs Hydrocyclone Cutaway (FL Smidth)
Flotation Technology. (2010). Retrieved from FL Smidth: http://www.flsmidth.com. Used with permission from FL Smidth. Letter attached.

Figure 2.6 Air Sparged Hydrocyclone Assembly (Ye, 1988)
Ye, Y., Gopalakrishnan, S., Pacquet, E., & Miller, J. (1988). Development of the Air Sparged Hydrocyclone - A Swirl-Flow Flotation Column. *Column Flotation '88 - Proceedings of an International Symposium* (p. 9). Denver: SME. Used with permission from Society of Mining, Metallurgy and Exploration. Letter attached.

Figure 2.7 Varying Phases of Air Sparged Hydrocyclone (Ye, 1988)
Ye, Y., Gopalakrishnan, S., Pacquet, E., & Miller, J. (1988). Development of the Air Sparged Hydrocyclone - A Swirl-Flow Flotation Column. *Column Flotation '88 - Proceedings of an International Symposium* (p. 9). Denver: SME. Used with permission from Society of Mining, Metallurgy and Exploration. Letter attached.

Figure 2.8 Pictorial Description of Imhoflot G Cell (Battersby, 2003)
Battersby, M., Brown, J., & Imhof, R. (2003). *The Imhoflot G-Cell - An Advanced Pneumatic Flotation Technology for the Recovery of Coal Slurry From Impoundments.* Cincinatti: Society for Mining, Metallurgy, and Exploration. Used with permission from Society of Mining, Metallurgy, and Exploration. Letter attached.

Figure 2.9 Turboflotation System (Ofori, 2000)
Ofori, P., Firth, B., & Howes, T. (2000). Assessment of the Controlling Factors in TurboFlotation by Statistical Analysis. *Coal Preparation: Volume 21*, 355-382. Used with permission from Taylor and Francis. Letter attached.

Figure 2.10 Pilot Plant Turboflotation Yield and Ash Content (Ofori, 2000)
Ofori, P., Firth, B., & Howes, T. (2000). Assessment of the Controlling Factors in TurboFlotation by Statistical Analysis. *Coal Preparation: Volume 21*, 355-382. Used with permission from Taylor and Francis. Letter attached.

Figure 3.1 Initial Flotation Cyclone
Yan, E. (2013). Initial Flotation Cyclone. Eriez Flotation Division. Used with permission from creator. Letter attached.

Table 2.2 Ash Removal Comparison between Separation Technologies (Miller, Camp, 1982)
Miller, J., & Van Camp, M. (1982). Fine Coal Flotation in Centrifugal Field With an Air Sparged Hydrocyclone. *SME Mining Engineering*, 1575-1580. Used with permission from Society of Mining, Metallurgy, and Exploration. Letter attached.

Table 2.3 Design and Operating Variables for Gold Flotation (Miller, 1985)
Miller, J., Misra, M., & Gopalakrishnan, S. (1985). Fine Gold Flotation From Colorado River Sand with the Air Sparged Hydrocyclone. *SME-AIME*. Albuquerque: Society of Mining Engineers of AIME. Used with permission from Society of Mining, Metallurgy, and Exploration. Letter attached.

Table 2.4 Comparison of Separation Results for Fine Gold Ore (Miller, 1985)
Miller, J., Misra, M., & Gopalakrishnan, S. (1985). Fine Gold Flotation From Colorado River Sand with the Air Sparged Hydrocyclone. *SME-AIME*. Albuquerque: Society of Mining Engineers of AIME. Used with permission from Society of Mining, Metallurgy, and Exploration. Letter attached.