# DEVELOPMENT AND EVALUATION OF A DENSE MEDIA CYCLONE FOR THE SOUTHERN AFRICAN MINERAL AND COAL INDUSTRIES

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# ABSTRACT

Dense media separation (DMS) plays an integral part in coal processing as well as in the upgrade of low grade ore bodies prior to further processing like flotation and leaching. Various separating vessels are used in DMS; of which the dense media (DM) cyclone is the most common. However, the limited materials of construction available for DM cyclones leave the industry with little choice. In addition, the DM cyclones available moved away from the basic principles of classification hydrocyclones which is to reduce the turbulence in the feed chamber in order to achieve improved efficiency. The ceramic tiled DM cyclones available in the market have ceramic tiles installed perpendicular to each other in the feed chamber, which increase the turbulence in the feed chamber at the expense of longer equipment life. For that reason, a research project was initiated to establish the current installed base of DM cyclones in Southern Africa and to develop and evaluate a DM cyclone with a laminar spiral feed chamber design to reduce the turbulence when feeding the cyclone, whilst achieving the same or superior wear properties.

Because Cavex is well proven in hard rock mining and coal classification, it was used as basis for the development of a DM cyclone. Individual moulds were developed and produced in order to fabricate a Cavex DM cyclone with the exact laminar spiral feed chamber that exists when moulded out of rubber. Afterwards, the Cavex ceramic tiled (CVXT) cyclone was manufactured, installed and commissioned. Using the basic operating principles of hydrocyclones, the predicted results were achieved. The 650CVXT results confirmed that a DM cyclone and a classification cyclone are inherently the same piece of equipment; it is the selection criteria and material of construction that differ. There is a definite benefit in focussing on metallurgical efficiency in conjunction with wear properties, rather than equipment life (wear life) as the only parameter during equipment fabrication. A simple way to achieve improved performance is to reduce the turbulence in the feed chamber.

Further work must include the evaluation of the Cavex CVXT DM cyclone in near gravity coal circuits and to investigate whether the same lower operating pressures (6D compared to conventional 9D) will result in improved coal yield, as was illustrated successfully in the separation of fluorspar by DMS, whilst achieving the desired ash contents. The work must include detailed studies on the effect of feed pressure vs. yield/recovery, when the turbulence in the feed chamber is reduced. Overall energy savings should be quantified by reducing the cyclone feed pressure.

## ACKNOWLEDGEMENTS

Personal thanks is expressed to Weir Minerals Africa for the opportunity to head up the design of the Cavex DM cyclone, resources and facilities, Nkosinathi Ntuli for his metallurgical input and assistance in commissioning and Verna Kruse for assistance in editing this report.

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# Nomenclature

| | |
|---|---|
| c (d) | - corrected partition value |
| $CaF_2$ | - Calcium Fluoride |
| CFD | - Computational fluid dynamics |
| CVXT | - Cavex tiled range of cyclones |
| D | - Diameter of cyclone |
| d | - particle size |
| $d_{50}$ | - particle size with 50% probability to report to the overflow or underflow |
| $d_{50c}$ | - corrected cut size |
| $D_c$ | - Inside diameter of a hydrocyclone measured at the bottom of the vortex finder |
| $D_i$ | - Inside diameter of a hydrocyclone inlet or $(4A_i/\pi)^{½}$ for non circular inlets |
| DM Cyclone | - Dense Media cyclone |
| DMS | - Dense Medium Separation |
| DSM | - Dutch State Mines |
| $D_o$ | - Inside diameter of the overflow or vortex finder of a hydrocyclone |
| $D_u$ | - Inside diameter of the underflow, or apex, orifice of a hydrocyclone |
| EPM | - Ecart Probable (moyen) |
| F | - Mass flow rate of the solids in feed |
| FeSi | - Ferro silicon |

| | |
|---|---|
| f(d) | - Weight fraction of particle size d in the feed |
| $F_{50}$ | - The weight median size of the feed solids in microns (50% passing size) |
| Fc | - Centrifugal forces |
| Fg | - Gravitational force |
| g | - Acceleration of gravity |
| h | - Free vortex height of a cyclone which is defined as the distance from the bottom of the vortex finder to the top of the underflow orifice |
| H | - Pressure drop across a hydrocyclone expressed in head of feed slurry |
| HLS | - Heavy liquid separation |
| ISO | - International Organisation for Standardisation |
| kPa | - kilo Pascal |
| LES | - Large Eddy Simulations |
| m | - Parameter in the Rosin Rammler type expression for a classification curve which serves as a measure of the sharpness of separation |
| mm | - millimetre |
| $m^3$ | - cubic meter |
| Mf | - mass of fluid displaced by particle |
| Mp | - mass of particle |
| OHSAS | - Occupational Health and Safety Assessment Specification |
| o/f | - overflow |
| P | - Pressure drop across ad hydrocyclone |

| | |
|---|---|
| p(d) | - partition number |
| Q | - Volumetric flowrate of hydrocyclone feed slurry |
| r | - Radius of cyclone |
| ROM | - Run of mine |
| RSM | - Renolds Stress Models |
| R&D | - Research and Development |
| r(d) | - fraction of material of size d bypassing classification |
| r(d) | - fraction of material of size d bypassing classification (Tromp curve) |
| $R_f$ | - Water recovery from the feed to the underflow |
| $R_v$ | - Recovery of feed volume to the underflow product |
| S | - Ratio of volumetric underflow rate to volumetric overflow rate |
| S.G | - Specific gravity |
| TBE | - Tetrabromoethane |
| t | - Metric tonnes |
| U | - Mass flow rate of the solids in underflow |
| u/f | - underflow |
| UG2 | - Upper Group 2 geological reef |
| u(d) | -Weight fraction of particle size d in the underflow |
| V | - Tangential velocity |
| v | - Velocity |
| $x_i$ | - particle size/corrected cut size |

| | |
|---|---|
| y’ | - mass fraction of a given size and density which will be directed to the underflow as a result of the classifying action (i.e. corrected for underflow liquid) |
| p(d) | - Partition number |
| $\rho$ | - Density of liquid |
| $\rho_p$ | - Pulp density |
| $\rho_s$ | - Solid density |
| R(f) | - Feed to the underflow |
| % | - Percentage |
| $\Phi$ | - Percent (%) by volume |

# 1 INTRODUCTION

## *1.1 BACKGROUND*

Dense media separation (DMS) is applied to the pre-concentration of minerals i.e. the rejection of gangue prior to grinding for final liberation (Wills 1997). In addition, it is also used in the upgrade of coal to produce commercially-graded end product(s) as well as the beneficiation of diamonds, iron ore, chrome, andalusite, fluorspar, base minerals, manganese, phosphate and tin, amongst others. The dense media (DM) cyclone has been installed in over one quarter of the coal preparation plants worldwide (Reeves, 2002). de Korte (2000) reported that about 93% of the 58 coal preparation plants in South Africa employed DM cyclones.

DMS processes offer some advantages over the other beneficiation processes (England et al, 2002):

- DMS have the ability to make sharp separations at any required relative density within the range normally required for coal preparation (and other minerals). A high degree of efficiency can be achieved even in the presence of high percentages of near density material (minerals/coal within ±0.1 relative density units of the cut point).
- The separating density can be closely controlled i.e. within a relative density of 0,005 and can be maintained under normal conditions for indefinite periods.
- The relative density and thereby the cut point, can be changed at will and fairly quickly to meet varying requirements.
- A wide range of sizes (+0.1mm – 150mm) can be handled (each size range in different units).
- Fluctuations in quality and quantity of feed can be handled with ease within the limits of the product-handling capacity of units.

Furthermore, pre-concentration by DMS, or a combination of DMS and optical sorting is an area of research interest in the platinum industry, especially the UG2 reef (Bryson, 2004b). Mining dilution in the UG2 reef can be eliminated with DMS where the waste rock is rejected by DMS prior to milling as well as increasing the feed grade to the mill (Bryson, 2004a). Bryson (2004a) further notes that DMS plants do exist in UG2 operations and it is predicted that some of the new operations will include this technology in future. The elimination of waste rock by DMS is applicable to most mining operations and not only to the UG2 Platinum reef.

Apart from the above, DMS of fine coal (-0.5mm) in South Africa is also being revisited by CoalTech 2020 (de Korte, 2002). DMS on fine coal was installed at Industrial Scale in South Africa in 1981 (van der Walt et al, 1981). de Korte (2002) further notes that, even though the fine DMS plant at Greenside Colliery has been in operation since the mid 1980's with superior results to any other beneficiation route in this size range, it never has been accepted as the preferred route for the fines (-0.5mm) fraction against spirals. However, this is likely to change as a large proportion of future coal will be mined from the number 4 seam coal which spirals cannot beneficiate to the required quality i.e. <7% ash. Leeuwpan colliery commissioned a fine coal (+100µm -1mm) DMS plant during 2008 (Lundt, 2009) and this author is of the opinion that this trend will continue.

Various separating vessels are used during the DMS process. These vessels include gravitational (static baths) and centrifugal (dynamic) vessels (Wills, 1997). Gravitational units comprise some form of vessel into which the feed and medium are introduced and the floats are removed by paddles or merely by overflow. The sinks aim to discharge without removing sufficient medium to cause disturbing downward currents in the vessel (Wills, 1997). Gravitational vessels include Wembco cone separator, Drum separators, Drewboy bath and Norwalt washer.

Centrifugal separators in the form of the cyclone DM separator are widely used in the treatment of ores and coal. DM cyclones proved a high centrifugal force and a low viscosity in the medium enabling much finer separations to be achieved than in gravitational separators (Wills, 1997). Feed to DM cyclones is typically deslimed at 0.5mm to avoid contamination of the medium with slimes and to minimize medium consumption but, as mentioned, this trend is likely to change.

To use the following coal example: According to the Department of Minerals and Energy (DME, 2009), South Africa sold 247,666,356 tonnes of coal in 2007. Taking into consideration that saleable coal accounts for approximately 50% of run of mine (ROM) coal and the fact that ±90% of coal mines in South Africa use DM cyclones to beneficiate coal, the DM cyclone plays an integral part in the South African coal industry, not to mention the diamond and chrome industries.

## *1.2 PROBLEM STATEMENT AND RESEARCH MOTIVATION*

Currently, the following DM cyclones are available to the South African mining industry:

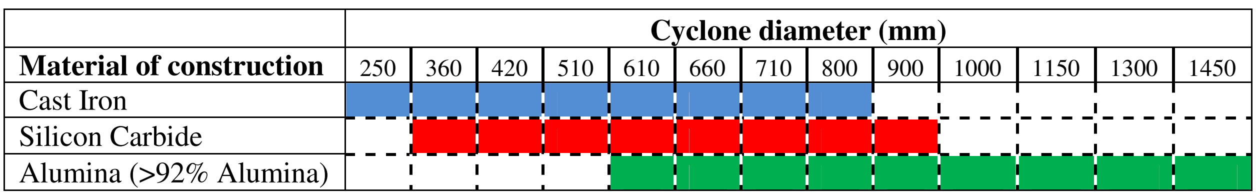



**Figure 1.2.1. Dense medium cyclones**

The limitations on material of construction for DM cyclones leave the coal and mineral industries with little choice. The research undertaken during this study considered whether the available cyclones are sufficient for the coal and mineral industries, what material of construction and cyclone size is most widely used in industry and what alternatives, if any, can be provided to industry. The ultimate aim of the research was be to introduce a product that reduces operating costs whilst responsibly caring for the environment. Implementing the correct, most efficient DM cyclone, complying with local legislation and conforming to environmental regulations and customer requirements are essential for the sustainability of South Africa's coal and mineral industries and the economy as a whole.

The project included a survey of the existing DM cyclone Market in Southern Africa to establish the current installed base for DM cyclones in the coal and mineral industries. From the survey, a DM cyclone size and construction material entry level was selected. This was followed by the development of design drawings and moulds. Local fabrication facilities were set up in South Africa and the prototype Cavex laminar spiral feed chamber DM cyclone was developed after a trial site was secured. The objective was to evaluate a DM cyclone with a laminar spiral feed inlet with the ultimate aim to provide industry with an alternative DM cyclone that will reduce operating costs whilst providing a superior final product for the mineral industry.

# 2 LITERATURE REVIEW

The principle and basic design of conventional hydrocyclones was first patented in 1891 but only found significant application in industry after the Second World War (Svarovsky, 1984). Hydrocyclones, commonly referred to as cyclones (Arterburn, 1982), have become the accepted standard method of classifying slurries in the mineral industry since the mid-1950s (Plitt, 1976). Classification hydrocyclones separate solid particles on the basis of size. A classification cyclone uses water as carrier and separating medium and is different from DM cyclones which use magnetite or ferrosilicon mixed with water as carrier and separating medium. The classification hydrocyclone is presented in figure 2.1 and 2.2 and the DM cyclone is presented in figure 2.3.

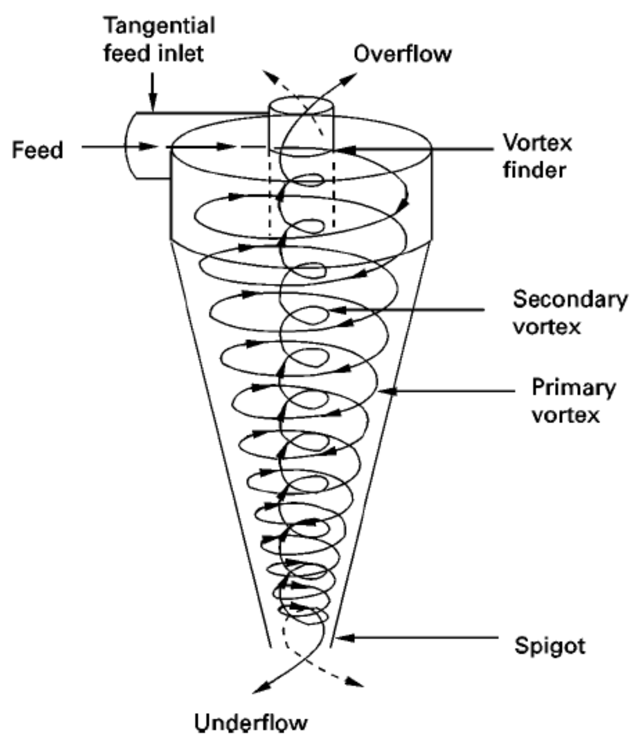



**Figure 2.1. Principle features of a hydrocyclone and flow patterns (Cilliers 2000)**

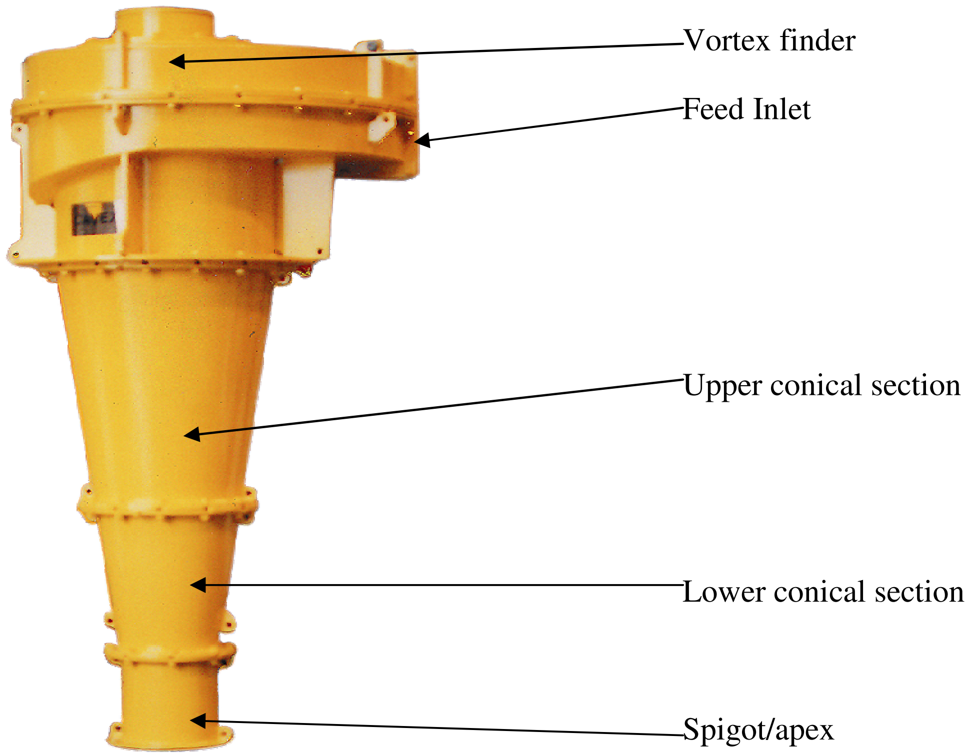



**Figure 2.2. Illustration of a 500mm classification hydrocyclone (Weir Minerals 2010)**

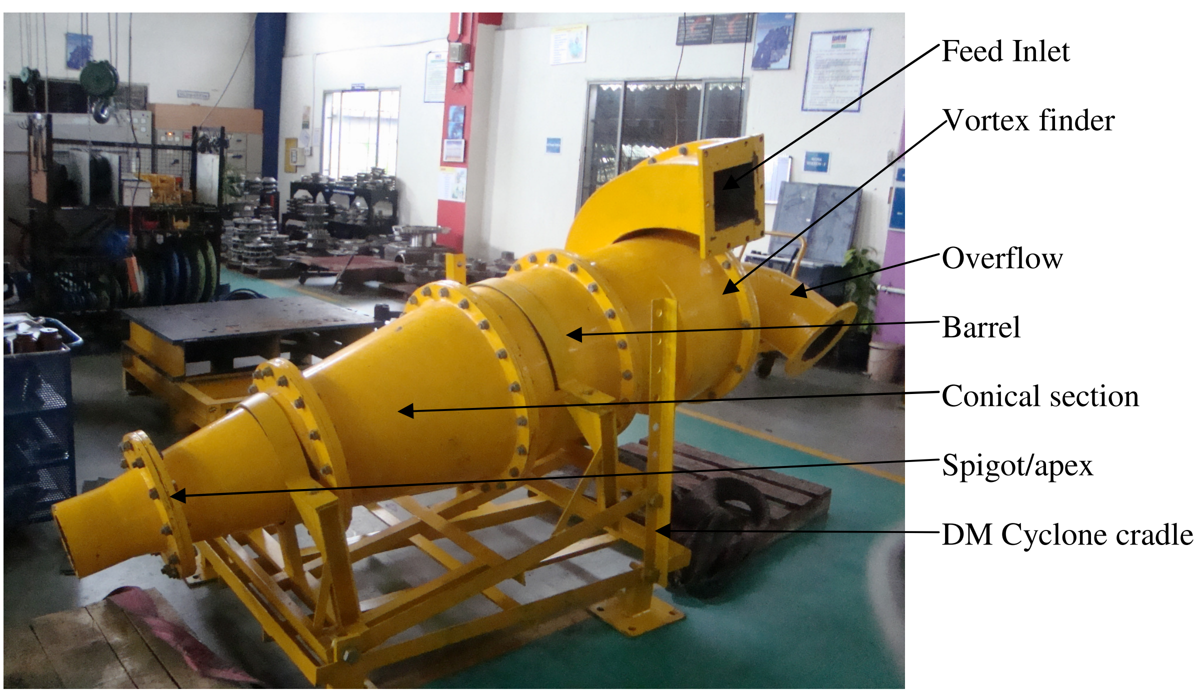



**Figure 2.3. Illustration of a 500mm DM cyclone (Weir Minerals 2010)**

The classical theory of hydrocyclone action is that the particles within the flow pattern are subjected to two opposing forces – an outward centrifugal force and an inwardly acting drag as presented in figure 2.4 (Wills, 1997). The centrifugal force developed accelerates the settling rate of the particles, thereby separating particles according to size and specific gravity (S.G). Faster settling particles move to the wall of the cyclone, where the velocity is the lowest, and migrate to the apex/spigot opening. Due to the action of the drag force, the slower settling particles move toward the zone of low pressure along the axis and are carried upward through the vortex finder to the overflow.

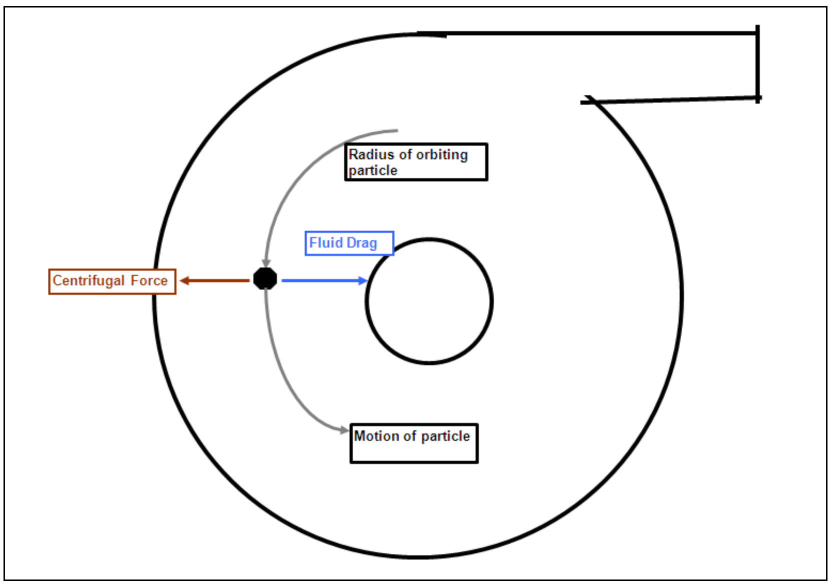



**Figure 2.4. Forces acing on an orbiting particle in the hydrocyclone (Weir minerals, 2010)**

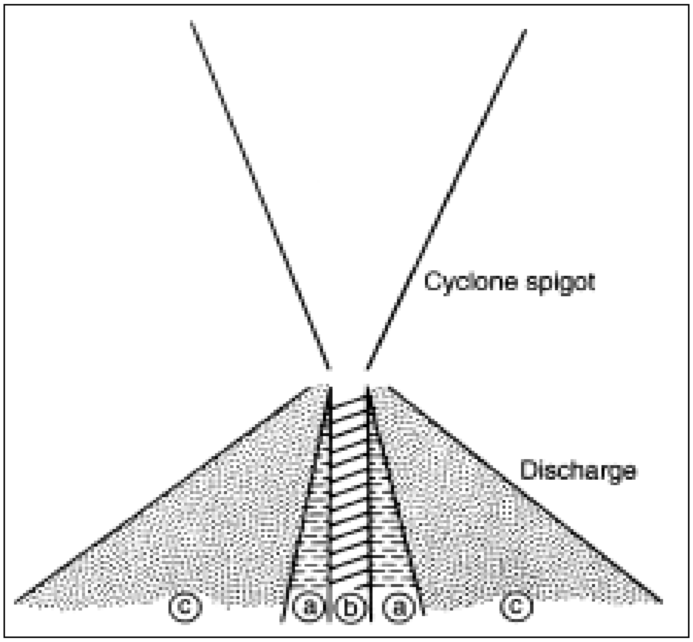
The two different cyclones will be evaluated separately.

## 2.1 CLASSIFICATION HYDROCYCLONES

### 2.1.1 OPERATING PRINCIPLE

The operating principle of a hydrocyclone is simple: the fluid carrying the suspended particles enters the cyclone tangentially, spirals downward and produces a centrifugal field in free vortex flow. Larger particles move through the liquid to the outside of the cyclone in a spiral motion and exit through the spigot with a fraction of the liquid (figure 2.1). Due to the limiting area of the spigot, an inner vortex, rotating in the same direction as the outer vortex but flowing upward, is established and leaves the cyclone through the vortex finder carrying most of the liquid and finer particles with it (Cilliers, 2000).

If the spigot capacity is exceeded, the air core is closed off and the spigot discharge changes from a high rotational spray discharge to an unstable operation that can lead to instabilities and finally to blockage of the equipment (Concha et al, 1996). This unstable discharge is known as roping. In spite of the fact that roping is an undesirable condition, hydrocyclones are operated in as close to roping conditions as possible to obtain the highest possible underflow concentration and thus minimizing the short circuit (Concha et al, 1996). According to Plitt et al (1987), the impact of rope discharge on cyclone performance is dependent on the feed solids concentration and quite possibly the cyclone size. This is especially the case at high feed solids concentration where the cyclone performance is very strongly affected, presumably through the interaction of the sediment layer with the classification process (Plitt et al, 1987). Figure 2.1.1 indicates the effect of the spigot size on the cyclone underflow.




*Zone (a) – correct operation; Zone (b) – "roping" – spigot too small; Zone (c) excessively dilute – spigot too large*

**Figure 2.1.1. Effect of spigot size on cyclone underflow (Wills, 1997)**

### *2.1.2 FEED CHAMBER DESIGN*

The area of inlet determines the entrance velocity and an increase in area increases the flowrate. The geometry of the feed inlet is essential in this regard (Wills, 1997). The most common shape of entry is developed from circular cross section to rectangular cross section at the entrance to the cylindrical section of the cyclone. This helps to "spread" the flow along the wall of the chamber. The inlet is normally tangential but involute feed entries are also common. Involute entries are said to minimize turbulence and reduce wear (Wills, 1997).

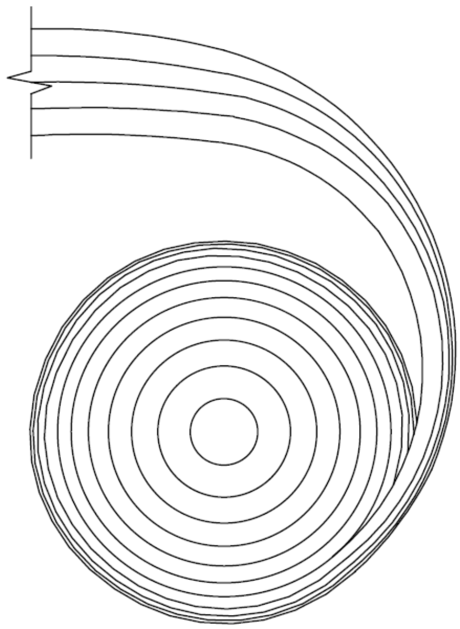


**Figure 2.1.2. Involute feed entry (Arterburn, 1982)**

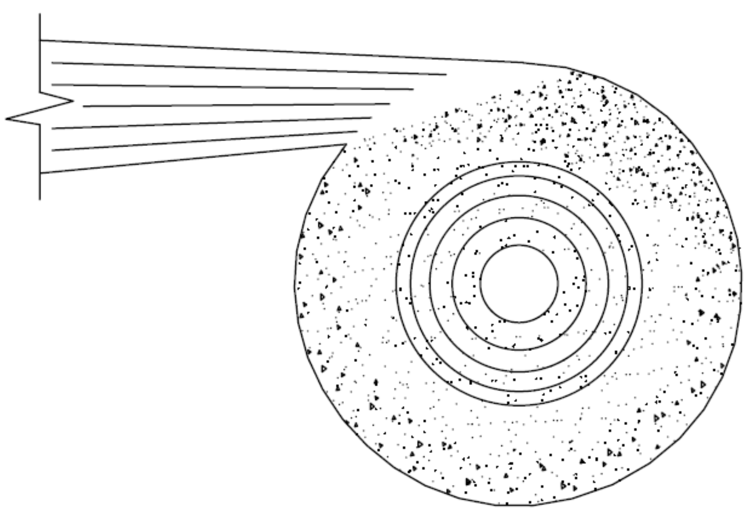


**Figure 2.1.3. Tangential feed entry (Arterburn, 1982)**

An alternative feed inlet to those presented above is the laminar spiral inlet (Cavex cyclones). For many years the internal flow in a hydrocyclone has been described as spiral within a spiral. The axial velocity of the external spiral is downwards and the internal spiral flow, close to the cyclone centre, goes up towards the vortex finder (Castro and Castillo, 2001). In recent years, with the use of laser doppler velocimetry and other techniques, many researchers have demonstrated a more complex nature of hydrocyclone flow. The understanding of the physics of the internal hydrocyclone flow motion has helped Weir Minerals group designers to improve hydrocyclone geometry. The hydrocyclone head geometry has the function of transforming kinetic energy into a centrifugal field. Mainly, the linear velocity of the fluid, delivered by the pump is transformed in rotational motion by the cyclone head geometry. The efficiency of this energy conversion process at the cyclone inlet head is related to an appropriate cyclone head geometry. The pressure drop in the cyclone is an indication of how good or bad the energy conversion process is. In other words, for the same flowrate, a greater pressure drop across the hydrocyclone indicates lower efficiency of energy conversion (Castro and Castillo, 2001). Castro and Castillo (2001) further explain that, focusing on the description of the cyclone internal flow at the head, it is determined that secondary motion occurs. The vortex finder and the cylindrical type geometry of the cyclone head can be described as two concentrical cylinders. The presence of vortices between the cyclone cylinder and the vortex finder external wall has been widely demonstrated. These vortices have a diameter very close to the gap between the "two cylinders" as can be seen in figure 2.1.4.

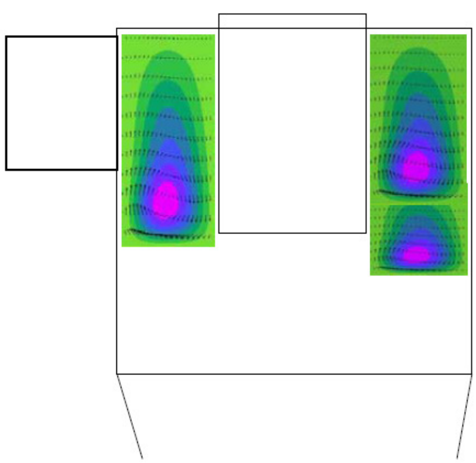


**Figure 2.1.4. Vortices at hydrocyclone inlet head (Castro and Castillo, 2001)**

It is evident that there are a lot of similarities between the vortices occurring within the feed chamber of a cyclone and the Taylor Vortices phenomena extensively described in the literature relating to cylinder viscometers (Castro and Castillo, 2001). The Taylor vortices kinetic energy is a function of the initial velocity and the gap between the concentric cylinders. Consequently, more energy will be wasted as the gap between inner and outer cylinders gets smaller. Extensive pilot plant work comparing a 360° inlet head volute cyclone against a conventional inlet head (tangential and involute), has shown a significant increase in cyclone capacity for the same cyclone at equivalent inlet area. The Cavex inlet head design allows less turbulence and a better use of the fluid kinetic energy at the hydrocyclone entrance by significantly reducing the intensity of eddies. Additionally, wear life is significantly increased as wear rate is proportional to $v^{2.5\text{-}3.0}$, where v is velocity (Castro and Castillo, 2001). The unique Cavex design is presented in figure 2.1.5.

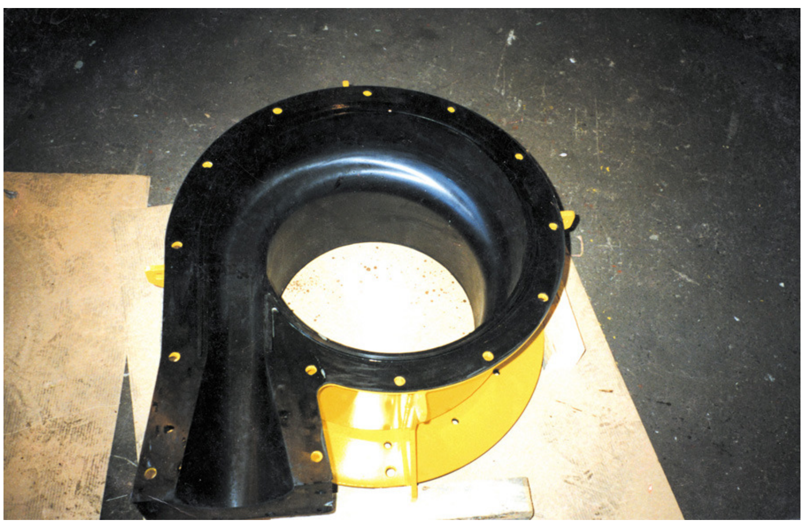


**Figure 2.1.5. Cavex inlet geometry**

### *2.1.3 CLASSIFICATION EFFICIENCY*

According to Cilliers (2000), in order to determine the classification performance of a hydrocyclone, a partition curve (also called the performance curve or Tromp curve after its founder) is used. The partition curve quantifies the weight fraction (or percentage) of each particle size fraction in the feed reporting to the underflow product. For any particle size d, the partition number, p(d), is calculated from:

$$p(d) = \frac{U.u(d)}{F.f(d)} \qquad \ldots(2.1.1)$$

The particle size at which the partition number equals 0.5 (50%) is called the $d_{50}$. Cilliers (2000) further explains that a fraction of fine particles always report to the underflow, hence experimentally observed partition curves do not asymptote to zero but to a minimum, called the bypass. This can be interpreted as a fraction of all particles in the feed bypassing classification and reporting directly to the underflow stream. Short circuiting of feed material to the overflow stream may cause the partition curve not to reach a partition value of 1 (100%) – but this is not common (Cilliers, 2000). The effect of bypass on the classification performance is taken into account by correcting the partition value:

$$c(d) = \frac{p(d) - r(d)}{1 - r(d)} \qquad \ldots(2.1.2)$$

The particle size at which the corrected partition number is 0.5 (50%) is called the corrected cut size ($d_{50c}$). Cilliers (2000) notes that it is often found that the bypass equals the water recovery from the feed to the underflow ($R_f$) although there is no fundamental reason why this should be so. A so-called fishhook may occur in the partition curve when, for particle sizes finer than that at the minimum partition value, progressively higher partition numbers are observed. This is more commonly observed for smaller diameter cyclones and is thought to be the result of turbulent dispersion. In such cases the water recovery may be significantly lower than the lowest partition value observed. Applying the correcting concept to such partition curves is meaningless (Cilliers, 2000). The partition curve gives a complete description of the selective separation of all sizes of solids entering a cyclone and can be used to predict the product size distribution and solids recovery for changes in feed size distribution. If the bypass is assumed to equal the water recovery, the liquid and volumetric balances can also be estimated (Cilliers, 2000).

In practice, the cut point is mainly controlled by the cyclone design variables such as inlet, vortex finder and apex openings and most cyclones are designed so that these are easily interchangeable (Wills, 1997). In order to obtain these variables the fundamental parameters must be determined in order to design and operate a hydrocyclone. The modelling of hydrocyclones is performed by either describing the fluid flow and particle motion within the cyclones, or by developing empirical (or semi-empirical) relationships between operating variables and measured responses. Fundamental models are appealing from a rigorous standpoint but have difficulty in describing satisfactorily the complex particle-particle and particle-fluid interactions for hydrocyclones operating at higher solids concentrations (Cilliers, 2000).

### *2.1.4 EMPIRICAL OR SEMI-EMPIRICAL MODELS*

Empirical or semi-empirical models which relate the parameters of the partition curve to cyclone design and operating variables are generally used for industrial hydrocyclone modelling and simulation. (Cilliers, 2000). Models like the Equilibrium orbit theory, Residence time theory, Crowding theory and computational fluid dynamics (CFD) have been developed with the objective to understand the principles that govern size separation in hydrocyclones (Cilliers, 2000).

CFD is the preferred approach for fundamentally based modelling of hydrocyclone performance. Complete flow modelling of the hydrocyclone involves predicting the liquid phase velocities, the slurry concentration profile, the turbulent viscosities and the slip velocities of particles with respect to the liquid phase for a range of particle sizes before predicting the partition curve. The solution is complex, because the governing fluid flow equations are nonlinear, simultaneous partial differential equations (Cilliers, 2000).

According to Cilliers (2000), the general approach to develop a complete CFD based model of a hydrocyclone must include a wide range of components. If it is assumed that variations of local density and viscosity are small for dilute slurries and that particle-particle interactions are negligible, the fluid and particle modelling can be decoupled. Liquid velocities are predicted by combining the fluid transport equations for the vorticity stream function and angular spin velocity with a modified Prandtl mixing length model which varies both radially and axially, for the turbulent viscosity. The set of simultaneous, nonlinear partial differential equations are solved by overlaying the hydrocyclone dimensions with a rectangular grid and using appropriate boundary conditions at the solid walls and liquid-air interface, to solve conditions within each cell of the grid. By balancing all the forces on the particle, the particle motion with the respect to fluid can be computed. The particle trajectories are found by calculating axial and radial slip velocities with respect to the fluid. Size classification performance is determined by following a particle of a given size from the inlet until it exits. This computation is repeated for each particle size across the inlet diameter yielding the partition curve. Advances in CFD methods such as computation, grid generation, numerical methods and computing resources are increasing the applicability of this modelling technique to improve hydrocyclone designs (Cilliers, 2000).

Empirical hydrocyclone models use the partition curve as a basis for describing size separation (Cilliers, 2000). Plitt (1976) was the first to document an empirical model to determine the sharpness of separation and therefore allow direct simulation of expected performance without any test work. The Plitt model is often used as a basis for development of models that include further variables, such as, for example, angle of inclination, or for an operating range in which the model has not been tested (Cilliers, 2000). According to Plitt (1976), the following four parameters for a given set of conditions make it possible to determine the complete mass balance together with the size distributions of the products of the cyclone:

1. The separation or cut size
2. The flow split between overflow and underflow
3. The sharpness of classification
4. The capacity – pressure drop

The fluid flow patterns of the hydrocyclone were initially documented by Kelsall (1952) and this work established the framework of work by Bradley (1964) and further work by Lynch and Rao (1975). Plitt (1976) concluded that the mathematical model of the hydrocyclone classifier which has been formulated as a result of his work is embodied in four basic equations. For empirical models, the Rosin-Rammler function describes the reduced partition curve (Cilliers, 2000).

$$c_i = 1 - \exp[-0.693 x_i^m] \qquad \text{…(2.1.3)}$$

where m indicates the sharpness of separation and $x_i$ is:

$$x_i = \frac{d}{d_{50c}} \qquad \text{…(2.1.4)}$$

For the separation or cut size, it is common practice to express the cut size in terms of the $d_{50}$ size. The $d_{50}$ is that size which has equal (50%) probability of reporting to either product from the classifier i.e. underflow or overflow (Plitt, 1976). The equation relating to the corrected $d_{50}$ size to the hydrocyclone variables is:

$$d50 = \frac{50.5 D_c^{0.46} D_i^{0.6} D_o^{1.21} \exp[0.063\Phi]}{D_u^{0.71} h^{0.38} Q^{0.45} (\rho_s - \rho)^{0.5}} \qquad \text{…(2.1.5)}$$

Units:

$d_{50}$ – microns

$D_c$, $D_i$, $D_o$, $D_u$, h – centimetres

$\rho_s$, $\rho$ – g/cm$^3$

$\Phi$ = percent by volume

Size distribution, or surface area of the solids, also affects the apparent viscosity of slurries. Lynch and Rao (1975) investigated how the feed size analysis affects cyclone performance. Reworking equation 1 to include the effect observed by Lynch and Rao (1975) due to the change in the particle size of the feed, equation 2.1.5 becomes (Plitt 1976):

$$d_{50} = \frac{50.5 D_c^{0.46} D_i^{0.6} D_o^{1.21} \exp[0.08\Phi/F_{50}^{0.052}]}{D_u^{0.71} h^{0.38} Q^{0.45} (\rho_s - \rho)^{0.5}} \qquad \ldots(2.1.6)$$

where $F_{50}$ is the weight median size of the feed solids in microns (50% passing size). Plitt (1976) notes that the effect of the feed size analysis in equation 2.1.6 is not very significant and, for normal situations, can be neglected. Equation 2.1.6 does, however, show the trend that as the particle size becomes finer, the $d_{50}$ size increases.

Plitt (1976) defined the flow split between overflow and underflow as per equation 2.1.7, after considering the implications of feed solids content, vortex finder diameter, underflow diameter, free vortex height, throughput, inlet size, cyclone diameter and solid density. The flow split was then defined as:

$$S = \frac{1.9 (D_u/D_o)^{3.31} h^{0.54} (D_u^2 + D_o^2)^{0.36} \exp[0.0054\Phi]}{H^{0.24} D_c^{1.11}} \qquad \ldots(2.1.7)$$

Units:

$D_o$, $D_u$, h – centimetres

H – meters

$\Phi$ = percent by volume

From 297 experimental tests, Plitt (1976) developed a regression equation with a correlation coefficient of 0.9. The resulting equation for pressure drop in a hydrocyclone is presented in equation 2.1.8.

$$P = \frac{1.88\ Q^{1.78} \exp[0.0055\Phi]}{D_c^{0.37}\ D_i^{0.94}\ h^{0.28}\ (D_u^2 + D_o^2)^{0.87}}$$

…(2.1.8)

Units:

$D_c$, $D_o$, $D_u$, $D_i$, h – centimetres

Q – litters/minute

P – kilo Pascal (kPa)

Φ = percent by volume

Svarovsky (1984) notes that an increase in pressure drop usually leads to:

- Higher throughput.
- Finer cut size.
- Decrease in the underflow to throughput ratio.
- Increase in total efficiency.
- More concentrated underflow.
- Cleaner overflow.

Plitt (1976) produced the following equation to determine the sharpness of separation:

$$M = 1.94 \exp[-1.58\ R_v]\ ((D_c^2.h)/Q))^{0.15}$$ …(2.1.9)

Units:

$D_c$, h – centimetres

Q – litters/minute

The above representation as formulated by Plitt (1976):

- Enables the performance of a hydrocyclone to be calculated with reasonable accuracy when no actual experimental data are available.
- The equations clearly reveal the independent effects and relative importance of all the major variables which influence the operation of a hydrocyclone.
- Although not necessarily more accurate than other hydrocyclone models, the broad data base and the inclusion of all the significant variables make the equations applicable over a wide range of operating conditions.
- With experimental data, the constants in the equations may be appropriately adjusted to serve as an operational model.

The effects of design variables are crudely summarized by Svarovsky (1984) as follows:

**Table 2.1.1. Effects of design variables in hydrocyclones (Svarovsky, 1984)**

| To increase capacity | To reduce the cut size ($d_{50}$) | To improve sharpness of classification | To increase the flow ratio |
|---|---|---|---|
| Increase cyclone diameter | Reduce cyclone diameter | Increase outlet diameter | Increase underflow diameter |
| Increase inlet diameter | Reduce inlet diameter | Increase cone angle | Reduce overflow diameter |
| Increase outlet diameter | Reduce outlet diameter | | Increase body length |
| Increase body length | Reduce cone angle | | Reduce vortex finder length |
| reduce vortex finder length | Reduce vortex finder length | | |
| | Increase body length | | |

## 2.2 *DENSE MEDIA (DM) CYCLONES*

### 2.2.1 *TEST WORK PROCEDURES*

Dense medium processes are based on the principles of float and sink processes. The first attempt to use this sink and float principle on an industrial scale was made by Sir Henry Besssemer who patented the first dense medium process in 1858 (England et al, 2002). In the laboratory clean coal is separated from refuse by immersing the raw coal in a fluid having a density intermediate between that of clean coal and of gangue material (England et al, 2002).

For coal sink and float analysis, organic liquids such as Certigrav, or mixtures of carbon tetrachloride, petroleum ether and bromoform, or a solution of zinc chloride in water, can be used to obtain an almost perfect separation of the coal from the dirt by means of the property of relative density. The influence of size and shape for particles bigger than 0.5mm is negligible (England et al, 2002).

For mineral separation in sink and float analysis, or heavy liquid separation (HLS), tetrabromoethane (TBE) is the main chemical utilised for HLS (Wills, 1997). TBE has a density of 2.96t/m$^3$ and it is diluted with acetone (density of 0.96t/m$^3$) to produce S.Gs below 2.96. S.Gs of higher than 2.96 are produced by producing a suspension with ferrosilicon (FeSi). Additional acetone is used after each cut is completed to wash any excess TBE from the products. On completion of HLS test work the masses of all density fractions are recorded and each fraction is sub-sampled and prepared for chemical analysis (Singleton and de Wet, 2007). From these weight fractions and chemical analysis, a partition curve or washability curve can be constructed. Density intervals of 0.05 or 0.1 or 0.2 relative density units can be used (England et al, 2002). Figure 2.2.1 illustrates the HLS/floats and sinks method procedure.

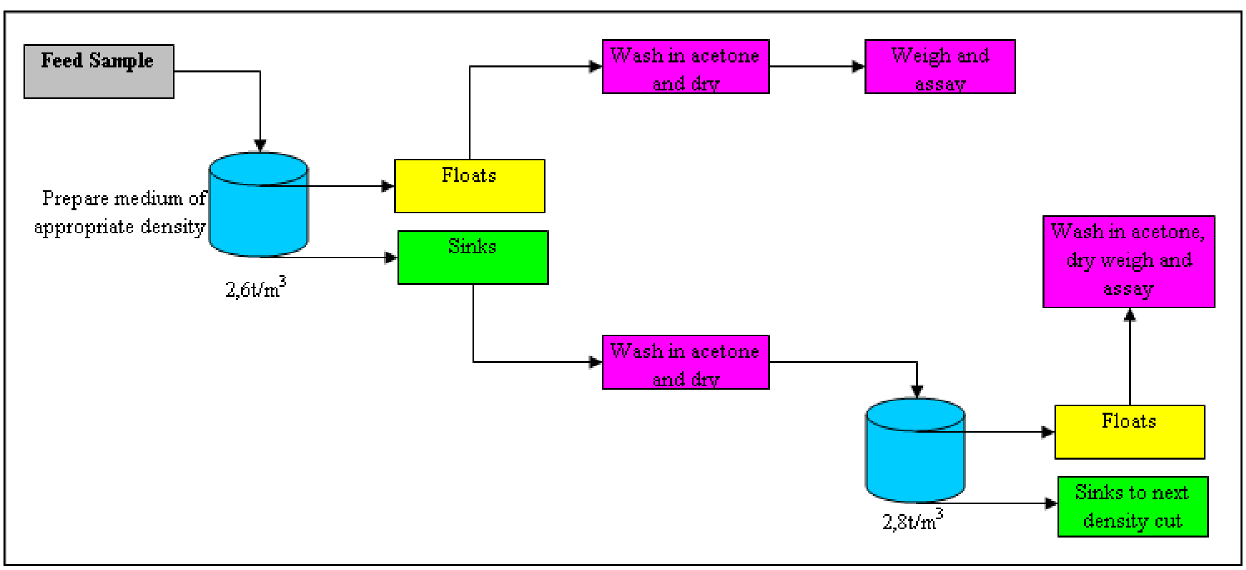



**Figure 2.2.1. Overview of HLS/sinks and float technique (Singleton and de Wet, 2007)**

The principle for dense medium separation is presented in figure 2.2.2.

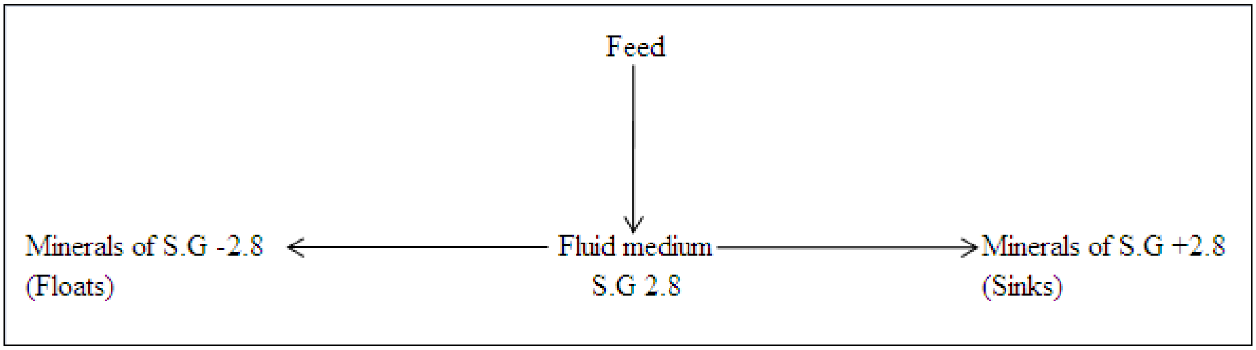



**Figure 2.2.2. Principle of dense medium separation (Wills, 1997)**

Float and sink analysis or HLS are carried out for three main reasons (adapted from England et al, 2002):

- Determination of washability characteristics of coal or minerals.
- Evaluation of the efficiency of separators.
- Day to day plant control.

From the HLS/sink and float analyses, the partition curve is derived in order to evaluate the efficiency of the operation. The partition curve provides (adapted from England et al, 2002):

- The yield of clean coal, or mineral recovery, from the washing operation.
- The float and sink analysis of the clean coal or upgrade of valuable minerals.
- The float and sink analysis of the discard, gangue minerals or mine dilution.

For practical reasons, the entire partition curve is seldom derived from test work. Instead, parameters derived from the partition curve are used, including (England et al, 2002):

- Separating density: the point at which the curve passes the 50% partition factor is normally defined as the partition density ($d_{50}$). This point is also known as the Tromp cut point and partition density.
- Ecart Probable (moyen) (EPM): defined as one half of the difference between the relative densities corresponding to the 75% and 25% ordinates as shown in the partition curve (($d_{75} - d_{25}$)/2).

In addition, the HLS/sink and float analysis test work also provides (adapted from England et al, 2002):

- The organic efficiency: the ratio (normally in percentage) between actual yield or recovery of a desired product and the theoretical possible yield or recovery, based on the reconstituted feed.
- Washability curves: show the relationship between ash content, or gangue minerals, and the amount of float or sink produced at any particular relative density cut point.
- Densimetric curve: shows the relationship between the relative density and the yield of clean coal or upgrade of minerals that float or sinks at that relative density.
- Ash curve: shows the relationship between cumulative yield for the float and ash content when evaluating coal.

### *2.2.2 SEPARATING MEDIA*

The ideal separating media in dense media processes would be a true liquid which is low in cost, immiscible with water, capable of adjustment over a wide range of relative densities, stable, non toxic, non corrosive and low in viscosity (England et al, 2002). Even though liquids are used in laboratory testing, they are too costly for industrial scale (England et al, 2002) and often toxic (Wills, 1997). The mediums now employed in all commercial dense medium processes worldwide are a suspension in water of insoluble solid particles of high relative density. This dense powder is typically ferrosilicon or magnetite in water. Ferrosilicon, also known as FeSi, is used for high density applications (medium density 3.2-4.2t/m$^3$) (Grobler et al, 2002). A FeSi suspension must have a large fraction of solids in water to achieve high densities. For example, to achieve a medium density of 4t/m$^3$, 7 tonnes of FeSi (solid density of 7t/m$^3$) should be added to 1m$^3$ of water to make up 2m$^3$ of heavy medium (Grobler et al, 2002). Similarly, 1 tonne of magnetite (solid density of 5t/m$^3$), mixed with 2m$^3$ of water, will give 2.2m$^3$ of heavy medium with a pulp density of 1.36t/m$^3$. England et al (2002) notes that magnetite is now the only solid used in the South African coal industry, even though sand has been used in the past. South African coal preparation plants operate between 1.3t/m$^3$ and 2.0t/m$^3$ cut points (England et al, 2002). Magnetite is relatively cheap and is used to maintain bath densities of up to 2.5t/m$^3$ (Wills, 1997).

Particle size of the medium plays an important part in various properties of media; the coarser the particle, the greater the risk of settling out of the solids. A medium from which the solids settle out rapidly is said to be unstable. Hence, coarse particles create unstable conditions, whereas fine particles create stable conditions (England et al, 2002). The shape of the medium particles (FeSi and magnetite) depends on the manufacturing process (milling or atomisation) (Grobler et al, 2002).

The stability of the medium is the property of a suspension considered as a non homogeneous two phase system – it is the rheology of the solid phase in an environment constituted by the liquid phase. The relative movement of the solids in the liquid phase under mass and surface forces determines the degree of homogeneity of the suspension, and important medium property in DMS (Grobler et al, 2002). In other words the stability of the medium suspension can be measured by the length of time required for the medium solids to settle out (England et al, 2002). Grobler et al (2002) further notes that the medium stability determines the density gradient of the medium in the separation zone and thus directly influences separation sharpness. The finer the size of the solids in the medium, the more stable the suspension and the lower the relative density of the solid, the more stable the suspension (England et al, 2002). An ideal medium has a high stability which results in high medium densities, fine medium particles and the presence of low density contamination from ore slimes (Grobler et al, 2002).

The rheology of the medium (thick and fast settling suspension) can be best described by its viscosity and stability. Viscosity is the measure of the medium resistance to fluid flow while stability is a measure of the tendency of the medium to settle. These two properties are strongly influenced by the parameters such as medium density, particle shape, particle size distribution and the level of contamination with slimes. The viscous characteristics of the dense medium are generally non Newtonian (viscosity is a function of shear rate) and the term apparent viscosity (at defined shear rate) is preferred (Grobler et al, 2002). Finely ground suspensions of concentration of below 30% by volume in water behave essentially as simple Newtonian fluids. Above this concentration, however, the suspension becomes non-Newtonian and a certain minimum stress, or yield stress, has to be applied before shear will occur and the movement of the particle can commence (Wills, 1997). It is further noted by Grobler et al (2002) that an ideal medium has a low viscosity to maximise separation and pump efficiency. A high viscosity is undesirable because it reduces the velocity of mineral particles being separated, increasing the chance of misplacement and reducing the separation efficiency. A low viscosity is typically obtained for a low medium density, coarse particles, smooth rounded particles and clean uncontaminated medium. Factors controlling viscosity are summarised by Napier-Munn and Scott (1990):

- Medium density – viscosity increases with solids concentration and thus with medium density, in a non linear manner, the increase being rapid above a certain critical concentration.
- Solids density – this controls the solids concentration required to achieve a given medium density; high density of solids requires a lower concentration (lower viscosity) to achieve a given medium.
- Particle size distribution – rounded or smooth particles produce lower viscosities than angular or rough particles.
- Fine contamination – contaminants, such as slimes from the ore or coal, usually increase viscosity due both to their lower solids density and fine particle size.

- Demagnetisation – commercial media are generally magnetic (FeSi and magnetite) to allow simple recovery and regeneration processes. Passage through the magnetic separations in the medium recovery circuit induces a residual magnetisation which causes flocculation or agglomeration of the magnetised particles. This effect generally increases viscosity. The effect can be minimised by demagnetising or depolarising the medium after medium recovery.

The medium is contaminated as the percentage of slimes increases. The increase of slimes increases the total volume of solids in the suspension and hence the viscosity and stability are also increased (England et al, 2002). Ore slimes contaminating the medium are usually smaller than 10µm (Grobler et al, 2002). The high viscosity mediums results in poor separation as the cut point tends to be higher than the relative density of the medium (England et al, 2002). Napier-Munn and Scott (1990) notes that the effect between medium viscosity and density for the contaminated (normal) and uncontaminated (fresh) medium is large. At a density of $2.95 t/m^3$, the contaminated medium has a viscosity about three times that of the fresh material. This differential increases with increased density.

To maintain the relative density of the suspension without it becoming viscous as well as reducing operating costs it is necessary to continuously extract some of the contaminating slimes from the medium. Napier-Munn et al (1995) conclude that there are normally only two possible routes by which medium can be lost from the plant:

- Medium adhered to the products of the separation, after draining and washing on screens.
- Medium is present in the final effluent from the medium regeneration process, usually magnetic separator but sometimes also including hydrocyclones, selling cones or other solid-liquid separation devices.

Napier-Munn et al (1995) also identified the two most important aspects of increased consumption, both of which have important implication for the minimisation of medium loss in dense medium plants:

- Medium loss increases with operating density: The increase occurs in both adhesion losses from drain and rinse screens and magnetic effluent losses from separator and hydrocyclones. The magnitude of the increase is large for quite modest increases in density. Operating density should therefore be minimised. This policy would favour separating vessels which cut at densities somewhat higher than the medium density, such as cyclones, particularly those with large cone angles.
- Medium consumption increases with plant downtime: This is as a result of water dilution by "washing" the medium back into the circuit after spillage or dumping. Scheduled and unscheduled maintenance should be minimised.

Both magnetite and FeSi have magnetic properties and are amenable to recovery and densification (England et al, 2002) (Wills, 1997). Medium preservation is as essential as coal yield or mineral recovery in the DMS process. Due to the fact that DM cyclone performance in mineral applications (FeSi dense media) is a strong function of medium viscosity, Napier-Munn and Scott (1990) suggested that demagnetising coils should be installed and operated in FeSi DM cyclone plants because

- Elevated viscosities can have a deleterious effect on cyclone performance.
- Sudden and unpredictable viscosity related declines in cyclone performance can occur and the use of coils would at least limit the probability of such events occurring.
- Napier-Munn and Scott (1990) showed that demagnetising coils do reduce the viscosity of circulating medium.

### *2.2.3 DENSE MEDIA SEPARATORS*

There are numerous variants of DM plants and flow sheets but the principles remain the same. As mentioned, dense medium baths and centrifugal separators are the most common DM separators employed. Only a few DM baths designs are now in commercial use. DM baths are classified into two main classes, deep baths and shallow baths. Deep baths contain more medium than shallow baths with the same rated capacity but are less prone to be affected by the accidental addition of water to medium (England et al, 2002). Deep baths include Chance sand cone, Barvoys and Tromp deep baths. Shallow baths include the Wemco drum, Drewboy, Teska, and Norwalt baths.

Cyclone dense medium separators have now become widely used in the treatment of ores and coal. DM cyclones provide a high centrifugal force and a low viscosity in the medium enabling much finer separations to be achieved than in gravitational separators (Wills, 1997). England et al (2002) summarises the history of the DM cyclone as follow: In the early days of coarse coal treatment by dense medium in Europe loess was used as a medium solid. Newly developed cyclones were used to reclaim and thicken the loess suspension. On an occasion when the cyclone thickener at the Maurits mine in the Netherlands plugged, the overflow was found to be filled with clean coal free of impurities. This observation led to the development of the cyclone as a cleaning device. During the period 1950-1980, the use of the DM cyclone gradually became widespread throughout the world and, over latter years, several other centrifugal washers were developed.

The principle of operation of the DM cyclone developed by Dutch state mines (DSM) is very similar to that of the conventional hydrocyclone where both the ore and medium are subject to drag and centrifugal forces, even though the DM cyclone separates on the basis of density and not size (Bosman and Engelbrecht, 1997) – see section 2.1. and figure 2.1. From figure 2.1, the overflow will contain the lighter than medium particles and the underflow will contain the heavier than medium particles (figure 2.2.2). DM cyclones currently process the vast majority of tonnes fed to coal preparation plants (Wang et al, 2009).

The feed to the DM cyclone, which is a mixture of dense medium slurry and coal/ore, enters tangentially near the top of the cylindrical section under pressure, thus promoting a strong swirling flow. The refuse or high ash particles move towards the wall where the axial velocity vector points downwards, and are discharged through the spigot. The lighter clean coal (or gangue minerals) moves towards the longitudinal axis of the cyclone where there is usually an axial air core present and the axial velocity vector of slurry flow points upward and passes through the vortex finder (Wang et al, 2009).

The flow in a DM cyclone is very complicated with the presence of swirling turbulence, air core and particle segregation and involves multiple phases: gas, liquid, coal and magnetic/non magnetic particles of different sizes and densities. Normally, the slurry including water, magnetite and non magnetic content is termed the medium (Wang et al, 2009).

In other words, the DM cyclone makes use of centrifugal force in order to obtain larger forces that can operate on the particles. This causes the "heavier" particles to move rapidly towards the wall of the cyclonic vessel and the "lighter" particles to move rapidly towards the centre of the vessel (England et al, 2002). The magnitude of the gravitational and buoyant forces that separate the particles is a primary consideration because it governs the velocity with which the particles separate, which in turn determines the capacity of the plant (England et al, 2002). In a static bath the net gravitational force minus the buoyant force may be given as:

$$Fg = (Mp - Mf)g \qquad \ldots(2.2.1)$$

Where
Fg = gravitational force
Mp = mass of particle
Mf = mass of fluid displaced by particle
g = acceleration of gravity

For float particles, Fg will have a negative value, i.e. Mf > Mp. For sink particles, Mf < Mp and the value will be positive. In a DM cyclone, relative density separations result primarily from the utilisation of gravitational and buoyant forces. In a DM cyclone, however, the acceleration of gravity is substituted by a centrifugal acceleration:

$$Fc = (Mp - Mf)(V^2/r)$$

Where

Fc = centrifugal forces

V = tangential velocity

r = radius of cyclone

Therefore, the smaller the radius, the greater the centrifugal force acting on the particles. Because of this, fine particles can be separated with smaller diameter cyclones. However, van der Walt (2002) commented that fine DMS is only likely to take off on a big scale with the development of a suitable large single stage design for fine (-0.5mm) DMS. The single large unit will be more practical and is likely to be developed through modern technology like CFD.

According to England et al (2002), the DSM cyclone is suitable for treating coal and minerals over a wide size range, 80mm – 0.1mm, with two major qualifications:

- The size of the cyclone required increases with the top size processed (maximum particle size taken as one third of the cyclone inlet diameter).
- The efficiency of separation of the -4mm fraction deteriorates significantly with large cyclones greater than 800mm in diameter. This statement is contrary to the view of the Australian coal industry, where Sedgman alone installed eighty +1000mm DM cyclones since 1999. Furthermore, there is no operating data proving that the Ep worsens for larger diameter DM cyclones, nor of a breakaway size for large diameter DM cyclones (Mackay et al, 2009).

The centrifugal force acting on the particle in the inlet region is typically 20 times greater than the gravitational force in a static bath. In the conical section of the DM cyclone, the tangential velocity is further increased and at the apex it is over 200 times greater than gravity (England et al, 2002).

The other essential operating parameter for DM cyclones is the feed pressure or the pressure at which the pulp is introduced into the cyclone. The feed pressure controls the forces within the DM cyclone and, for coal, typically vary between 70kPa and 105kPa. The feed can be introduced either via a pump or gravity fed (England et al, 2002). The capacity of the cyclone increases in proportion to the square root of the operating head which is normally set at a minimum 9 x "D", the cyclone diameter. The pressure is a compromise between the system power requirements, pump sizes and wear, versus the minimum acceptable separation efficiency (England et al, 2002).

### *2.2.4 DENSE MEDIA CYCLONE DEVELOPMENTS*

DSM recorded many principle guidelines for DM cyclones which are used to this day. The DSM guidelines are summarised in table 2.3.4.1.

**Table 2.3.4.1. DSM DM cyclone selection guidelines vs. current trends (Honaker, 2011)**

| **Parameter** | **DSM recommendations** | **Current manufacturing trends** |
|---|---|---|
| Cyclone Diameter ($D_c$) | <800mm | Up to 1500mm |
| Inlet size ($D_i$) | 0.2 x $D_c$ | 0.20, 0.25, 0.30 x $D_c$ |
| Vortex finder ($D_o$) | 0.43 x $D_c$ | 0.43 – 0.50 x $D_c$ |
| Spigot or apex ($D_u$) | 0.3 x $D_c$ | 0.3 – 0.4 x $D_c$ |
| Barrel length | 0.5 x $D_c$ | 0.5 to 2.5 x $D_c$ |
| Cone angle | $20^o$ | $20^o$ |
| Cyclone feed pressure | $9D_c$ | $6D_c$ to $18D_c$ |

Even though the normal flow pattern of the DM cyclone is similar to the classification cyclone (figure 2.1), recent research work is being conducted utilising CFD and other models in order to describe the flow and performance of a DM cyclone more accurately. As mentioned, CFD is the preferred approach for fundamentally based modelling of hydrocyclone performance. Complete flow modelling of the hydrocyclone involves predicting the liquid phase velocities, the slurry concentration profile, the turbulent viscosities and the slip velocities of particles with respect to the liquid phase for a range of particle sizes before predicting the partition curve. The solution is complex because the governing fluid flow equations are nonlinear, simultaneous partial differential equations (Cilliers, 2000).

Various publications are available describing CFD modelling. Suasnabar and Fletcher (1999) developed a Newtonian model based on the Eulerian-Lagnarian approach. Hu et al (2001) developed a model to predict partition curves of the smaller particle size fractions from experimental data of density tracer tests at large particle sizes in DM cyclones.

Narasimha et al (2006a) used Large Eddy Simulation (LES) coupled with the mixture model and lagrangian particle tracking to study medium segregation and coal particle partitioning in a DM cyclone using the software Fluent. The predicted Ep values derived from these simulations were very close to the experimental values although a slight deviation in the cut point predictions were observed. Narasimha et al (2006b) also illustrated that numerical simulation of turbulent driven flow in a DM cyclone with magnetite medium using Fluent can be used to predict the air core shape and diameter close to experimental results when measured by gamma ray tomography. Narasimha et al (2006b) noted that the LES turbulence model with Mixture multi phase model can be used to predict the air/slurry interface accurately although the LES may need a finer grid.

Narasimha et al (2007b) developed multi phase simulations of turbulent driven flow in a DM cyclone with magnetite medium in Fluent using the Algebraic Slip Mixture model to model the disperse phases and the air core and both the LES turbulence model and Reynolds Stress Models (RSM) for turbulence closure. The predicted air core shape and diameter were found to be close to the experimental results measured by gamma ray tomography. Narasimha et al (2007b) further concluded that it is possible to use LES turbulence model with ASM multi phase model to predict the air/slurry interface accurately.

Narasimha et al (2007a) further developed a CFD model of the DM cyclone using Fluent by coupling component models for the air core, the magnetite medium and coal particles. During this work, the partition characteristics for the DM cyclone for particles between 0.5mm and 8mm in diameter were modelled using Lagrangian particle tracking. For the first time the pivot phenomenon, in which partition curves for different sizes of coal pass through a common pivot point, had been successfully modelled using CFD. The values of Ep predicted by the Lagrangian particle tracking are very close to the experimental values although cut point predictions deviate slightly. This comprehensive CFD model provides a tool for new DM cyclone design with clear advantages over approaches based on constructing and trialling of new designs experimentally.

Further numerical simulations using CFD software Fluent was conducted by Shen et al (2009) on small DM cyclones. This was followed by a mathematical model to describe the flow system in the DM cyclone by means of combining Discrete Element Method (DEM) with CFD (Chu et al, 2009a). This was followed by further numerical studies on a 1000mm DM cyclone by Wang et al (2009) and particle density distribution studies utilising CFD-DEM on a 1000mm DM cyclone by Chu et al (2009b). CFD analysis on large DM cyclones will make the improvement of DM cyclones possible without experimental data and may also allow the development of a large diameter DM cyclone for the treatment of fine coal beneficiation as requested by van der Walt (2002).

# 3 RESEARCH METHODOLOGY

The following procedures were used during this project:

The project included a physical survey of the existing DM cyclone Market in Southern Africa to establish the current installed base for DM cyclones in the coal and mineral industries. The surveys involved the completion of questionnaires to various operating sites. From the survey, a DM cyclone size and construction material entry level was selected. The primary objective in selecting the material of construction was cycle life of the cyclone (wear rate) to ensure minimum downtime. High alumina ceramic tiles were selected as material of construction. This was followed by the development of design drawings and slip cast moulds for the individual radius tiles used in the Cavex feed chamber. The slip cast moulds for the Cavex laminar spiral feed chamber design was individually developed with 3D modelling to ensure the Cavex feed chamber shape is preserved, and as a result, the turbulence in the feed chamber is reduced.

Local fabrication facilities were set up in South Africa which included steel fabrication facilities for the production of the cyclone casing as well as ceramic slip cast mould facilities. Each feed chamber tile, containing +92% alumina, is produced by individual slip cast and fired at +1200$^{o}$C. The cone tiles are standard engineered tiles (+92% alumina) at various thicknesses to preserve the Cavex geometry. The prototype Cavex laminar spiral feed chamber DM cyclone was developed after a trial site was secured. The objective was to evaluate a DM cyclone with a laminar spiral feed inlet with the ultimate aim to provide industry with an alternative DM cyclone that will reduce operating costs whilst providing a superior final product for the mineral industry.

The facilities of Weir Minerals Africa were used for the fabrication of the 650CVXT cyclone. A proactive approach towards health, safety and the environment was adopted. Furthermore, conformance to all appropriate health, safety and environmental systems, OHSAS 18001 and ISO 14001 are ensured at all times. In line with this all work related to the development and fabrication of a DM cyclone was executed after:

- Compliance with all statutory legislation, by-laws and other requirements as determined by Weir Minerals Africa in respect of the above.
- Promoting and maintaining the principles of health and safe working practices within Weir Minerals Africa through training and continued awareness programmes.
- Identifying all hazards and risks and effectively managing these thereby preventing all forms of incidents, accidents, ill health and pollution within Weir Minerals control.
- Reducing the use of natural energy resources and promoting the principles of waste management.
- Establishing the requisite emergency procedures to deal with health, safety and environmental issues.
- Promoting health, safety and environmental awareness amongst Weir Mineral's contractors, visitors, suppliers and customers.
- Ensuring ongoing conformance and continual improvement through internal audits and by setting objectives and targets.

# 4 RESULTS AND DISCUSSION

The Research and Development (R&D) project consisted of two phases. The first phase involved design and fabrication of the newly developed laminar spiral feed chamber DM cyclone, followed by installation and commissioning phase. The following subchapters outline the results of the R&D.

## *4.1 LAMINAR SPIRAL FEED CHAMBER DM CYCLONE DESIGN*

All equipment and resources were provided by Weir Minerals Africa. A material survey on the current installed DM cyclones indicated that +92% Alumina ceramic tiles lasted about 1.5 times as long as 27% Cr cast iron. Silicon Carbide was excellent in high wear applications like spigots, but if the entire cyclone was cast in Silicon Carbide, the impact of the slurry caused the Silicon feed chamber to crack when pump fed. The market survey was comprehensive and included the DM cyclone installation sizes of 72 different operations across Southern Africa. After the market survey, it was concluded that +92% Alumina tiles were the most cost effective material of construction for DM cyclones. The 27% Cr Cast iron cyclones are almost exclusively used in diamond operations. The DM cyclone installed base survey is summarised in table 4.1.1.

**Table 4.1.1. DM cyclone installed base survey in Southern Africa**

| Cyclone size (mm) | Ceramic tiled cyclones | 27% Cr Cast iron | Silicon Carbide |
|---|---|---|---|
| **1150** | 2 | 0 | 0 |
| **1000** | 5 | 0 | 0 |
| **900** | 7 | 0 | 0 |
| **800** | 37 | 0 | 0 |
| **710** | 18 | 0 | 0 |
| **660** | 7 | 0 | 0 |
| **610** | 73 | 25 | 20 |
| **510** | 20 | 47 | 40 |
| **420** | 12 | 54 | 0 |
| **360** | 64 | 15 | 0 |
| **250** | 0 | 15 | 0 |
| **Estimated market size (ZAR pa)** | R26 581 651 | R31 934 636 | R7 207 770 |
| **Life/cyclone (yr)** | 1,5 | 0,3 | 2 |

The Cavex ceramic tiled DM cyclone design (CVXT) was based on the well-proven Cavex classification range of cyclones. The operating principle was to reduce the turbulence when feeding the cyclone in order to increase the residence time of the particles in the classification zone and, as a result, increase efficiency by reducing the misplaced particles. This required the design and fabrication of radius-moulded tiles to follow the exact profile of the Cavex feed chamber whilst using +92% Alumina tiled ceramics instead of the traditional natural rubber moulds. The design drawings were based on the existing Cavex classification cyclone which is well-proven in hard rock mining and coal classification across the world. Each radius tile in the feed chamber and cover of the cyclone was developed on a 3D model to maintain the exact profile of the rubber mould, comparing the radius of the rubber against the radius of the tiles after shrinkage during the firing process. A drawing was produced for every individual tile. The tile drawings were followed by individual mould drawings, a mould for every individual tile. Figure 4.1.1 illustrates the CVXT feed chamber after tiling is complete:

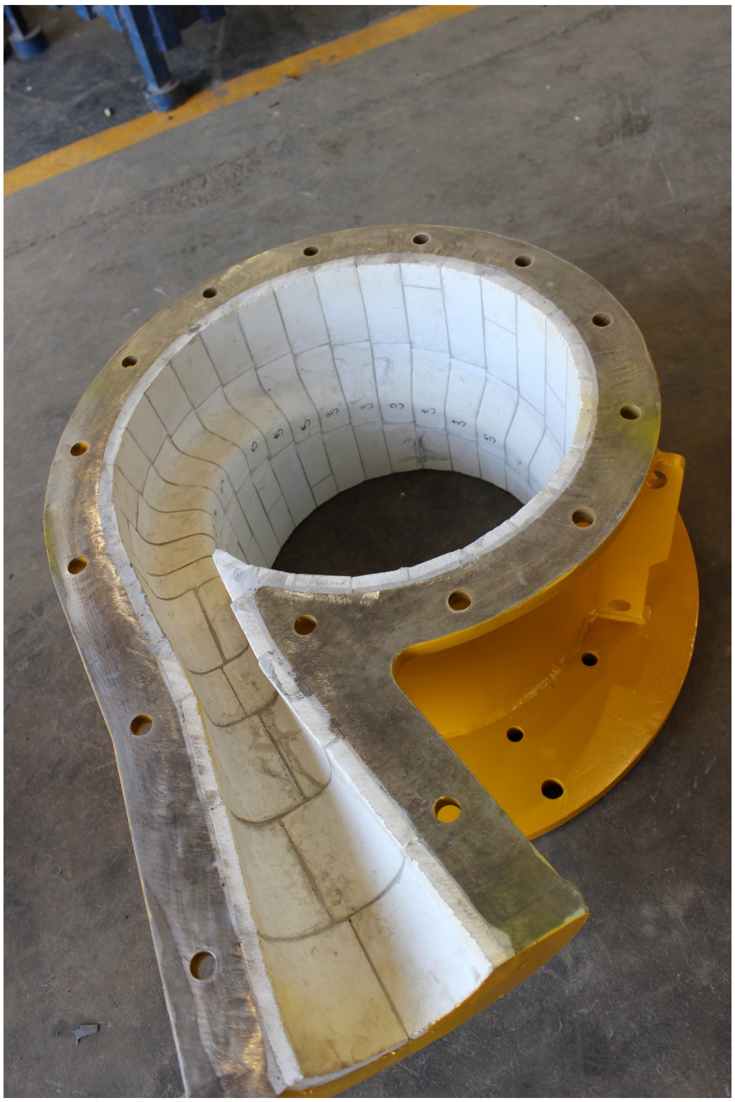


**Figure 4.1.1. Cavex inlet geometry – tiled (CVXT)**

A Fluorspar mine gave Weir Minerals the opportunity to install and optimise a 650CVXT DM cyclone. The 650CVXT DM cyclone had a cyclone diameter of 650mm. The 650CVXT feed chamber produced for the 650CVXT trial is presented in figure 4.1.2 and the cyclone in figure 4.1.3.

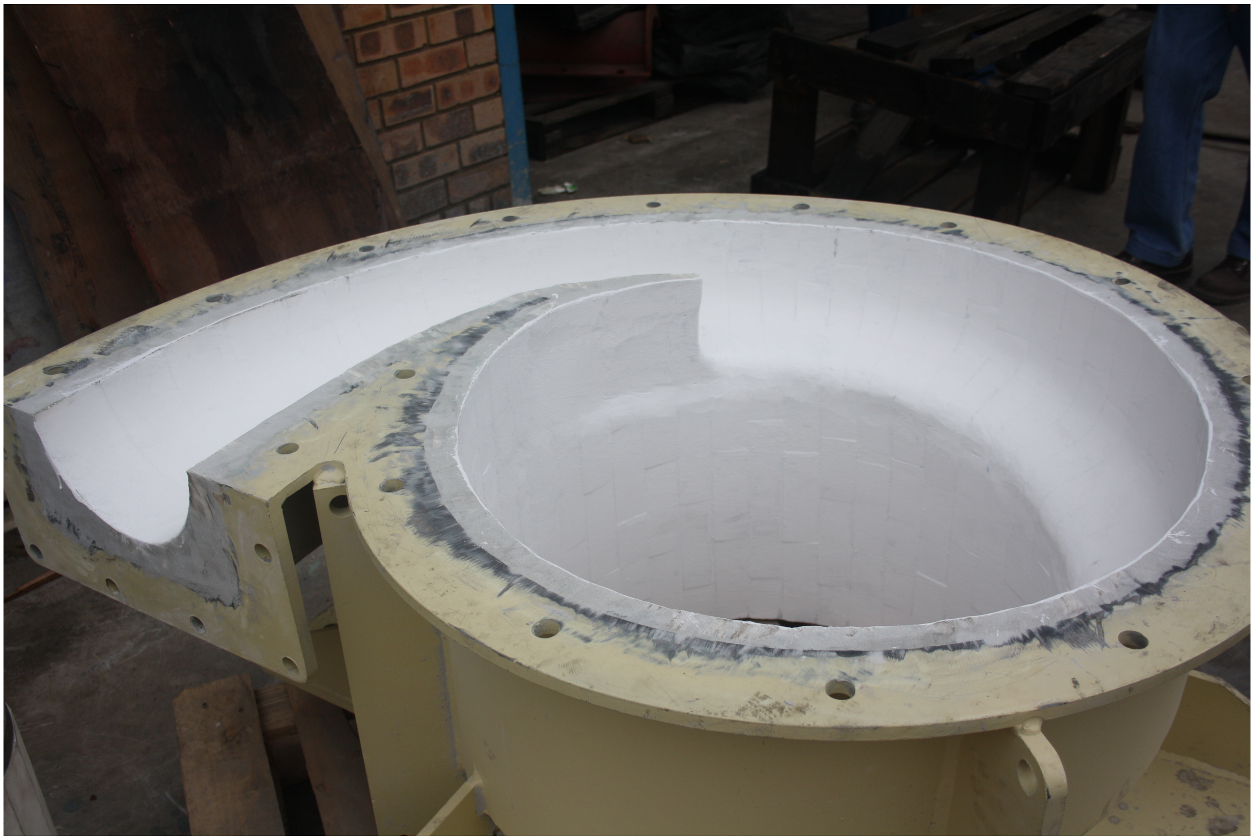


**Figure 4.1.2. 650CVXT test cyclone feed chamber (Corner-less Laminar spiral feed chamber – designed to reduce turbulence in feed chamber)**

**Figure 4.1.3. 650CVXT test cyclone prior to despatch**

The available DM cyclones in the market have engineered tiles installed perpendicularly towards each other as presented in figure 4.1.4.  This makes the CVXT design unique.

**Figure 4.1.4. Available DM cyclone feed chambers in the market (inclined to increase turbulence)**

## *4.2 METALLURGICAL TEST RESULTS*

Following the design of the 650CVXT DM cyclone, a mass balance was produced to match the flow of the new DMS circuit to the capacity of the cyclone and to determine the ideal vortex finder and spigot ratio and sizes to achieve optimum performance for the specific site conditions. The predicted mass splits were determined during the mass balance and cyclone selection process and this was used as the basis for the initial evaluation. Because the 650CVXT was supplied to a new plant expansion, all comparisons were made to the initial theoretical prediction and not to existing DM cyclones. The original design mass balance is presented in table 4.2.1.

**Table 4.2.1. Original design mass balance**

| MINERAL | Chromite | FEED | U/F | OVERFLOW |
|---|---|---|---|---|
| Ore SG | | 2.80 | 3.30 | 2.78 |
| Pulp SG | | 2.80 | 2.80 | 2.80 |
| | | | | |
| Solids Mass Flow | t/h | 150 | 5.88 | 144.12 |
| Solids Volume Flow | m³/h | 53.6 | 1.78 | 51.83 |
| | | | | |
| Liquid Mass Flow | t/h | 600.5 | 5.88 | 594.58 |
| Liquid Volume Flow | m³/h | 214.4 | 2.10 | 212.35 |
| | | | | |
| Slurry Mass Flow | t/h | 750.5 | 11.75 | 738.70 |
| Slurry Volume Flow | m³/h | 268.1 | 3.88 | 264.18 |
| | | | | |
| Solids % w/w | | 20.0 % | 50 % | 20% |
| Solids % v/v | | 20.0 % | 46% | 20% |
| M : O | | 4.0 | | |
| Slurry RD | t/m³ | 2.800 | 3.03 | 2.80 |

The cyclone selected to achieve the required duty was a 650CVXT with 211mm inlet, 200mm vortex finder and 190mm spigot. The expected operating pressure was 9D (157kPa). The Fluorspar recovery was originally predicted at 78% Fluorspar recovery. This cyclone was fabricated for installation. The above cyclone is presented in figure 4.2.1. The DMS feed was deslimed at 2mm and the +2 -20mm size fraction was fed to the 650CVXT.

The fluorspar operation consisted out of major amount of near gravity material. Near gravity material is defined as lots of minerals with solid S.G's close to the required cut point. In this operation a large quantity of minerals contains solid S.G',s closest to 2.8, which complicated the DMS process.

**Figure 4.2.1. 650CVXT installed at a Fluorspar DMS plant**

The cyclone was mechanically installed correctly using the provided 15° cradle. A pressure gauge on the cyclone's inlet feed chamber pipe was installed and functional during the operation of plant. The overflow and underflow discharged without forming any vacuums. During start up, the design throughputs were not achieved as a result of ore unavailability. This resulted in too low a pressure drop over the cyclone and not sufficient density difference between the overflow and underflow. The spigot was changed to 120mm to have a smaller Vortex finder: Spigot ratio.

Samples were taken by sampling entire overflow and underflow streams. The samples were dried, weight and pulverised for analysis. Every fifth sample was repeated to trace repeatability.

The dry run of the cyclone with dense media only (without ore) visually established 60:40 overflow to underflow volume flow percent split with 200mm vortex finder, 120mm discharge spigot and cyclone inlet pressure at ~110kPa. During the operation of the cyclone the dense media (magnetite suspension) relative density was taken with cyclone overflow measuring 2.2t/m³ whilst underflow read 2.25t/m³. Ore solids between 20mm and 2mm particle size at ~80t/hr mixed together with medium when fed into the cyclone resulted in practically 2 : 98 overflow to underflow ore solids percent split with ~2.3t/m³ medium density. A drastic split change was observed when spigot size was altered to ~60mm. An overflow to underflow ore solids split of 65:35 seen was mainly due to spigot overloading and resulted in pre-concentration from 20% fluorspar head grade to 30% in cyclone underflow according to onsite laboratory results of a concentrate heap sample being taken.

Further cyclone configuration revision by altering vortex finder and spigot size to 150mm and 80mm respectively proved not only inefficient but regressive. There was approximately 95% underflow mass pull at between 180kPa and 200kPa cyclone operating pressure, whilst slowing the pump speed for operating pressure at ~80kPa resulted in ~81% underflow mass yield and what seemed to be classification mainly by size and not density. Circulating medium density was recorded at 2.4t/m³. From approximately 2.3t/m³ and higher medium density a viscose and pseudo-plastic behaviour of the medium was noticed. The low or zero underflow and overflow medium densities differential during medium only cyclone runs was indicative of a viscosity problem. Medium specification and its influence on cyclone performance was not originally considered by the client (Ntuli, 2012).

It was recommended that the magnetite medium should be substituted either with FeSi 100D at medium density of 2.6t/m³ with solids mass flow at 120t/h. Further recommendations included minimum medium to ore ratios of 5:1 and maximum 8:1 for cyclone specification of 200mm vortex finder and 120mm spigot, optimally 130mm spigot. For cyclone specification using 150mm vortex finder the suitable spigot was expected to be 80-100mm with a minimum medium to ore ratio of 4:1 and a maximum medium to ore ratio of 6.5:1. Although maxima have been given, higher medium to ore ratios will not result in poor performance provided the density cut is kept well above 2.85 t/m³ with maximum operating pressure 240kPa (14D).

After further optimisation, the 650CVXT was fitted with a 150mm vortex finder and 80mm spigot feeding 60tph solids consistently because of upstream difficulties. The reduction in solids was made up with increased medium to ore ratios. The following results were 33 hours of consistent operation when feeding 60tph solids and a FeSi to ore ratio of 8:1.

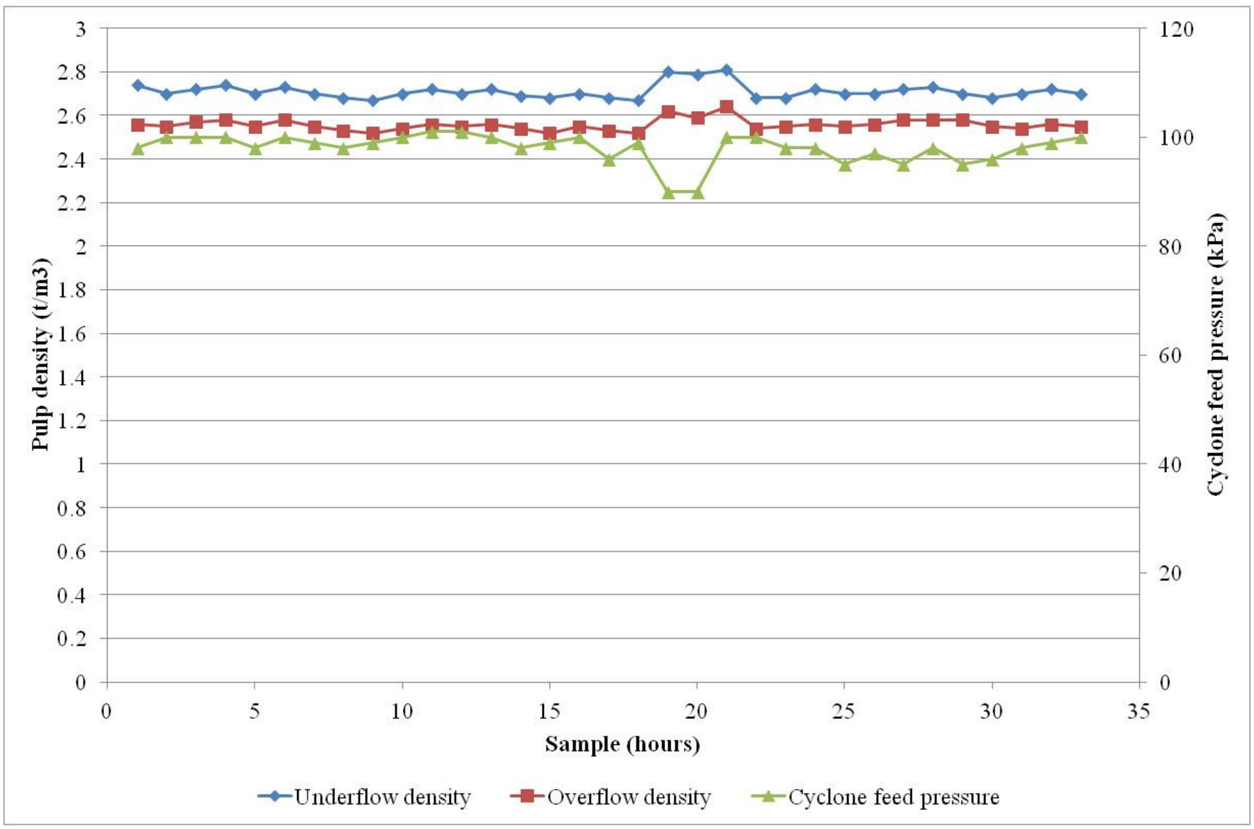



**Figure 4.2.2. Consistent 650CVXT results after plant was stabilised**

With consistent feed pressure, the overflow densities and underflow densities could be predicted accurately. The average fluorspar recovery was 80%, which compared favourably to the initially predicted 78% recovery in the design mass balance. The feed grade of 21.9% $CaF_2$ was upgraded to an average concentrate grade of 35.9% $CaF_2$. That was 64% improvement in grade to downstream processes which would enhance overall plant performance. Using the EPM formula (England et al, 2002)), which defines the densities corresponding to the 75% and 25% ordinates as shown in the partition curve (($d_{75}$ – $d_{25}$)/2), the EPM of the 650CVXT varied between 0.02 and 0.03.

# 5 CONCLUSION

DMS plays a significant role in the global coal and mineral industries. This R&D project established that almost all coal plants use DMS in order to produce saleable yields. The most common DMS separating vessel was a DM cyclone. Limited materials of construction exist for DM cyclones but with data available at present (after an extensive market survey consisting out of questionnaires), the +92% alumina tiled DM cyclone was found to be the most cost effective DM cyclone available for coal and mineral separation. DM cyclones were available in sizes from 250mm to 1500mm and can treat -80mm particles. The limitation of DM cyclone sizes is found to be based on upstream pumps and downstream screens rather than the DM cyclone being the limited equipment in the DMS process.

Because the essential role DM cyclones play in the mineral and coal industries, an alternative DM cyclone with a laminar spiral feed inlet (Cavex) was designed and developed to comply with customer requirements. This was done in order to ensure the sustainability of South Africa's coal and mineral industries and the economy as a whole. The Cavex ceramic tiled (CVXT) cyclone was manufactured, installed and commissioned. The basic principle of the design was to reduce the turbulence in the feed chamber in order to increase efficiency, whilst reducing wear.

Using the basic operating principles of hydrocyclones, the desired result of achieving 78% Fluorspar recovery was achieved The original 650CVXT DM cyclone was supplied to a plant extension, so comparisons were made to the original theoretical prediction and not to existing DM cyclones. Reducing the turbulence in the DM cyclone feed chamber, resulted in improved operation and consistent Fluorspar recovery. The 650CVXT results confirmed that a DM cyclone and a classification cyclone were essentially the same piece of equipment; it was the cyclone selection and operating criteria as well as the material of construction that differed. Over the years DM cyclones have been developed from the DSM principles but with modern CFD modelling, the boundaries and expectations have been moved for DM cyclones and the industry is slowly moving away from rigid DSM recommendations, provided the cyclone feed pressure and medium to ore ratio are controlled.

The results from this investigation proved that the latest classification technology can be implemented successfully in DMS. This was successfully illustrated by the commissioning and optimisation of the Cavex 650CVXT DM cyclone in a fluorspar operation. When the correct medium (FeSi) was introduced, vortex finder and spigot relationship corrected and the feed pressure to the cyclone was kept constant, the cyclone produced improved recoveries, even at lower pressures than the DSM prescribed 9D. The new developments and results from this research project indicated that lower feed pressures can have numerous benefits, as indications are that the feed pressure to the DM cyclone can be reduced if the turbulence in the feed chamber is reduced, resulting in increased residence times for classification. Over the life of the mine, the reduction of 60kPa (6m static head) on the cyclone feed pump could have significant reduction in energy consumption per cyclone feed pump. The evaluation of reduced feed pressures in the DM cyclone was outside the scope of this report.

The metallurgical results of the 650CVXT indicate that the 650CVXT DM cyclone consistently delivers more than 64% improvement in the $CaF_2$ concentrate grade at a feed pressure of 100kPa (6D) and fluorspar recovery of 80%. .

Reduction in the operating costs of DMS operations as a result of lower cyclone feed pressures whilst also improving recovery/yield will add much benefit to the coal and mineral industries and ensure a sustainable beneficiation environment. It is hoped that this will contribute to the sustainability of South Africa's coal and mineral industries and the economy as a whole.

# 6 RECOMMENDATIONS

The following recommendations are offered:

Further work should include the evaluation of the Cavex CVXT DM cyclone in near gravity coal circuits in order to establish whether the same lower operating pressures (6D) will result in improved coal yield, as occurred in the recovery of fluorspar, whilst achieving the desired ash contents. The work should include detailed studies on the effect of feed pressure vs. yield, when the turbulence in the feed chamber is reduced. Overall energy savings should be quantified as a result of reducing the DM cyclone feed pressure.

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# APPENDIX A: SAMPLE RESULTS

| | U/F Density | O/F Density | Feed Pressure | Sinks | Floats | Feed |
|---|---|---|---|---|---|---|
| | $t/m^3$ | $t/m^3$ | kPa | % $CaF_2$ | % $CaF_2$ | % $CaF_2$ |
| Hour 1 | 2.74 | 2.56 | 98 | | | |
| Hour 2 | 2.7 | 2.55 | 100 | | | |
| Hour 3 | 2.72 | 2.57 | 100 | | | |
| Hour 4 | 2.74 | 2.58 | 100 | 40 | 14 | 24 |
| Hour 5 | 2.7 | 2.55 | 98 | | | |
| Hour 6 | 2.73 | 2.58 | 100 | | | |
| Hour 7 | 2.7 | 2.55 | 99 | | | |
| Hour 8 | 2.68 | 2.53 | 98 | | | |
| Hour 9 | 2.67 | 2.52 | 99 | 39 | 9 | 21 |
| Hour 10 | 2.7 | 2.54 | 100 | | | |
| Hour 11 | 2.72 | 2.56 | 101 | | | |
| Hour 12 | 2.7 | 2.55 | 101 | | | |
| Hour 13 | 2.72 | 2.56 | 100 | | | |
| Hour 14 | 2.69 | 2.54 | 98 | 33 | 18 | 20 |
| Hour 15 | 2.68 | 2.52 | 99 | | | |
| Hour 16 | 2.7 | 2.55 | 100 | | | |
| Hour 17 | 2.68 | 2.53 | 96 | | | |
| Hour 18 | 2.67 | 2.52 | 99 | | | |
| Hour 19 | 2.8 | 2.62 | 90 | 35 | 14 | 24 |
| Hour 20 | 2.79 | 2.59 | 90 | | | |
| Hour 21 | 2.81 | 2.64 | 100 | | | |
| Hour 22 | 2.68 | 2.54 | 100 | | | |
| Hour 23 | 2.68 | 2.55 | 98 | | | |
| Hour 24 | 2.72 | 2.56 | 98 | 36 | 12 | 23 |
| Hour 25 | 2.7 | 2.55 | 95 | | | |
| Hour 26 | 2.7 | 2.56 | 97 | | | |
| Hour 27 | 2.72 | 2.58 | 95 | | | |
| Hour 28 | 2.73 | 2.58 | 98 | | | |
| Hour 29 | 2.7 | 2.58 | 95 | 35 | 14 | 20 |
| Hour 30 | 2.68 | 2.55 | 96 | | | |
| Hour 31 | 2.7 | 2.54 | 98 | | | |
| Hour 32 | 2.72 | 2.56 | 99 | | | |
| Hour 33 | 2.7 | 2.55 | 100 | 33 | 15 | 21 |
| **Average:** | **2.71** | **2.56** | **98.03** | **35.86** | **13.71** | **21.86** |