Modelling Dense Medium Cyclone Using CFD

Supervisor: doc. Ing. Vladimír Čablík, Ph.D.
ACKNOWLEDGEMENTS

I would like to express my gratitude to all those who helped me creating this thesis with their assistance, kindness and interest.

I would like to thank Professor Vladimír Čablík for undertaking promoting, care and assistance.

In particular I wish to express profound thanks to Professor Peter Fečko for personal involvement in carrying out the work and for his kindness, advice and assistance.

I wish to thank Professor Zbigniew Stęgowski for helping understanding and implementing Computational Fluid Dynamics models.
I declare that the work undertaken during period 2008–2013 was carried out by myself, with guidance from my supervisors and support from my colleagues. I declare that this is the only course for which I have been registered to study for the period from 01.09.2008 to 30.09.2013. In accordance with Section 47a of the Act (c) 111/1998 Sb., on universities and change and complement to the next law I agree publishing the text of this work on the Web page of the HGF VŠB-Technical University of Ostrava.

Ostrava, 15 of July 2013

MSc. Maciej Tora
Table of Content

Abstract ...................................................................................................................................... 8
Nomenclature ............................................................................................................................. 9
List of figures ........................................................................................................................... 11
List of tables ............................................................................................................................. 14
1. Introduction and scope of investigation ............................................................................ 15
2. Processing and classification ............................................................................................ 17
   2.1. The history of separators (the example of stream separators) ................................... 18
       2.1.1. Reichert cones ................................................................................................ 19
       2.1.2. Humphrey spiral concentrator ...................................................................... 20
       2.1.3. Concentrating tables .................................................................................... 21
       2.1.4. Bartles-Mozley separator ............................................................................ 22
   2.2. Various methods of separation ................................................................................... 23
       2.2.1. Gravity separation .......................................................................................... 23
       2.2.2. Magnetic separation ....................................................................................... 29
       2.2.3. Eddy current and electric separation .............................................................. 32
       2.2.4. Flotation ......................................................................................................... 33
   2.3. The application of hydrocyclones and dense medium cyclones ................................ 36
3. Mechanical coal processing .............................................................................................. 39
   3.1. The characteristics of coal processing in Poland ....................................................... 39
       3.1.1. Coal processing in Kompania Węglowa S.A. ................................................. 40
       3.1.2. Coal processing in Katowicki Holding Węglowy S.A....................................... 42
       3.1.3. Coal processing in Jastrzębska Spółka Węglowa S.A ..................................... 43
       3.1.4. Coal processing in Lubelski Węgiel “Bogdanka” S.A..................................... 45
   3.2. The technology of mechanical coal processing in the Hard Coal Mine
       Zofiówka (Kopalnia Węgla Kamiennego Zofiówka) .................................................. 47
       3.2.1. The final product of processing plants .............................................................. 48
       3.2.2. The storage in the heaps .................................................................................. 49
       3.2.3. The slime-water circuit ................................................................................... 49
9.2.2. Software used ......................................................................................... 136
9.2.3. Parameters of the model ........................................................................ 136
9.3. Discussion of results .................................................................................. 137
References .......................................................................................................... 143
Publications of the Author ................................................................................. 151
Appendix A. Scheme of Processing Plant in KWK Zofiówka ....................... 153
Appendix B. Sampling of mechanical processing plant in KWK Zofiówka ....... 154
Appendix C. Graphic presentation of CFD results ............................................. 181
Abstract

The aim of the investigation was to create mathematical model of heavy media cyclone. The foundation of the model was industrial scale sampling data obtained from Mechanical Coal Processing Plant in Hard Coal Mine “Zofiówka” in Poland. Over 10 thousand tons of coking coal per day is mined in this plant.

In this work mathematical model of cyclone working in real industrial plant is developed and presented. Model is based on a data obtained from sampling of working industrial scale cyclone.

The obtained results allow utilization of the model based on Computational Fluid Dynamics (CFD) simulations as the base for optimization of classification and beneficiation processes in dense medium cyclone. Moreover obtained results show models based on CFD simulations can be used at basic stage in industrial cyclones design practice, i.e. to choose and optimize geometry and working parameters of industrial scale cyclones. Results of sampling and established model of optimizing beneficiation in dense medium cyclones allowed proposing way to further optimize of technological setup of cyclone.

This work presents a method of simulation of solid particles selection in a cyclone. Computational Fluid Dynamics is the concept used. The FLUENT software has been used for the simulation. For water flow simulation, within the cyclone, the $k$–$\varepsilon$ turbulent flow model has been used. This allows calculating the velocity, pressure and turbulence field in the cyclone. The FLUENT software enables simulation of the solid particles flow within the flowing liquid.

Solid particles (coal and ash) of different sizes, flowing in this cyclone have been simulated. The simulation gives the entire profiles of flow values like: velocity, pressure, turbulence, etc. Several dozen graphical illustrations of obtained results are presented. Finally the particle size distribution curves, as a basic characteristic of the cyclone, were calculated.

Keywords: cyclone, dense medium cyclone, computational fluid dynamics, coal processing
Nomenclature

Cyclones’ parameters

A Area of inlet,
$A_C$ Cross-sectional area of the particle,
$a$ Percent combustible in the feed,
$a$ Pooled/mean gradient of the regression lines,
b Percent incombustible in the feed,
$D_p$ Diameter of a spherical particle (cm),
$d_{g/x}$ Mean separation between the regression lines,
$D_{inlet}$ Inlet diameter,
g Gravitational acceleration (cm/s$^2$),
$K_1$ and $K_2$ Constant,
$l_n$ lognormal,
n Total number of data points,
n$_A$ Total number of data points in stream 2,
n$_B$ Total number of data points in stream 1,
P Feed inlet pressure,
p$_f$ Slurry density,
p$_s$ Density of the solid,
p$_u$ Percent solid in the underflow,
$Q$ Slurry flow rate (tph),
r Radius of tangential motion,
$R_V$ Radial distance from the axis of symmetry in cylindrical coordinates (cm/s),
$R$ Correlation coefficient,
$S$ Spigot diameter,
$U$ Underflow solid tonnage,
v$_r$ Radial velocity of the fluid at a point in the cyclone,
u$_r$ Radial velocity of the particle,
$v_p$ Volume of the particles,
v$_t$ Tangential component of the particle,
u Percent combustible in the clean coal,
$V_F$ Vortex finder diameter,
$V$ Tangential velocity of the fluid (cm/s),
v Percent combustible in the rejects,
\( V_{(dy/x)} \) Variance of the separation between the regression lines, 
\( x_A \) Mean feed ash of dense medium cyclone from stream 2, 
\( x_B \) Mean feed ash of dense medium cyclone from stream 1, 
\( y_A \) Mean clean coal yield of dense medium cyclone from stream 2, 
\( y_B \) Mean clean coal yield of dense medium cyclone from stream 1.

**Turbulent flow model**

\( C_D \) Drag coefficient,  
\( d_p \) Particle diameter (m),  
\( F_D \) Drag force (N),  
\( g \) Gravitational force component (m/s),  
\( H(x) \) Heaviside function,  
\( p \) Pressure (Pa),  
\( Re \) Reynolds number,  
\( S_{ij} \) Strain rate (m^2/s),  
\( t \) Time (s),  
\( u_i \) Velocity component (m/s),  
\( \bar{u} \) Filtered velocity (m/s),  
\( \bar{u}' \) Subgrid-scale velocity (m/s),  
\( u_p \) Particle velocity (m/s),  
\( V \) Volume cell (m^3),  
\( x_i \) Dimensional component (m),  
\( \alpha \) Volume fraction,  
\( \mu \) Molecular viscosity (P),  
\( \mu_{eff} \) Effective viscosity (P),  
\( \mu_s \) Subgrid scale viscosity (P),  
\( \mu_t \) Turbulent viscosity (P),  
\( \rho \) Density (kg/m^3),  
\( \tau_{ij} \) Subgrid stress tensor (Pa),  
\( k \) Turbulent kinetic energy,  
\( \varepsilon \) Dissipation rate of the turbulent kinetic energy,  
\( \sigma_k \) and \( \sigma_\varepsilon \) The turbulent numbers for \( k \) and \( \varepsilon \) respectively.
List of figures

Fig. 2.1. Stream separator ........................................................................................................ 19
Fig. 2.2. Reichert cones ............................................................................................................ 20
Fig. 2.3. Forces effecting a particle in a spiral separator ......................................................... 21
Fig. 2.4. Concentrating table .................................................................................................... 22
Fig. 2.5. The principle feature of gravity separation in liquids ................................................ 24
Fig. 2.6. Classification of liquids applied for gravity separation in heavy liquids ................. 26
Fig. 2.7. Densimetric analysis of coal can be used to determine its upgradeability .............. 27
Fig. 2.8. Correlation curve showing ash content in different density fractions of coal as a function of their mean density ................................................................................... 28
Fig. 2.9. Types of magnetic separations classified according to particle size and way of separation .......................................................................................................................... 30
Fig. 2.10. Contact angle $\theta$ can be measured by the sessile drop methods ....................... 35
Fig. 3.1. Scheme of coal processing in KW processing plants ................................................. 41
Fig. 3.2. Simplified technological flowchart for beneficiation plant in JSW ....................... 44
Fig. 3.3. The technological flowchart of Mineral Processing Plant in LWB ...................... 46
Fig. 3.4. Simplified technological flowchart for Mechanical Processing Plant in KWK Zofiówka ......................................................................................................................... 51
Fig. 3.5. Cyclone diameter versus $D_{50,c}$ for Standard cyclone ............................................ 53
Fig. 3.6. Influence of feed concentration on separation ........................................................ 54
Fig. 3.7. Influence of pressure drop on separation .................................................................. 55
Fig. 4.1. Modes of separation in media ................................................................................... 58
Fig. 4.2. Selected parameters of a hydrocyclone .................................................................... 60
Fig. 4.3. Flowsheet used at Tertre and Winterslag .................................................................. 63
Fig. 4.4. Childress wash-to-zero heavy-medium circuit for 38x0 mm coal ......................... 65
Fig. 4.5. Fine coal heavy-medium cyclone circuit at Marrowbone, USA ......................... 66
Fig. 4.6. FRI heavy-medium pilot plant flowsheet .......................................................... 68
Fig. 4.7. Simplified flowsheet of the Greenside heavy-medium plant ......................... 69
Fig. 4.8. 150 mm cyclones at the Greenside heavy-medium plant ............................... 72
Fig. 5.1. Schematic diagram of a vorsyl separator and a dense medium cyclone .......... 80
Fig. 5.2. Schematic diagram of the ASH ................................................................. 81
Fig. 5.3. Schematic diagram of the main components of the conventional and three-
product hydrocyclones .............................................................................................. 84
Fig. 5.4. Schematic diagram of the three-product cyclone test rig set-up .................... 85
Fig. 6.1. Graphical representation of division of material into products (quantity
versus identity) ........................................................................................................... 87
Fig. 6.2. Schematic diagram showing spray, semi-robe and discharge at the
underflow of a hydrocyclone ................................................................................... 88
Fig. 6.3. Resistivity images showing the air core below the feed inlet of a 44 mm
diameter hydrocyclone ........................................................................................... 89
Fig. 6.4. The influence of feed solids' concentrations on the air core size .................. 90
Fig. 6.5. DM cyclone lay out in the plant ................................................................. 94
Fig. 6.6. Effect of feed inlet pressure on relative density differential ......................... 97
Fig. 6.7. The relationship between spigot diameter and spigot capacity for coal .......... 99
Fig. 6.8. The relationship between spigot diameter and spigot capacity for coal .......... 99
Fig. 6.9. The relationship between spigot diameter and spigot capacity for industrial
minerals and ores increasing feed pressure ............................................................... 100
Fig. 7.1. Experimental and simulated RTD functions ............................................... 107
Fig. 7.2. Experimental flow visualization .................................................................. 108
Fig. 7.3. Flow path-lines of tracers released at the jet inlet ....................................... 108
Fig. 7.4. Velocity vectors on three horizontal planes located 0.1, 0.25 and 0.4 m from
the bottom of the tank .............................................................................................. 109
Fig 7.5. Velocity profiles in the reactor at different levels on the plane perpendicular
to the flow visualization plane (along the diameter of the tank) ............................. 110
Fig. 7.6. Comparison between predicted (LES-ASM) medium densities (left) and
those measured by gamma ray tomography ............................................................ 117
Fig. 7.7. Comparison between contours predicted (LES-ASM) by CFD (left) and those measured by gamma ray tomography ................................................................. 117

Fig. 7.8. Comparison between contours predicted (LES-ASM) by CFD (left) and those measured by gamma ray tomography ................................................................. 118

Fig. 7.9. Comparison between contours predicted (LES-ASM) by CFD (left) and those measured by gamma ray tomography ................................................................. 118

Fig. 7.10. Distribution of different magnetite sizes inside the cyclone at 9D inlet pressure and RD at 1.243 feed density ................................................................. 119

Fig. 7.11. Comparison between contours predicted (LES-ASM) by CFD and those measured by gamma ray tomography ................................................................. 120

Fig. 7.12. Dimension for six new designs ................................................................. 122

Fig. 9.1. Pressure on a vertical plane inside cyclone ................................................ 138

Fig. 9.2. Volume fraction of Phase 4 (coal) in the mixture on 3 horizontal planes .... 139

Fig. 9.3. Volume fraction of Phase 15 (ash) in the mixture on one vertical plane ...... 139

Fig. 9.4. Vectors of velocity on three horizontal planes, colored by magnitude of velocity ................................................................. 140

Fig. 9.5. Value of velocity in vertical upwards direction only .................................... 140

Fig. 9.6. Value of velocity in vertical downwards direction only ............................... 141

Fig. 9.7. Values of turbulent energy on a vertical plane ......................................... 141

Fig. 9.8. Comparison of particle size distribution of sample and model ................. 142
List of tables

Table 2.1. Dynamic viscosity (Newtonian or plastic) of different liquids................................. 26
Table 3.1. Specification of coal parameters and properties produced in KWK Zofiówka........... 50
Table 3.2. Analysis of coal from KWK Zofiówka....................................................................... 50
Table 4.1. Results from the Tertre heavy-medium plant......................................................... 63
Table 4.2. Wash-to-zero plants owned by the Island Creek Coal Company, USA............... 64
Table 4.3. Improvement in magnetite recovery at the FRI pilot plant...................................... 74
Table 5.1. The output of hydrocyclones 350 i 100 mm in different grain classes................. 77
Table 5.2. Technological characteristics of the feed and product in hydrocyclones.............. 78
Table 6.1. The values of the exponent \( n \) obtained by various authors................................. 98
Table 7.1. The RTD results obtained by numerical and experimental approaches............... 107
Table 7.2. Flow densities predicted for 350 mm DSM cyclone compared to measured densities (kg/m\(^3\)) ............................................................................................................. 115
Table 9.1. Characteristics of samples of the feed for 33" cyclone........................................ 129
Table 9.2. Grain size analysis of the average sample of feed for 33" cyclone ....................... 130
Table 9.3. Characteristics of samples of the overflow for 33" cyclone ................................ 130
Table 9.4. Grain size analysis of the average sample of overflow for 33" cyclone ................ 130
Table 9.5. Characteristics of samples of the underflow for 33" cyclone............................... 131
Table 9.6. Grain size analysis of the average sample of underflow for 33" cyclone ............... 131
Table 9.7. Characteristics of samples of the feed for 26" cyclone........................................ 132
Table 9.8. Grain size analysis of the average sample of feed for 26" cyclone ....................... 132
Table 9.9. Characteristics of samples of the overflow for 26" cyclone ................................ 132
Table 9.10. Grain size analysis of the average sample of overflow for 26" cyclone ............... 133
Table 9.11. Characteristics of samples of the underflow for 26" cyclone ............................. 133
Table 9.12. Grain size analysis of the average sample of underflow for 26" cyclone .......... 133
1. Introduction and scope of investigation

Understanding and modelling the dynamic behaviour of particulate systems has been a major research focus worldwide for many years. Discrete particle simulation, for example computational fluid dynamics, plays an important role in this area. This technique can provide dynamic information, such as the trajectories of and forces acting on individual particles which is difficult to obtain by the conventional experimental techniques. Consequently, it has been increasingly used by various investigators for different particulate processes. In spite of a large bulk volume of scientific paper, little effort has been made to comprehensively review and summarize the progress made in the past. However, to a review of major works in this area has been recently completed. The studies have covered various subject groups, chiefly particle flow and particle–fluid flow.

Zhu et al. (2008) discussed the issue with emphasis on the microdynamics including packing/flow structure and particle–particle, particle–fluid and particle–wall interaction forces. It was then concluded that discrete particle simulation is an effective method for particle scale research of molecular matter. As we can read in Magway and Bosman (2008), the influence of the cyclone geometry and operating conditions on the spigot capacity of dense medium cyclones was investigated, and parameters of importance in this regard were identified. What is more, in their paper, Experimental investigation of the motion trajectory of solid particles inside the hydrocyclone by a Lagrange method, Wang et al. (2008) enumerated other methods of analysing the operations inside devices used in processing coal. Experimental investigations on the trajectory of solid particles inside the hydrocyclone were successfully carried out by using a high-speed motion analyser to track the particle movement. For each single particle, the trajectory was featured with stochastic characteristic. However, for the overall samples of particles, their motions hold the statistical property. The initial position of particles at of hydrocyclone entrance heavily affects the trajectory of the particles inside a cyclone and consequently the separation performance. The results in the mentioned study are valuable for understanding the stochastic and statistical behaviours of particle motion in the separation process inside hydrocyclones, and provide some valuable information for finding some effective way to improve the separation performance in the hydrocyclone.

The goal of this work is to show the application of CFD modelling in the study of the operations of dense medium cyclones in coal processing.
Dense medium cyclones separate particles primarily according to their differences in density, and are used in the beneficiation of coal, iron ore, diamonds and others. The cyclone can, however, be constrained by the ore carrying capacity of the spigot, especially for ores in which a significantly large proportion of the feed particles need exit through the sinks stream. Currently, the spigot capacities used in the sizing and selection process for dense medium cyclones are based mainly on those capacities provided by the original developers of the dense medium cyclone. Further, it is not clear which parameters, other than the spigot diameter, have an influence on the spigot capacity of dense medium cyclones. We hope that these and other questions can be answered with the aid of CFD modelling.

**Scope of the investigation**

The aim of this work was to establish and to verify in the industrial scale mathematical model of coal classification in dense medium cyclone. The model is build up for real cyclone installed in beneficiation plant. Empirical data was collected in Coal Beneficiation Plant in Hard Coal Mine “Zofiówka” during factor experiment. Cyclones are broadly used devices (e.g. in copper and coal processing for classification of output of mills, thickening products of beneficiation). Cyclones can be easily controlled (e.g. cut size grain can be altered by change of feed pressure).

The base of the model (input parameters) is data collected in working industrial system (hard coal mine). The concept of the model is based on Computational Fluid Dynamics (CFD). CFD is a very good tool for modelling, as it is very flexible. One is able to model any shape of device and also one is able to use different equations describing physical processes.

Differential equations (e.g. conservation of momentum, continuity of flow, transport equations for the energy) are used in the model. The k-ε flow model is used for modelling turbulences.

The results obtained in this work are base for optimization of the real industrial process.
2. Processing and classification

Coal is an organic rock originating mainly from plants which lived millions of years ago. From the chemical point of view each sort of coal is a mix of a fine organic fuel with a mineral substance. The sort of coal is obtained basically from the content of the so called ballast in the coal – that is the content of water and ash and the main distinguishing feature of which is the rate of carbon content. The amplitude of carbon content is based on the content of the C element in the ashless and water-free coal substance. This basically means that the more C element it contains the higher is the rate of carbon content.

The production of coal in the world in year 2006 amounted c. 5.37 billion tones where the hard coal amount was about 4.6 billion tones. After the year 2000 there was an increase in coal exploiting (c. 9% per year). The production of coking coal was increasing comparably slower – from 509 million tonnes in 1985 to c. 716 million tonnes in 2006.

The main producers of hard coal in the world are: China, the USA, India, South Africa, Australia, Russia, Indonesia, Poland, Kazakhstan and Colombia. The largest part of the world production of coal (c. 84%) is used in the countries where the coal is mined. The countries of the largest producers and exporters of hard coal are also the countries which are most rich in coal – over 87% of the world resources of coal are located in their territories (the hard coal and the lignite). These resources are estimated to be 909 billion tones.

According to the commercial 2012 BP Statistical Energy Survey, Europe & Eurasia had end 2011 coal reserves of 304 604 million tonnes, equivalent to 242 years of current production and 35.38% of the world total. The world's largest reserves are held by the USA, Russia, China, Australia and India. Russia is Europe's biggest consumer of coal followed by Germany, Poland and Ukraine. In Poland over 85% of coal is used in power plants whereas the rest serves, among others, the production of coke. The two main centres of coal mining are Upper Silesia Coal Basin and Lublin Coal Basin – among which the latter is a developing one. Polish coal is mainly sold to such industrial branches as: power plants, coke manufacturing, industrial and communal energy production, paper and mineral industry and individual clients (Nycz 2000: 3–4).

---

It is estimated that coal will maintain to be the basic source of energy due to the low cost of the energy's production (compared to other sources) as well as for the fact that its huge resources are still present on all continents.

One of the main technical and economic issues concerning coal is its processing. In this process specific grades of coal are being produced. These possess specific technical and technological values – important for the coal’s further use. The exploited coal undergoes a number of processes in coal processing plants².

The processing plant is a complex of facilities, machines and buildings where the process takes place. Throughout the production the raw material undergoes a variety of processes the aim of which is to prepare it for further commercial use. A modern processing plant is equipped with automatic machinery which controls the converting process that lead to obtaining the product. Mechanical coal processing is one of the basic branches of technology and techniques of mining. In case of multi-mineral raw material the separation of each mineral is also necessary. The mechanical processing is used for preparing the raw material according to its industrial use which takes into account such issues as the needed particle size and the proper particle classes. Depending on the sort and the final consumer of the material, different processing technologies are used.

In this chapter we would like to focus on the general description of processing and classification which are performed with the aid of various methods. We would also like to provide a description of the devices used for processing and classification (stream separators, Reichert cones, spiral concentrators, concentration tables, Bartles-Mozley separator) and describe the physical processes which take place in them (such as gravity separation, electric separation or flotation) in order to provide a background and a context for the processes applied in hydrocyclones which are of most concern to us³.

2.1. The history of separators (the example of stream separators)

The description of stream separators dates back to the 16th century. They were depicted by Agricola (1494–1555) in one of his illustrations dealing with mining and mineral processing. They are estimated to be simple separators. In Figure 2.1 we present a general scheme of the stream separator and the mode of its operation.

---
² Pol. Zakłady Mechanicznej Przeróbki Węgla.
³ We shall not concentrate on flocculation and oil agglomeration for the reason of these processes being more chemical in nature than physical and having no relation with the cyclones’ usage background.
As one can learn from the literature, these separators do not possess movable parts. Their performance is based on falling of particles and their friction against the walls of the separator in the course of particle moving together with the liquid. For very thin streams it is difficult to give a precise description of particles separation. As it is described in Drzymała (2007: 183) ‘when the stream is thick, approximate description of particles separation can be performed on the basis of the relations valid for hydraulic separation in horizontal stream of water with inclination taken into account. For proper operation of the stream separator, it is important to ensure, for a particular feed, appropriate speed of the liquid flow and position of openings through which the concentrate and intermediate products are collected’.

2.1.1. Reichert cones

Another example of a simple separator is a Reichert cone. Its advantage is also the lack of movable parts. They are also stable and are less prone to break or overuse. Although separation in Reichert cones is similar to that in stream separators, the design of the Reichert cones is different (Fig. 2.2).
The feed, which contains water and particles, flows on the surface of flattened cones with a diameter of about 2 meters. Obviously, heavy particles run downwards and concentrate in the lower part of the stream. The light particles, together with water, are transported upwards in the device while the heavier run towards the opening at the bottom. In Reichert cones heavier particles are later subject to a multiple separation on the subsequent cones which results in obtaining diversified products. As explained by Drzymała (2007), ‘one separating device contains from several to a dozen of stacked cones and its height can be up to 6 m. The Reichert cones are very efficient and can process about 100 Mg of the feed per hour. They can be used for all kinds of ores and raw materials’ (ibid.: 184).

2.1.2. Humphrey spiral concentrator

Drzymała (2007: 184) explains: ‘spiral separators operate similarly to stream separators, but additionally, a centrifugal force is present in the system. The centrifugal force is a result of the shape of the separator. The material travels down through a stationary spiral with many turns of mean radius (20 cm) with a fall per turn of about (28 cm)’.

Figure 2.3 presents the thin stream separation that effect particles in the spiral separator. The forces responsible for separation involve: centrifugal \( (F_c) \), gravity \( (F_g) \), friction \( (F_f) \), and liquid pressure \( (F_n) \) forces but the forces which really determine the separation are \( F_f \) (friction) and \( F_n \) (pressure) and they operate in the same direction as horizontal component of the gravity forces \( F_c(x) \), as well as the horizontal component of the centrifugal force.

The excess force \( (F_{ex}) \) pushes the particles towards the side of ‘light’ and low friction particles or towards heavy and high friction particles. Its work results from the force balance.
The particles are stratified in the stream until they reach a proper place in the fan and the excess force \( (F_w) \) disappears. In the Humphrey spiral separator the tailing containing light particles is collected in the lowest coil. The concentrate is collected from the neighbouring 2–3 coils through the holes in coil surface bottom. The particles larger than about 0.05 mm in diameter are the ones suitable for separation (comp. Drzymała, 2007: 184–186).

### 2.1.3. Concentrating tables

When speaking about the usage of concentrating tables one should note that they can be applied to process nonferrous metals ores which contain particles smaller than about 3 mm. In the case of coal, because of its considerably lower density, larger particles (up to 6 mm in diameter) could be processed. Additional factor improving separation is the use of asymmetric shaking of the concentrating table (Drzymała, 2007: 187). Concentrating tables spread particles on their surface – as shown in Figure 2.4.

![Fig. 2.3. Forces effecting a particle in a spiral separator (Drzymała, 2007)](image-url)
The work of the gravity force and friction forces yields in formation of stratified particles on the concentrating table. The ordering force acting on particles is the sum of vectors representing the forces. One has to note that from the physical point of view the stratification force for heavy particles has a different direction than that for light particles as it is mainly determined by the inertia force, while other forces (friction, gravity and water pressure force) are of low values – as noted by Laskowi and Łukaszewicz (1989) for light particles it is otherwise.

In this device it is recommended to use smooth surfaces of the concentrating table for fine particles of about 0.1 mm in size. In other cases the surface of the concentrating table is modified by grooves and strips.

2.1.4. Bartles-Mozley separator

There are also other devices for separation in thin stream of water. One of these is the Bartles-Mozley table (Burt and Ottley, 1974).

In the Bartles-Mozley separator it is possible to process materials consisting of particles from 5 to 100 micrometres. This separator operates periodically which means that in the first cycle the feed is transported through the separator, consisting of 40 plates of
1.1×1.4 m in size and inclined at the angle of 3°–4°. The first stage lasts for about 36 minutes and heavy particles sediment on the plate surfaces, while light particles, together with water, flow out of the separator. In the second stage, the supply of feed is stopped and the separator is inclined at the angle of 45° to the level while the water is used to remove heavy particles from the separator. When the particles settle, the whole separator vibrates. Authors note that the separator output should be about 2.5–4.0 Mg per minute (comp. Drzymała, 2007).

2.2. Various methods of separation

As we have noted various separators’ work is based on the processes that take place within them. The following division is based mostly on the physical phenomena such as interaction between particles and forces the devices take advantage from.

2.2.1. Gravity separation

During gravity separation the particles which are denser than the liquid (in which particles are suspended) sink – the particles of lower density float. When gravity separation is carried out slowly, separation is effected only by the gravity ($F_c$) and buoyancy forces ($F_w$) and the resulting ordering force is:

$$F = F_c - F_w = m_p \cdot g - m_c \cdot g = \rho_p \cdot g - \rho_c \cdot g = \rho_p - \rho_c \cdot g$$

(2.1)

where:

$m_p$ – particle mass,

$\rho_p$ – particle density,

$m_c$ – mass of liquid displaced by particle,

$\rho_c$ – liquid density,

$g$ – gravity constant,

$v$ – particle volume.

Thus, the main material feature of static gravity separation is $\rho' = (\rho_p - \rho_c)$ that is the density of material in the medium in which separation takes place. When density separation takes place as a dynamic process, the result of separation is effected by additional parameters, mostly the friction force $F_f$, which depends on particle diameter ($d$), velocity of moving particle ($v$), and liquid viscosity ($\eta$) (Fig. 2.5):
Fig. 2.5. The principle feature of gravity separation in liquids is:
a) difference between densities of particle and liquid ($\rho_p - \rho_c$) (static process), while in dynamic processes is: b) difference between densities of particle and liquid as well as the time of separation (Drzymała, 2007)

The viscosity results from friction occurring between moving molecules of the liquid as well as moving particles. When viscosity does not depend on the movement of particles and molecules and is constant at particular temperature, the liquid or suspension are called Newtonian. However, viscosity of suspensions containing mineral particles is usually not constant. Their viscosity depends on particles content in the liquid and such suspensions are termed non-Newtonian. The viscosity of a non-Newtonian suspensions is described by general equation (Kelly and Spottiswood, 1982):

$$\tau_s = \eta_{app} \cdot q$$  \hspace{1cm} (2.2)

where:

- $\tau_s$ – shear stress,
- $q$ – shear rate,
- $\eta_{app}$ – apparent viscosity (dependent on shear stress).

For many non-Newtonian liquids the relation between the shear rate and shear stress can be described by an approximate equation (Laskowski, 1969):
Thus, the equation describes a system in which at low shear stress there is no movement of liquid (or the liquid moves very slowly). At a certain shear stress ($\tau_d$) the liquid starts to flow. Taking into account the friction force ($F_T$) in the balance of forces acting on a particle leads to the following relation:

$$ F = F_c - F_w - F_T $$ (2.4)

The last equation indicates that at a high resistance force a rapid separation of particles cannot take place as the difference in liquid and particle densities is reduced by particle friction against molecules of the suspending liquid (Drzymała 2007). The gravity separation is usually performed in water. The density of water is 0.99823 g/cm$^3$ at 293 K (20°C) (CRC, 1986). However, except for ice and some polymers having densities lower than 1 g/cm$^3$, solid materials are denser than water. Therefore, liquids denser than water (heavy liquids), have to be used for gravity separation.

The heavy liquids can be homogeneous or heterogeneous (containing two or more phases and forming suspensions). The homogeneous inorganic heavy liquids usually consist of water and water soluble salts$^4$. Homogeneous organic liquids are seldom applied for gravity separation, though, as noted by Drzymała (2007), many technologies and patents are available for this purpose. Other liquids can also be used, but special attention should be paid to their potential toxicity, reactivity with particles, and harmful ability to penetrate pores of the particles. The following figure (Fig. 2.6) presents how the separation process in the devices looks like:

---

$^4$ The homogeneous liquids can be divided into organic and inorganic liquids.
Fig. 2.6. Classification of liquids applied for gravity separation in heavy liquids.

* Mixture of thallium formate and thallium malonate (COOTl)₂CH₂ + HCOOTl, ** contains HgI₂·2KI·H₂O.

As we learn, homogeneous inorganic liquids like salts solutions (especially ZnCl₂ and CaCl₂) dissolved in water are much more advantageous for application than organic liquids. They also are seldom used by industry. In practical applications heavy liquids in the form of suspension are most often applied. Heavy suspension liquids are used for the cleaning of coal and other materials, including ores and raw materials (Laskowski et al., 1979; Stepinski, 1964). Heavy suspension liquids featuring high dynamic shear stress (τd) and low values of plastic viscosity (ηplas) are best suited for separation because they are more stable when the τd is high. In flotation (which we will describe later on) to improve the stability of heavy suspension the same dispersing agents are used. These include: polyphosphates, short chain polyelectrolytes, modified carbohydrates, and glass water. Typical values of viscosity of heavy liquids – following Laskowski et al. (1979), Laskowski and Łuszczkiewicz (1984) and Drzymała (2007) – are shown in table 2.1.

The densimetric analysis allows determination the content of different fractions in the feed and separation products. The results of densimetric analysis are tabled and usually plotted as the content of each particular density fraction as a function of either average, lower or upper boundary of fraction density. The densimetric analysis of the feed and separation products is a good base for the analysis of separation’s effectiveness. The densimetric analysis is usually performed for coal to establish its efficiency (comp. Drzymała 2007: 195). Figure 2.7 shows a way of conducting densimetric analysis:

---

5 The substances which are used in suspension are also called the dead weights. Dead weights are for example magnetite, ferrosilicon, quartz, barite, clay minerals.
Table 2.1. Dynamic viscosity (Newtonian or plastic) of different liquids (Drzymała, 2007)

<table>
<thead>
<tr>
<th>Liquid</th>
<th>Viscosity</th>
<th>Density</th>
<th>Remarks</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>cP [10^{-3} N·s/m²]</td>
<td>g/cm³</td>
<td></td>
</tr>
<tr>
<td>Water</td>
<td>1.002</td>
<td>0.99823</td>
<td>293 K (20°C)</td>
</tr>
<tr>
<td>Bromoform</td>
<td>1.89</td>
<td>2.89</td>
<td>298 K (25°C)</td>
</tr>
<tr>
<td>Methylene bromide</td>
<td>1.09</td>
<td>2.48</td>
<td>288 K (15°C)</td>
</tr>
<tr>
<td>CaCl₂ in water</td>
<td>5%</td>
<td>1.21</td>
<td>1.043</td>
</tr>
<tr>
<td></td>
<td>35%</td>
<td>4.88</td>
<td>1.33</td>
</tr>
<tr>
<td></td>
<td>51%</td>
<td>33.20</td>
<td>1.52</td>
</tr>
<tr>
<td>ZnCl₂ in water</td>
<td>20%</td>
<td>1.94, 298 K (25°C)</td>
<td>1.18, 293 K (20°C)</td>
</tr>
<tr>
<td></td>
<td>50%</td>
<td>4.04, 298 K (25°C)</td>
<td>1.55, 293 K (20°C)</td>
</tr>
<tr>
<td></td>
<td>75%</td>
<td>1.37, 298 K (25°C)</td>
<td>2.07, 293 K (20°C)</td>
</tr>
<tr>
<td>Fe₃O₄ in water</td>
<td>100</td>
<td>2.3</td>
<td>0.017 mm particles</td>
</tr>
<tr>
<td></td>
<td>45</td>
<td>2.3</td>
<td>0.026 mm particles</td>
</tr>
<tr>
<td></td>
<td>45</td>
<td>2.6</td>
<td>0.038 mm particles</td>
</tr>
<tr>
<td></td>
<td>30</td>
<td>2.6</td>
<td>0.051 mm particles</td>
</tr>
</tbody>
</table>

Fig. 2.7. Densimetric analysis of coal can be used to determine its upgradeability. It provides content λᵢ of different density fractions in the sample. The density of heavy liquid (g/cm³) is given as a number on the container. After ash content (λ) determination in all density fractions it is possible to determine the upgradeability of the sample. Then, the content of each fraction (λᵢ) has to be treated as yield (γ) of potential products of ideal separation (Drzymała, 2007)

In order to conduct densimetric analysis, heavy liquids of increasing density have to be prepared. Drzymała (2007) provides the reader with the following data:

- for coal they can be solutions of ZnCl₂ of density ranging from 1.3 to 1.8 g/cm³ as coal density is from 1.17 to 1.35,
- for ash forming minerals from 1.8 (shale) to 5.2 (pyrite) g/cm.

On the basis of the densimetric analysis one can draw the relationship between the content of different density fractions in the sample and their densities in the form of the
frequency or cumulated distribution curves. These curves characterize one product of separation. When either the other separation product or the feed are the subject to densimetric analysis, one can draw pairs of frequency, distribution, and other separation curves.

As it has already been mentioned the fractions obtained by the densimetric analysis of feed can be subjected to chemical analysis for the content of selected component. In the case of coal this can be ash. Then, the densimetric analysis becomes the method of ideal separation and the density fractions are products of separation. The results can be a good base for the content analysis. The obtained curves can be the reference for separation results obtained by other methods or in different devices.

On the basis of the results of the two mentioned analyses of the examined material density fractions, it is possible to produce other relations, including correlation curves combining the content of a component (ash) in different density fraction and density of the fraction. The mentioned type of the curve, for coal, is shown underneath on Figure 2.8:

![Fig. 2.8. Correlation curve showing ash content in different density fractions of coal as a function of their mean density. Based on data from figure 2.7. (Drzymała, 2007)](image)

The gravity separation of particles can also be done for separation of materials of high density. The magnetic liquids for instance change their apparent density depending on the intensity of the magnetic field. Their apparent density can be changed from about 1.3 (no magnetic field) to about 20 g/cm$^3$ at high intensity of the magnetic field. The magnetic liquid usually contains nanoparticles (~10 nm) of magnetite which are suspended in such liquids as water, oil, esters, naphtha which also contain surfactants which provide stability of the magnetic suspension. Fine magnetic particles are produced by precipitation from aqueous solutions containing Fe(II) and Fe(III) salts (Odenbach 1998). Separation in magnetic liquids
can be applied for nonmagnetic material of significantly different densities, especially non-ferrous metals and plastics from car scraps (Drzymała 2007: 196–197).

### 2.2.2. Magnetic separation

A particle placed in a magnetic field interacts with this field. As a result, the particle moves in the field. This phenomenon, based on the material’s magnetic susceptibility, is used in separation of particles of different materials. The phenomenon is termed magnetic separation.

From a practical point of view substances are divided into magnetic and non-magnetic materials. Non-magnetic particles are characterized by zero or negative value of magnetic susceptibility while magnetic particles by positive values taking into account actual magnetic properties resulting from impurities which can considerably alter magnetic susceptibility of particles (comp. Rayner and Naper-Munn 2003: 158). SnO\(_2\) or (Ca, Mg)CO\(_3\) can serve as an example, which as pure chemical substances are non-magnetic, while their mineral equivalents, cassiterite and dolomite, always show some positive magnetism. Magnetic impurities can cause that magnetic susceptibility of contaminated material depends on the intensity of magnetic field. It should be added that the same minerals, originating from different sources, can feature different magnetic susceptibility. This fact influences the choice of the device, because different magnetic susceptibilities requires different mechanical applications. LIMS separators, for example, are applied for strongly magnetic particles. For this reason we would like to present the scheme of magnetic separation classified according to particle size and the way of separation on Figure 2.9.

Magnetic separation can be also conducted in the devices featuring considerable gradient of the magnetic field. This kind of separation is applied both for fine and coarse particles. The latter separation is usually conducted in water. Variation of the magnetic field can be obtained due to filling the spaces between magnet or electromagnet poles with magnetic material\(^6\). Materials having negative values of magnetic susceptibility are called diamagnetics, while the ones with positive magnetic susceptibility are called paramagnetics. Thus, susceptibility characterizes magnetic properties of substances and is the main material parameter of magnetic separation. Magnetic separation is possible for particles of different signs or susceptibilities.

---

\(^6\) The filling can be metal wool or metal inserts shaped in a special way, which guarantees a high dH/dx value, i.e. significant curvature of magnetic field lines.
Fig. 2.9. Types of magnetic separations classified according to particle size and way of separation: drum MS – separation in a field of low intensity of magnetic field (or shortly LIMS), HIMS – separation in high intensity magnetic field, e.g. IMR (induced magnetic roll), HGMS – separation in high gradient magnetic field, OGMS – separation in an open gradient field in superconducting devices, IDS – separation in isodynamic magnetic field (Drzymała, 2007)

In these separators the process of separation is performed in a cyclic mode. In the first stage, the feed is passing through a separator while the magnetic particles are arrested by the filling of the separator. In the second stage the magnetic field is turned off or the filling removed and then, magnetic particles are washed out by water. In order to obtain a continuous process, carousel type devices are applied, whose special mechanisms turn batch processes into a cyclic process. The separator possesses many sections which gather magnetic particles in one cycle, i.e. provide non-magnetic product. In the second cycle, they provide magnetic particles. Both types of sections work simultaneously. Therefore, the whole separator works as a continuous device.
When the magnetic field of intensity $H$ (A/m) is generated in vacuum, induced magnetic field is $B_0$ (Vs/m$^2$ = T):

$$B_0 = \mu_0 \cdot H$$

(2.5)

where:
- $\mu_0$ – magnetic permeability of vacuum, $\mu_0 = 4\pi \times 10^{-7}$ V·s/(A·m) = $4\pi \times 10^{-7}$ H/m
- $H$ – henry$^7$,
- A – ampere,
- V – volt,
- s – second,
- m – meter,
- Wb – weber,
- T – tesla.

$$T = \frac{V \cdot s}{m^2} = \frac{Wb}{m^2} = \frac{H \cdot A}{m^2}$$

(2.6)

Drzymała (2007) summarises: ‘when a particle is placed in vacuum in the same magnetic field $H$, generated magnetic induction will be different from that of vacuum due to a different magnetic permeability of the particle $\mu$’. This gives $B = \mu \cdot H$, where $\mu$ is particle magnetic permeability which, like magnetic permeability in vacuum $\mu_0$, is expressed as henry per 1 meter’ (ibid.: 237). Generated magnetic induction (B) depends both on magnetic field $H$ and on the substance magnetization $M$. Thus:

$$B = \mu_0 \cdot (H + M) = B_0 + \mu_0 \cdot M$$

(2.7)

where $M$ is magnetization expressed, like magnetic field $H$, in A/m.

As the vector of magnetic induction $B$, in the presence of a particle in a magnetic field, is different from the vector of induction in vacuum, i.e., in the original field, the increment of magnetic induction over that in vacuum can be expressed as:

$$\chi = \frac{B - B_0}{B_0} = \frac{\mu - \mu_0}{\mu_0} = \frac{\mu_0 \cdot M}{B_0}$$

(2.8)

where $\chi$ is volumetric magnetic susceptibility, which is a dimensionless quantity, either positive or negative. When $\chi$ is negative, induced magnetic polarization has an opposite sign to the applied field $H^8$.

---

$^7$ it should not be mistaken with symbol $H$ which denotes intensity of the magnetic field (in A/m)
There are three kinds of magnetic susceptibility: volumetric, mass (also called specific), and molar susceptibility. However, they can be expressed in different units (SI or obsolete electromagnetic CGS system). Transformation of dimensionless volumetric magnetic susceptibility \( \chi \) expressed in CGS system into a SI unit can be done using the relation:

\[
\chi = \chi^{SI} = 4\pi \cdot \chi^{CGS}
\]  

(2.9)

The specific susceptibility (\( \chi_w \)) in both systems of units can be obtained by taking into account material density, i.e. calculating it from the relation:

\[
\chi_w = \frac{\chi}{\rho}
\]  

(2.10)

where:

- \( \chi \) – volume magnetic susceptibility (dimensionless),
- \( \chi_w \) – specific susceptibility, cm\(^3\)/g,
- \( \rho \) – material density, g/cm\(^3\).

Magnetic properties of substances result from magnetic properties of their chemical elements. Thus, magnetic properties of elements depend on their structure, especially of the outer electrons, and are characterized by magnetic moment of an atom. The magnetic moment is resultant of magnetic moments of valence electrons, other electrons present in the atom, and, to a small extent, the nucleus. The resultant magnetic moment of diamagnetics is negative while that of paramagnetics is positive. They can be recognized by placing them in a non-uniform magnetic field. Paramagnetic substances are attracted towards more dense lines of magnetic field forces, while diamagnetic ones are pushed out of the field. It results from the appearance of force \( F \) which acts on a particle in magnetic field \( H \). This phenomenon is described in more detail in Hopstock (1985) and Svoboda (1987).

When diamagnetic substance contains paramagnetic impurities, its magnetic susceptibility can be positive and it can depend on the temperature and the intensity of magnetic field. Dolomite or cassiterite can serve as an example (Svoboda, 1985).

### 2.2.3. Eddy current and electric separation

As one can read in the literature the idea of electric separation is the interaction of electrically charged particles with the electric field. During the separation the so called

---

8 In his profound work Drzymała adds that ‘the values of magnetic susceptibility for different substances can be found in many handbooks e.g. CRS Handbook of Chemistry and Physics (CRS, 1986/87; Gupta, 1986/87, Smith, 1986/87) or in the monograph by Svoboda (1987). While reviewing literature one should pay attention to the units system of magnetic susceptibility.’ (Drzymała 2007: 237)
Lorentz force is used. Drzymała (2007) enumerates the following data for electric conductivity (in $\Omega^{-1}\cdot m^{-1}$) at 293 K are: aluminium $3.77\cdot10^7$, coppers $5.96\cdot10^7$, silica glass (Clear, at 623 K) from $2.5\cdot10^{-4}$ to $3.3\cdot10^{-9}$ quartz ~0, plastic ~0. It results that the Lorentz force does not depend on the magnetic properties of particles because the eddy currents occur both in paramagnetic (aluminium) and diamagnetic (copper) conductors. As Drzymała (2007) states himself, ‘the eddy currents separator usually consists of a belt conveyor which transports the material subjected to enrichment and a drum placed at the belt end. The drum is covered with magnets placed alternatively as to their N–S poles and it moves independently and faster than the belt. Fluctuating magnetic poles of the rotating drum induce eddy currents in the objects conducting electric current. The Lorentz force of the eddy currents accelerates conducting particles along the direction of the rotation drum pushing the particles to fall much further than the non-conductive particles falling just behind the drum’ (ibid.: 259).

For the fact that the forces affecting the particles are of low values the dielectric separators are rarely used on a larger scale (Kelly and Spotiswood 1982). The substances for which the dielectric constant does not depend on the field intensity are called paraelectrics. Drzymała (2007) notes that dielectric constants can be affected by such parameters as temperature, intensity of the electric field and its frequency.

The substances whose dielectric constant depends on the electric field intensity are called ferroelectrics. These are KH$_2$PO$_4$, NH$_4$HSO$_4$, BaTiO$_3$, NaNO$_3$, PbTiO$_3$. It should also be noted that there exists a certain temperature called the Curie point or temperature above which ferroelectrics become paraelectrics. Ferroelectrics are used for the production of high capacity and small size condensers, as well as for modulation of the frequency of electromagnetic oscillation. Another group of media used in eddy current separators are antiferroelectrics. This group includes such substances as: NH$_4$H$_2$PO$_4$, NaNO$_2$, PbZrO$_3$, NaNbO$_3$, WO$_3$. Their structure consists of permanent dipoles located on adherent planes but they are arranged opposite to each other which partially cause their compensation (comp. Drzymała 2007: 260).

### 2.2.4. Flotation

As it was already noted, one of very popular methods of separation substances is flotation. It is being used in various devices. Flotation process can be used to separate particles from water but it is most often used to separate particles of different materials. In such a case the applied collector has to be selective. Flotation collectors effect not only particle hydrophobicity but also other parameters of flotation, including the time of particle-bubble contact required to form a stable particle-bubble aggregate, as well as the stability of froth (Drzymała 2007: 307). The quoted author describes it as a dynamic and time-independent process and, when commenting on it, he states that ‘very often we want to know
how much time is needed for flotation. Complete description of flotation, including the time as a parameter is difficult. There are some general models of flotation which combine many parameters’ (ibid.: 300).

Flotation can be used for the separation of phases. In some devices it is used to remove solid particles or oil drops from water. Flotation is used for separation of particles having different hydrophobicities. The last feature, namely hydrophobicity, is a feature of material characterizing its ability to be wetted with a liquid in the presence of a gas phase. Solids which can be easily wetted with water are called hydrophilic, while solids with limited affinity for wetting are called hydrophobic. What results in hydrophobicity is that particles adhere to a gas bubble forming a particle-air aggregate which is lighter than water, and travels upwards to the surface of water. Unlike hydrophobic particles, the hydrophilic particles do not adhere to the bubbles and, therefore, fall down to the bottom of a flotation tank (comp. ibid: 270, 307).

Giving a detailed examination of the particles during a flotation process Drzymała (2007) notes that ‘(the) contact angle is expressed and measured as an angle between gas and solid phases, through the water phase. The contact angle can be also defined as the angle between solid and water phases, through the gas phase. Both ways of expressing contact angle are equally valid, since the sum of contact angle measured through the water phase, as well as the angle expressed through a gas phase is 180 degrees’ (ibid.: 270–271) It was discovered that contact angles measured on polished plates made of different naturally hydrophobic minerals are usually higher than the flotometric ones and that measurements of contact angle revealed that most minerals are hydrophilic. The simplest method of contact angle determination relies on a direct measurement of an angle of bubble attachment to a mineral surface immersed in water. It is called the captive bubble method.

As far as hydrophobic substances are concerned, the substances used include acids, bases, salts, organic compounds which are added to the solid water system. Their action can be explained with the Young equation since they modify all three interfaces (Drzymała 2007). The quoted author also adds that ‘as a rule, (…), one of the interfaces is mostly effected. The compounds modifying hydrophobicity and floatability of naturally and rendered hydrophobic materials can be classified into four groups: collectors, hydrophilization reagents, electrolytes (potential determining substances and salts), and modifiers (activators, depressors, frothers). Each of these groups differently effects hydrophobicity’ (ibid.: 280).

When hydrophobicity is regulated with the concentration of the collector, the interfacial energies of the solid-gas, solid–water, and water-gas interfaces decrease. However, the most significant is the drop in the interfacial energy of the solid-gas interface. According to the Young equation it leads to the increase in hydrophobicity and improves flotation (Drzymała 2007: 281). Hydrophobizing substances, i.e. collectors, can increase the contact
angle up to 110°, which is the value characteristic for paraffin. Generally high values of contact angles can be obtained with increased concentrations of collectors and collectors with larger hydrocarbon chain are more powerful.

As it has been already mentioned, another feature used in flotation is the materials’ hydrophilicity. Hydrophilic materials can be rendered hydrophobic with appropriate reagents. Artificially made hydrophobic materials acquire surface properties similar to those of naturally hydrophobic substances. Following Adamson (1967) and other sources Drzymała (2007) presents the following chart which illustrates the exact contact angles between a bubble of a used medium and a hydrophobic substance’s particle in the process of separation (Fig. 2.10):

![Diagram showing contact angle θ](image)

**Fig. 2.10.** Contact angle θ can be measured by the sessile drop methods (Drzymała, 2007)

Authors also note that when using this method it is important for an air bubble not to stick to particle edges because both different hydrophobicity of the edge and additional forces cause distortion of the results. There still remains the need to determine surface energy of solids in vacuum (in the devices) and in water, as well as the data regarding water adsorption on these bodies, which is crucial for determination of the film pressure in the analyses of the contact between a liquid drop and the separated substance\(^9\). Hunter (1978) specifies that surface energy of solids is difficult to determine and, therefore – to put it in his own words – ‘the data available in the literature can include a considerable error. Surface energy of solids can be estimated on the basis of surface tension of the substance melted at higher temperature and linear extrapolation to a desired temperature, without taking into account possible non-linear alterations of surface energy during crystallization’ (ibid.: 275).

What should be mentioned when talking about flotation is that Söhnel and Graside (1992) pointed out that the interfacial energy is usually determined utilizing the solubility

\[^9\] The \(\pi\) value can be, for instance, evaluated following the relation (Adamson, 1967):

\[
\pi = \gamma_g - \gamma_{gs} = -\int_{g}^{s} \gamma \, d\gamma = RT \int_{g}^{s} \Gamma \, d\ln P
\]
measurements for different particle sizes, speed of nucleation, critical supersaturation indispensable for homogenous nucleation, or kinetics of crystal growth.

Flotation, as any other process of separation, depends on properties of material used as the feed, flotation device and the way flotation is conducted. It should be summarized here that, generally speaking, the main material parameter of the flotation is hydrophobicity. The fact that the hydrophobicity is necessary for flotation (though not always sufficient) can be proved by simple experiments. Gypsum, for instance, subjected to flotation in water does not float and a drop of water put on a larger piece of gypsum wets completely the surface pointing to the contact angle equals zero. Such a simple experiments can fail in the case of other minerals, e.g. quartz or calcite, which are also hydrophilic and do not float, but a certain contact angle is sometimes detected on these minerals. Flotation can be treated as a process analoical to a chemical reaction. It takes place between particle and bubble both immersed in water:

\[
\text{particle + bubble} = \text{particle–bubble aggregate}
\]  

(2.11)

The change of the free enthalpy (thermodynamic potential) (mJ/m²) of the process can be calculated taking into account the fact that attachment a unit area of particle with a unit area of bubble creates a unit area of the particle–bubble interface.

As related by Drzymała (2007), based on Ralston (1992), ‘the contact angle in flotation is effected not only by parameters included in the Young equation, i.e. energy of interfaces involved in forming the particle–bubble aggregate but also the size of bubble and particle as well as their densities. Therefore, in some cases contact angle will not be formed, since a particle will be too heavy and will detach from the bubble. Flotation can also fail when the hydrophobic particle is too light (lack of successful collisions)’ (Drzymała 2007: 299).

2.3. The application of hydrocyclones and dense medium cyclones

Among many different devices hydrocyclones and dense medium cyclones (henceforth DM cyclones) are often used in the mineral processing industry (Kraipecha et al. 2002). The idea of using centrifugal acceleration for separation purposes was first applied to the problem of removing dust from air streams. In 1885 patents were granted for such a device in the United States and Germany to the Knickerbocker Company, USA. Bretney patented the application of the air cyclone to liquid streams in 1891 but the first recorded use of a hydrocyclone was by an American phosphate company in 1914. Unfortunately, the hydrocyclone did not see widespread industrial application until decades later when in 1939 M.G. Driessen described an application of the hydrocyclone to the dewatering of a water/sand slurry in a coal industry application. For this application, the cyclone worked very well.
In the 1940s, the hydrocyclone began to be used as a density separation device in both the coal and pulp/paper industries. The device operated well in both of these applications, primarily because of the large difference in density of the two materials that were being separated. In the coal industry, it was desired to separate rock from coal and in the pulp and paper industry the hydrocyclone was used to separate sand from paper pulp. In these applications, it was easy for the hydrocyclone to perform well as a density separation device. Because of the success in these fields, the use of hydrocyclones grew quickly. The 350+ articles that were published between 1949 and 1957 evidence the high level of interest in hydrocyclone separation. The basic design of the hydrocyclone has not significantly changed since the early acceptance of the hydrocyclone as a standard industrial device.

Dense medium cyclones are now popular in processing minerals including coal, diamonds and iron.

The cyclone separates mineral according to their differences in densities which means that the heavier particles are sent to the sinks. The centrifugal forces are generated by the flowing liquid on spiral trajectory within the vessel (Nowak et al. 2005). In the first stage of coal processing via a cyclone, the feed enters tangentially through the inlet under high pressure so that a vortex would be created inside the cyclone. In the world coal industry centrifugal dense medium separators, which operate on a principle similar to dense medium cyclones, include for example LARCODEMS, Vorsyl and Dyna Whirpool separators. LARCODEMS and Dyna Whirpool separators have been widely used to process larger particle sizes that are thought to be too large for cyclones (Reeves 2002). Dense medium cyclones are usually installed at a near horizontal inclination angle to allow usage of relatively large spigot sizes for sinks removal, and drainage of the cyclone contents during shutdown. This inclination angle is typically 10 degrees.

In a hydrocyclone an air core that extends from the floats through the sinks along the axis of the cyclone is developed. Inside the hydrocyclone two vortexes are formed. The inward one carries the floating particles to the floats stream while the outward vortex carries the sinking particle to the sinks stream. A slurry mixture of material particles and an aqueous suspension of very fine medium particles constitute the feed slurry (Nowak et al. 2005). According to the data collected by Reeves (2002) cyclone feed is normally de-slimed beforehand to remove the finest particles, typically smaller than 0.5 mm, which would have an adverse effect on the medium quality and cyclone performance.

The medium particles can either be magnetite of ferrosilicon. As one my read in Nycz (2000) magnetite is employed as medium mainly in the coal preparation. For other minerals such as diamonds and iron ore ferrosilicon is employed. There are two types of ferrosilicon namely: milled and atomised. Milled ferrosilicon is used at medium density ranges between 2.0 and 3.0RD, while atomised ferrosilicon is employed at densities between 3.0 and 4.2RD.
(Smith, 1989 and Grobler, 2006). The application range of magnetite is typically between 1.3 and 2.0RD.

Dense medium cyclones have the ability to possess high capacities, and simultaneously obtain sharp separations and high separation efficiencies. However, this piece of equipment’s capacity is constrained by the solids carrying capacity of the spigot. This is termed the **spigot capacity**. Once the spigot capacity is exceeded, the separation efficiency of the cyclone suffers. As noted by Bosman (2003a) once relatively large quantities of the feed material exit through the sinks, the spigot acts as a ‘bottleneck’. This means that it becomes necessary to install larger or more cyclones in order to achieve the required output. A detailed analysis of the application of DM cyclones will be given in chapter 3.

Having summarized the data on the most popular means of separation we shall now put a comment on the devices used in Poland for processing of coal.
3. Mechanical coal processing

Coal processing is an independent branch of mineral processing and has been an object of analyses for the past several decades. The engineers involved in developing the production of coal now form a group of specialists working in various centers of coal processing. Among the main factors that decide on the level of processing of coal one should enumerate: the quantitative and qualitative demand of the domestic and foreign clients, the characteristics of coal in the beds and the development of the local technical facilities. Apart from the fact that some mines are being equipped with machines from any foreign producers many of the facilities used for coal processing are built in Poland10. Depending on the current trend for processing the ways of producing coal have been changing for a long time (comp. Nycz 2000).

3.1. The characteristics of coal processing in Poland

Due to a decrease of the coal exploitation and closing of mines part of the processing facilities were also closed. Basing on the demand some changes in the technological systems were introduced in order to simplify them and to minimalize the costs, to improve the technology as well as in order to gain products of better quality. What is worth mentioning, even though the technology of coal processing is developing, data show that in various ways coal processing is still not satisfactory – according to Nycz (2000), for example, up to the year 2000 the production of the energetic coal still had had ash content of 22% (Nycz 2000:4).

Since 1990 the existing processing plants are still being modernised via replacing the machinery and all the necessary equipment (Nycz 2000: 26). In the years 1995–2005 in the Polish coal industry there were noticeable organizational changes. At present the processing plants exist in the following structures:

- Kompania Węglowa S.A.,
- Katowicki Holding Węglowy S.A.,
- Jastrzębska Spółka Węglowa S.A.,
- Lubelski Węgiel Bogdanka S.A.,
- Południowy Koncern Węglowy S.A.,
- Polsko-Węgierska Spółka Akcyjna „Haldex”,

10 In 2000 88% of the machinery was bought from domestic producers (Nycz 2000: 8, 18)
Henceforth we present the characteristics of the coal beneficiation in the processing plants of the largest coal producers. The further information is, among many others, based on the data collected by Nycz (2000).\(^{11}\)

### 3.1.1. Coal processing in Kompania Węglowa S.A.

Kompania Węglowa S.A. (KW) is presently the biggest coal producer in Europe. It includes 15 mines (some of them are combined mines). Production abilities of KW amount: 40 million tonnes of coal per year.\(^{12}\)

As a variety of types of coal is being exploited in KW the technology of beneficiation is also varied. Basically the coal beneficiation is based on the level of carbon content. The higher the level is the bigger the demand is and the more varied the processes of beneficiation are. The used beneficiation processes are mainly based on the date of starting or modernizing the processing plant.

A typical technological model of processing plant in the mines belonging to KW is:

- processing of coal +20 mm in dense media separators
- processing of part of the product of coal 20–0.5 mm in washers (in case of coke coal enriching part of fine coal) flotation of the slimes in case of enriching coke coal.\(^{13}\)

In the processing plants that process the coal of 31–32 type the energetic fine coal is not enriched at all or it is processed only partially.

In the processing plants which process the 33–34 type the fine coal is processed partially (more than a half in most plants) or fully. Usually the fine coal of 34 type is being completely processed. A number of mines of KW use hydrocyclone installations or spiral separators to process the particle class 3–0.2 mm. A scheme of coal processing in KW is provided below on Figure 3.1.

---

\(^{11}\) Comp. R. Nycz (2000: 8)

\(^{12}\) Obtained form the company’s website.

\(^{13}\) See also: Nycz (2000: 14)
A complete coal beneficiation is performed in the mines Anna and Marcel. In KW there are already two new facilities for fine coal processing. A plant for processing energetic
slime 10–0.2 mm in the mine Pokój with the output 450 t/h gross with the use of heavy liquid separators.

The beneficiation and de-pyrite of slimes plant in the mine Bolesław Śmiały with the output of 350 t/h gross uses a Batac-type jigger, hydrocyclones and spiral separators. All plants have closed water-slime cycles, however part of the produced slime is not further used.

The equipment of individual technological sections is diversified. This fact is associated with the period of installation or modernisation of individual plants. The quality of the coal sold by KW is strongly diversified and it is based on the type of exploited coal, the level of waste material in the output as well as in the bed and the efficiency of mechanical beneficiation.

After the processing of energetic coal type 31–32 the thick and the middle processed sortiments acquire the calorific value 23–27 MJ/kg. The processed fine coal acquires the calorific value 20–25 MJ/kg and the unprocessed fine coal – 16–22 MJ/kg.

In the processing of fine coal type 33–34 thick and middle sortiments acquire low heating value of 25–30 MJ/kg and the unprocessed fine coal 18–22 MJ/kg. Most of the processed fine coal is a type of coal of an average value. Most of the coal produced in KW has low sulphur content (average of 0.79% S).

### 3.1.2. Coal processing in Katowicki Holding Węglowy S.A.

Katowicki Holding Węglowy S.A (henceforth KHW) consists of six modern mines. The production ability of KHW is 17 million tons per year. In the structure of processing thick sortiments are c. 3 million tonnes per year and middle sortiments 1.25 million tonnes per year. The rest of the quantities cover fine sortiments.

All mines in KHW have their own processing plants where the processing is attuned to the qualities of the actually and prospectively exploited coal beds. Nearly all plants operate in two-system units model with full processing of thick and medium sortiments in DISA concentrators with dense medium.

Jigger tanks for processing graded sortiments are used in the plants: Murcki, Mysłowice and Staszcic. The mine Murcki is the only plant to use flotation processes of coal slime. The aim to lower the slime grade (<0.5 mm) is best illustrated by the use of slime-water circuits which are used in closed cycles with water clarification in the clarifiers of DORR type.

Medium parameters of coal produced in KHW are:

- Calorific value 23.89 MJ/kg,
- ash content 16.2%,
- sulphur content 0.62%,

---

14 Obtained from the company’s website.
Thick grades:
- calorific value 28.8 MJ/kg,
- ash content 4.1%,
- sulphur content 0.43%.

Middle grades:
- calorific value 22.56 MJ/kg,
- ash content 19.5%,
- sulphur content 0.68%.

Fine coal grades (<20 mm):
- calorific value 22.56 MJ/kg,
- ash content 19.5%,
- sulphur content 0.68%.

3.1.3. Coal processing in Jastrzębska Spółka Węglowa S.A.

Jastrzębska Spółka Węglowa S.A. (henceforth JSW) consists of six coal mines and is the main producer of coke coal in Poland. At present it is the only producer of coking coal type 35. The products is usually used in metallurgy. The basic product of JSW is the coke coal type 35.1 and 35.2. It is produced in the mines Borynia, Pniówek, Zofiówka and JasMos. Its main feature is the noticeable clearance and very high coking qualities. This coal possesses high duetility and agglomerating capacity. Its reaction and post-reaction index (CRI, CSR) seem profitable. Coals from different mines are used as basic ingredients of feed and they determine the variability of parameters of the coal.

All mines of JSW have processing plants which are characterised by full technology of coal processing and conduct of all processing operations in which the entire output of the mine is being processed.

In the plants which exploit coke coal type 35.1 and 35.2 (Borynia, JasMos, Pniówek, Zofiówka), depending on the grain size, the processing is conducted in three production sections:
- processing in a dense-medium washer of grain class over 20 mm,
- processing in a jig washer of grain class 20-0.5 mm,
- flotation (grain class under 0.5 mm).

In the mine Krupiński which exploits gas-coke coal type 34 the processing is limited to two sections:
- processing in a jig-washer of grain class 70-0.5 mm,
- flotation of grain class under 0.5 mm.

All plants have closed water-slime circuits and a separate post-flotation of tailings. Three plants are also equipped with drying equipment for the coal of smallest grain class.
For economic and ecological reasons this model of production is being withdrawn from and settling centrifuges are usually used. This eliminates the thermal process of coal drying. The technological scheme of the processing plant is illustrated in Figure 3.2.

Fig. 3.2. Simplified technological flowchart for beneficiation plant in JSW

Processing plants in JSW mines (except for Krupiński mine) work with comparable technological model whereas the existing differences come from projects’ terms, different facilities and realisations.
3.1.4. Coal processing in Lubleski Węgiel “Bogdanka” S.A.

The Lublin Coal Basin covers the area of about 5000 km$^2$ and has geological resources of 38 bln tonnes of hard coal. Lubleski Węgiel “Bogdanka” S.A. (LWB) is placed in the Central Region of Lublin Coal Area\textsuperscript{15}.

The technological sequence consists of the following processing stages:

- preparation and classification of raw coal,
- processing of raw coal class 200–20 in dense medium separators,
- processing of raw coal class 20–1,5(0) mm in water jigs,
- classification and filtration of slime,
- haulage, loading and storage of the products of processing.

The output from the mine is transferred on belt conveyors to the station of the preparation of the coal where it undergoes cleaning up from any surplus, such as wood and rocks. After that the raw coal is directed for preliminary classification where two grain classes are selected: 200–20 mm, 1.5–0 mm. The raw coal of grain size 200–20 mm is processed in triple-product dense medium washer. Due to considerable washing-out of clay rocks in processing tailings are eliminated in the first place.

After dewatering and classifying into grades the concentrate is transported to a container for trade products or after crunching to become fine product (Nycz 2000: 21).

After dewatering the tailings are processed to a tailings container and further on for trade use or mine storage. The indirect product after dewatering and crunching is forwarded to processing in washers.

The raw coal 20-1.5 mm is processed in two-product pulsator jig. After dewatering the tailings are transported to containers of fine coal.

The washery effluent from both washers is processed for classification in a system of hydrocyclones. The condensed and decayed coal slime after being dewatered in filters are transported to the fine coal storage, whereas the clayed slime (overflows) after densing in clarifiers (beam tanks) and dewatering in filtrating conveyors are directed to market use or to the waste containers. The technological scheme is illustrated in Figure 3.3.

\textsuperscript{15} General data about the preparation of raw coal is also elaborated on in Nycz (2000: 19)
The coal from Lublin type 32.1 and 32.2 is characterised by being considerably prone to crunching and washing-out. The basic products of the processing plant are coal slime (20-0 mm) with calorific value of 23 MJ/kg, which stands for about 86% of production. The remaining parts of production are concentrates of thick and medium-sized coal of calorific value 27–28 MJ/kg.
3.2. The technology of mechanical coal processing in the Hard Coal Mine Zofiówka (Kopalnia Węgla Kamiennego Zofiówka)

Underneath we present a detailed description of the technology of coal processing in KWK Zofiówka mine and we present the quality parameters of the coke coal with the densimetric analysis.

The Mechanical Processing Plant of Zofiówka mine includes the following sections:
− preparation yard,
− initial classification,
− grade washer,
− fine coal washer,
− dewatering of concentrate,
− dewatering of the interlayer,
− dewatering of the rocks,
− flotation,
− settling centrifuge,
− loading station,
− heap.

The transport of raw coal is performed with conveyors from hoists with capacity 2×20 Mg. The preparation yard serves the preparation of the output of the mine and here the product undergoes the following processes: separation of the grain class +200–0 mm on bar screens WK–1 1.8×3.0, eliminating the waste materials (wood, metal, etc.), crunching the raw coal +200 mm in the crunchers Brieden BB60 to the grain class 200–0 mm. There is a possibility of storing the raw coal 200–20 mm in the containers \( V = 2 \times 1000 \) Mg. The raw coal 200+0 mm, after being taken from the preparation yard, via the storing containers is directed to vibrating screens type PZ–2275. It is done so in order to separate 20–0 mm and 200–20 mm classes. There is a possibility to store the raw coal 20–0 mm in reserve tanks \( V = 2 \times 500 \) Mg.

Before being directed for processing in a dense medium washer, the coal 200–0 mm undergoes desludging in washing cradles with a grizzly. The class 200–20 mm is being processed in dense medium separators type DISA2 S-B=300 and DISA2 S-B=2000 in two technological systems. Dewatering of outputs of processing from the dense medium washer takes place in the vibration screens PWP:

a) Concentrate – PWP–2 2.2×5.5,
b) Byproduct – PWP–2 1.25×5.5,
c) Tailings – PWP 1.8×5.5.

The crunching of products of processing from the dense medium washer takes place in the crunchers type UP to the class 20–0 mm:
a) Concentrate UP $\varphi$965×1200,
b) Intermediate product UP $\varphi$840×700.

Regeneration of the heavy liquid is conducted in recuperators M1 and MR–20. The crushed half-product from the dense medium washer is sieved on a grizzly with a mesh of 6 mm. The dewatered interlayer of 10–0 is directed to processing in a fine coal jig OM–18.

Three-product processing of raw coal 20–0 mm takes place in water jigs OM24 and OM18PE. Dewatering of 20–0 mm concentrate is conducted on screens OSO 3200, OSO 2400, OSO 2000 and vibrating centrifuges WOW 1.3, HSG 1200, HES 1300 and centrifugal drainers NAEL3A and H–1000. Dewatering of the indirect product takes place in belt-bucket elevators. The operations that follow the dewatering stage are:

1. The clarification and thickening of slime waters – conducted in hydrocyclones $\varphi$540 mm and deslimers Dorr $\varphi$30 m;
2. The preparation of flotation feed together with the flotation reagent and two-product processing of coal slime 0–0.5 mm – done in flotation machines IZ-5. Dewatering of flotation concentrate takes place in sediment centrifuges SB 6400;
3. Thickening and clarification of post-flotation tailings – in deslimers Dorra $\varphi$30 m in the process of flocculation;
4. The mechanical dewatering of the thickened post-flotation tailings – on a chamber filter press, directed to filling-in the spaces under the ground;
5. The clarification and storage of surplus of technological waters and post-flotation tailings – in a ground settling tank of $V = 6.5$ mln m$^3$ capacity;
6. The storage of the raw coal, the moulds and the interlayer elements with ZOWA equipment (whereas the collecting of the outputs is done with loaders ŁWK 103 and KWKG 125);
7. The transport of the slime and waste suspensions as well as the clear and regenerated water is done with the use of impeller pumps;
8. The transport of raw coal as well as the transport and loading of the output of processing is conducted with the use of belt conveyors.

### 3.2.1. The final product of processing plants

1. Coke coal – produced in type 35.1 grade MII 20-0 mm is directed for export and the local buyers in class 7/10 and in class 8/10. The control of the charge coal is based on taking samples of the coal which is loaded onto the railroad cars and determining the ash content, moisture and the calorific value with the WILPO analyzer. The quality parameters form the samples are determined by the central laboratory;
2. The intermediate product of 19/27 class (and lower) is transported to the power plants and the neighbouring heating and power plants;
3. The slime – which constitutes a minor percentage of the final output is sold to individual buyers;

4. The tailings:
   - Tailings from dewatering of 200–0 mm class are directed to the dumping ground,
   - Postflotation tailings are directed to the dumping ground,
   - The pulp of the post-flotation tailings is used as hydraulic filling in the mine.

3.2.2. The storage in the heaps

The following output is directed to the heaps:
- raw coal class 200–0 mm and 20–0 mm,
- charge coal class 20–0,
- intermediate class 20–0 mm.

3.2.3. The slime-water circuit

The water circuit of the processing plant is a closed circuit. Losses in this circuit are covered by the water from the neighbouring sewage treatment plant and the rain water.

The clarification of technological waters:
1. The washery's water cycle – the basic equipment for clarifying the slime suspensions of the washery are the hydrocyclones and the Dorr deslimers;
2. The flotation water cycle – the main equipment for clarifying the post-flotation water are the Dorr deslimers and the ground settling tank. Dewatering of the output of flotation is conducted in settling centrifuges and filtering presses. The equipment which closes the water cycle are the filtering presses, the Dorr deslimers and settling tanks.

In the Dorr deslimers, in the water circuit of flotation, in the aim to make the sedimentation and the clarification of water faster, the process of flocculation is conducted.

3.2.4. The reagents applied during processing

During the processing the following reagents are applied:
- Magnetite creating the dense medium,
- Flotmix – a collector in the process of flotation and a foam-forming agent,
- Carbosol – the agent for spraying the cars with the products of processing during the winter season,
- Magmaflot – a flocculation agent,
- Zinc chloride – used for conducting the analysis,
- Corbosol – an antifreezing agent for spraying the post-flotation tailings in the winter season.
Underneath we present the parameters of the coking coal (table 3.1 and 3.2) and a technological scheme (Fig. 3.4) of the processing plant in Zofiówka mine as well as the dewatering schemes and classification of concentrates. In the scheme we give the names and types of the applied processing machines with their basic technological parameters.

Table 3.1. Specification of coal parameters and properties produced in KWK Zofiówka.

<table>
<thead>
<tr>
<th>Parameter or property</th>
<th>Unit</th>
<th>KWK Zofiówka</th>
</tr>
</thead>
<tbody>
<tr>
<td>Type of coal</td>
<td>—</td>
<td>(35.1 i 35.2 A)</td>
</tr>
<tr>
<td>Grain class</td>
<td>mm</td>
<td>0–20</td>
</tr>
<tr>
<td>Ash content</td>
<td>A_d (%)</td>
<td>6.0–7.5</td>
</tr>
<tr>
<td>Moisture content</td>
<td>W_i (%)</td>
<td>8.5–10.0</td>
</tr>
<tr>
<td>Volatile content</td>
<td>V_daf (%)</td>
<td>24.5–26.8</td>
</tr>
<tr>
<td>Caking power</td>
<td>Roga Index</td>
<td>77–85</td>
</tr>
<tr>
<td>Free Swelling Index</td>
<td>FSI</td>
<td>7.5–8.5</td>
</tr>
<tr>
<td>Dilatation</td>
<td>b (%)</td>
<td>100–160</td>
</tr>
<tr>
<td>Vitrinite reflectance</td>
<td>R_0 (%)</td>
<td>1.15</td>
</tr>
<tr>
<td>Plasticity according to Gieseler</td>
<td>ddpm</td>
<td>1 100</td>
</tr>
<tr>
<td>Type of coke as to Gray-King</td>
<td>—</td>
<td>G8</td>
</tr>
<tr>
<td>Coke Reactivity Index</td>
<td>CRI (%)</td>
<td>25.0</td>
</tr>
<tr>
<td>Coke Strength after Reaction</td>
<td>CSR (%)</td>
<td>68.8</td>
</tr>
<tr>
<td>Grindability</td>
<td>Gr H</td>
<td>89</td>
</tr>
<tr>
<td>Calorific value</td>
<td>Q_i (MJ/kg)</td>
<td>29.3–29.9</td>
</tr>
</tbody>
</table>

Table 3.2. Analysis of coal from KWK Zofiówka.

<table>
<thead>
<tr>
<th>Element</th>
<th>Amount [%]</th>
</tr>
</thead>
<tbody>
<tr>
<td>Carbon C_a</td>
<td>83.0</td>
</tr>
<tr>
<td>Hydrogen H_a</td>
<td>4.0</td>
</tr>
<tr>
<td>Nitrogen N_a</td>
<td>1.44</td>
</tr>
<tr>
<td>Sulphur S_i</td>
<td>0.66</td>
</tr>
<tr>
<td>Oxygen O_a</td>
<td>3.70</td>
</tr>
<tr>
<td>Phosphorus P_a</td>
<td>0.052</td>
</tr>
<tr>
<td>Chlorine Cl_a</td>
<td>0.11</td>
</tr>
</tbody>
</table>
3.2.5. KREBS cyclones used in processing plant in Zofiówka

Krebs cyclones are used in Zofiówka. Cyclones are operated in two independent settlements, two devices in each. They are working as classification cyclones, with cut-size D50 of 0.3 mm. Function and installation in organisational scheme of the processing plant are described in previous subparagraphs of this chapter.

According to the producer, the device must be provided with a proper way of keeping. The cyclones are lined with rubber and the plant workers using them have to be aware of the fact that Krebs cyclones should be kept away from sunlight and heat and be protected from extreme weather conditions. The best location for the cyclone is a cool, well-conditioned building – away from any device producing ozone to which the rubber elements are highly sensitive. In case of being kept outside, they should be covered with heavy, non-transparent facing made of plastic. The lining should allow for proper ventilation and prevent from
temperature changes as the elements made from natural rubber are heat and cold sensitive. However, in a proper way of maintenance, the external temperature should be lower than 49°C. The equipment should not be affected by low temperature if it is kept dry. One ought to note that temperatures below 0 might be a problem for the proper maintenance of the equipment.

The operations of the Krebs cyclone are typical of most of cyclones in general. As feed enters the chamber, a rotation of the fluid begins, causing centrifugal forces to act on particles that have sufficient mass, moving them toward the outer wall of the cyclone. The particles migrate downward in a spiral pattern into a conical section. At this point, as it was already roughly described, the smaller mass particles migrate toward the centre and spiral upwards and out through the vortex finder. The overflow is normally discharged at or near atmospheric pressure. The higher mass particles, upon reaching the conical section, remain in their downward spiral path along the walls of the conical chamber and gradually work their way out through the apex orifice. The underflow, as with the overflow, is normally discharged at atmospheric pressure.

The proper functioning of the cyclone is ensured by a good relation between the inlet, outlet and the overflow. This factor we shall describe in details in chapter 6. KREBS cyclones are designed in a way which allows the users to exchange the nozzles in an easy way allowing for an efficient exploitation of the device. The bow-shaped inlet directs the feed to the cylinder. Its shape lowers the probability of turbulence and of accidental transport of the oversize grain to the sink. It allows also for obtaining lower pressure and for using bigger sinks – as related to adequate cyclones with orthogonal inlets. The overflow determines the pressure which means that the bigger its diameter is the bigger solid faze enters the overflow. The role of the underflow is to allow for disposal of the heavy phase once it reaches a proper density. Here one should note that the flow at the underflow changes from a ‘cone-shaped’ discharge with an air core in the middle to a ‘rope-shaped’ flow – it is a consequence of the fact that when a rope discharge commences at the spigot then in this part of the cyclone the air core collapses. The slurry then occupies the entire cross-section of the spigot (with no air core at the underflow). Such phenomenon is referred to as ‘roping’. For details we shall come back to this phenomenon in chapter 6.

The conversion of the flow and speed to kinetic energy is due to the energy provided by the pumps. Every change in the set of pumps affects the variables in the cyclone. It is important to obtain proper design of the pomp sump. Many operators are of the opinion that a full sump guarantees elimination of changes of the diameter of the pumped material. This appears to be an inadequate approach as vertical piping that make the sump work can allow the air in – consequently interrupting the pressure at the inlet. It should be noted, however, that it is difficult to acquire a stable size in any pump set. The elimination of the low
aberrations in the size is usually obtained by installing float valve in the inlet of clean water. The float should be placed in such a way that the valve reacts only in case when the level of the suspended matter in the sump. Such solution allows preventing from full evacuation of the substance in the sump and letting the air enter the device.

It should be noted here that if an overflow is obtained for a given particle diameter, then the recovery to the underflow of that particle diameter must be about 99%. This corresponds to a value of 2.2 times larger than the D50 value as shown on the reduced recovery graph underneath. Therefore, the relationship between D50 and the 1% to 3% level in the overflow or so-called “mesh of separation” is simply related by the factor 2.2.

The base starting point in determining the separation of a given cyclone is to begin with the base D50 point for that cyclone as shown in Figure 3.5.

![Graph](image)

**Fig. 3.5. Cyclone diameter versus D50, for Standard cyclone**

(Arterburn 1997)

The micron size shown in the graph is the basic separation that can be achieved by a device one may call a typical cyclone. For example (as the producers estimate), a 10"
diameter cyclone has a base D50 point of 24 microns and a mesh of separation of 2.2 times 24 microns or 53 microns (270 mesh on the Tyler Series).

The basic value of D50 must then be corrected to reflect the variables such as pressure drop, concentration of feed solids, and the specific gravity of the liquid and solids involved. This correction is simply done by multiplying a set of correction factors times the basic D50 value. The formula is shown in equation (3.1).

\[ D_{50_{	ext{corrected}}} = D_{50_{	ext{base}}} \cdot C_1 \cdot C_2 \cdot C_3 \]  

where \( C_1, C_2, C_3 \) represent correction factors for the different variables.

The first variable is the concentration of the solids contained in the feed fluid. The correction for this variable is shown in Figure 3.6 and points out the importance of this variable.

![Fig. 3.6. Influence of feed concentration on separation (Arterburn 1997)](image)
For example, a feed concentration of 30% solids by volume would result in a separation 3.3 times coarser than the base separation. It should be pointed out that this graph is a relative measure of slurry viscosity and is affected by such things as the size of particles present as well as particle shape. For example, a feed that contains a large amount of “slimes” would tend to shift this curve to the left or result in a coarser separation whereas an absence of slimes may tend to shift this curve to the right or result in a finer separation. Other variables such as the liquid viscosity also affect this graph and a certain amount of “experience” must be applied, as the curve shown is for average conditions. The next variable to consider is the pressure drop across the cyclone as measured by taking the difference between the feed pressure and the overflow pressure. This pressure drop is a measure of the energy being utilized in the cyclone, and its effect on separation is shown Figure 3.7. As seen on the graph, a higher pressure drop results in a finer separation and a lower pressure drop in a coarser separation.

![Graph](Arterburn 1997)
Further variables to be considered are the specific gravities of the liquids and solids involved as well as the particle diameter. These, however, are going to be described in the following chapters.
4. Hydrocyclones and dense medium cyclones

So far we have been calling the cyclones which use water as medium ‘hydrocyclones’ while those operating with dense medium we named ‘dense medium cyclones’. The term ‘cyclone’ has covered both devices. It should be noted, however, that depending on the country the terms do not find clear differentiation. For this reason the author has to opt for such a terminology that he or she finds most convenient.

4.1. Hydrocyclones and classification

The basic principle the hydrocyclones operate on is the hydraulic classification (or separation). The product of hydraulic separation, containing fast settling particles, is usually called ‘underflow’, while the second product, containing slowly settling particles, is called ‘overflow’. As one may read in handbooks classification can be also conducted in a medium passing through the classifier with a spiral particle movement. This kind of movement is obtained in various devices, such as air cyclones, centrifuges, and hydrocyclones, and it results from a cylindrical shape of separators, as well as from pushing the feed stream tangent to the classifier wall. Hydraulic separators can be used to remove very fine particles from the feed, separate larger and heavier particles from lighter and smaller ones, divide the feed into narrow size fractions and, as one may read in the literature, ‘limit lower and upper range of particles size because of requirements by the applied technology, as well as to regulate the size reduction during grinding’. Hydraulic separation can be carried out in stationary media, in the media moving vertically, horizontally, sideward, and in a pulsating or spiral stream (Drzymała 2007: 168–169). Figure 4.1 depicts of a scheme of a cyclone.
As mentioned previously, one of the core phenomenon engaged in separation via cyclones is gravity, which allows for separating heavier particles from the lighter ones. The initial step for it to perform its role in separation is the creation of a vortex inside the device. Simply speaking, the suspension pumped tangent to hydrocyclone walls, due to its variable diameter, forms a spiral, directed downwards the stream around the walls of a hydrocyclone. Drzymała summarizes: ‘since large particles are more strongly subjected to centrifugal force, the spiral carries mostly large particles. Specific design of hydrocyclone causes that around its axis there is an air core, close to
which there is a second spiral stream of the liquid, directed upwards. The secondary spiral stream carries fine particles to the overflow’ (Drzymała 2002: 179).

The equation, which determines the relation between the velocity of settling of spherical particles \( v \) (m/s) and their density in the medium \( \Delta \rho \) (kg/m\(^3\)) having diameter \( d \) (m) and resistance factor \( \zeta \) (dimensionless factor which characterizes the resistance of the medium to a moving particle) is as follows:

\[
v = \sqrt{\frac{4 \cdot \Delta \rho \cdot d}{3 \cdot \zeta - \rho_c}}
\]

where:

\( \Delta \rho \) – difference between densities of particle in vacuum \( \rho_p \) (kg/m\(^3\)) and medium \( \rho_c \) (kg/m\(^3\)), i.e. \( \rho_p = \Delta \rho = \rho_p - \rho_c \) (kg/m\(^3\)).

This equation is based on the balance of the forces taking part in the process, that is the weight of particle in the medium \( F_c \), resistance force \( F_o \) which opposes particle falling down and the force of inertia \( F_i \). Particle movement in the hydrocyclone results from the force balance in the system. The sum of forces should equal zero:

\[
F_c + F_o + F_i = 0
\]

As a result of the spiral movement, the particles are subjected to centrifugal force which is the main separating force (Drzymała 2007: 178). The operation principle of a hydrocyclone and its design is shown in Figure 4.2.
Resultant force \( F_w \) effecting the particles is the difference between the centrifugal force \( F_{\text{cent.}} \) causing particle movement towards hydrocyclone walls and medium resistance force \( F_R \) directed to the cyclone axis:

\[
F_w = F_{\text{cent.}} + F_R
\]  \( (4.3) \)

The general equation describing particles movement in a hydrocyclone is:

\[
m \cdot \frac{dv}{dt} = \frac{\pi \cdot d^2}{6} \cdot (\rho_\text{p} - \rho_\text{c}) \cdot \frac{\nu_t^3}{R} - \xi \cdot \frac{\rho_\text{c} \cdot \nu_t^3}{2} \cdot \frac{\pi \cdot d^2}{4}
\]  \( (4.4) \)

where:
- \( m \) – particle mass \( (\pi \cdot d^2 / 6) \),
- \( t \) – time,
- \( \nu \) – relative velocity of particle movement in the suspension in hydrocyclone,
- \( \nu_t \) – liquid tangent velocity in the suspension depending on inlet pressure,
- \( \rho_\text{p} \) – particle density,
- \( \rho_\text{c} \) – liquid density,
- \( R \) – hydrocyclone radius in the consideration site,
- \( d \) – particle diameter,
- \( \xi \) – particle resistance factor.

Since the description of particle movement may appear complicated Drzymała (2007) suggests referring to empirical relations that are used for hydrocyclone design. The mentioned author provides his reader with equation from Koch and Nowaryta (1992):

\[
d_{\text{eq}} = \sqrt{\frac{18 \cdot \eta_\text{c} \cdot V_o \cdot R \cdot \rho_\text{hu}}{2 \pi \cdot d_i \cdot h \cdot (\rho_\text{p} - \rho_\text{c}) \cdot \Delta p}}
\]  \( (4.5) \)

where:
- \( \eta_\text{c} \) – suspension viscosity,
- \( V_o \) – suspension stream (volume per unit time),
- \( d_i = 2 \cdot r_i \) – overflow outlet diameter,
- \( h \) – hydrocyclone height from underflow to the overflow pipe,
- \( g \) – acceleration due to gravity,
- \( \rho_\text{c} \) – suspension density,
- \( R \) – hydrocyclone radius,
- \( \Delta p \) – pressure drop in hydrocyclone.
Coming back to the very idea of using cyclones in practice we should note that there are many types of cyclones that make use of water as the separator. When speaking of their categories, we ought to understand both the materials that are subject to the operations as well as the producer’s projects. From the technical point of view separators can be divided into cylindrical, conic and cylindrical–conic hydrocyclones. They are applied in grinding circuits for removal of fine particle from the systems. The main advantage of hydrocyclones is simplicity of their operation, their small size and low price. Hydrocyclones do not possess movable parts. They can work under different angles, although vertical position is recommended to create gravitation discharge of the underflow. Separation time in these classifiers is short. Hydrocyclones, however, have some disadvantages. One of them is lack of always-precise particle classification. The devices are not suitable for extremely fine particles, especially of a micron size. They also require stable composition of feed (Drzymała 2007: 180).

Features including low capital cost, no moving parts, the compactness of design, minimal maintenance and operator attention, and efficient operation make this device attractive for a wide variety of applications (Aktaş 1998: 242–246).

4.2. Dense-medium separators

As it is summarized in the literature, dense-medium gravity/centrifugal separators are widely used in the mineral processing industry for classifying particles by density and size. Rao et al. (2003) wind down the so-far usage in a statement that ‘vast quantities of coal and mineral fines in a broad size range of 0.5–100 mm are treated in a host of dynamic separators, such as dense-medium cyclone, Vorsyl separator, Larcodems, Chance Cone Separator, Dyna Whirlpool, Tri-Flo separator and others’. As the authors say, in these systems, the feed to the separator consists of de-slimed coal or ore particles in a dense-medium suspension. Particles of different sizes, shapes and densities are, pursuant to what was already stated, separated from each other due to the differential settling rates in the dense-medium fluid whose density can be controlled. The presence of centrifugal force in centrifugal separators enables separation of fine particles at a rate faster than that achievable in gravity-only concentrating units (Rao 2003: 443).

Having discussed general operations and geometry of cyclones we would like to present a sketch of types of cyclones used in industry and put a short comment on their technical differences and applications in chapter 5.

4.3. Literature review on dense medium cyclones

The concept of fine coal beneficiation by heavy medium is neither new nor untried. The first plant that employed heavy medium to beneficiate fine coal was built at Tertre in
Belgium in 1957. Approximately nine years later a second plant was built at Winterslag, also in Belgium. This plant is reported to have operated satisfactorily for 20 years.

Resulting from work carried out at the Fuel Research Institute (FRI) in South Africa, a plant employing heavy medium to produce a low-ash coal from fine coal was built at Greenside Colliery and commissioned in April 1980. Plants of similar design were also built at Newcastle Platberg Colliery and at High Carbon Products in Natal. At Rooiberg Tin mine, a heavy-medium plant was also constructed to beneficiate fine tin ore. The Greenside plant remained in operation for about 15 years.

In America, a number of wash-to-zero plants as well as two fine coal heavy-medium plants were built during the 1970s and the early 1980s. The fine-coal-only plants were at Marrowbone in West Virginia and at Homer City in Pennsylvania.

More recently, a fine coal heavy-medium plant was built and commissioned at the Curragh Queensland mine in Australia.

Today, none of these plants is in operation despite the fact that heavy-medium cyclone beneficiation of fine coal has been proved to be effective. The reasons for the demise of these plants are not all fully known. There have, however, been quite a number of reports written on the topic of heavy-medium beneficiation of fine coal.

This project aims to investigate the subject of fine coal cleaning by heavy medium and to evaluate the potential of the technique for reconsideration and adaptation to our local coal processing industry.

### 4.3.1. History

**The Netherlands/Belgium**

The flowsheet for the plants built at Tertre and Winterslag in Belgium was designed by DSM/Stamicarbon and Evence Coppee in 1957. The flow sheet is shown in figure 4.3.

The plant is unique in the sense that there is no desliming of the raw coal feed to the heavy-medium cyclone circuit. The complete 10 mm × 0 mm size fraction is mixed with medium and gravity-fed to two 500-mm heavy-medium cyclones operating at a head of 4.5 m. The product and reject coal from the 500-mm cyclones is screened on conventional vibrating drain-and-rinse screens, fitted with wedge wire decks with 0.75 mm apertures.

The minus 0.75 mm coal and discard material drains through the screen decks with the medium and reports to the correct density medium tank. A calculated part of this medium is drawn off by a separate pump and fed to two 350-mm heavy-medium cyclones operating at a feed pressure of 10 metres liquid column.
The overflow and underflow from the 350-mm cyclones report to separate primary magnetic separators. The concentrate of these magnetic separators is pumped back to the main circulating medium tank.

The underflow material from the primary magnetic separators gravitates to separate dilute medium tanks. The separate dilute medium streams from the product and discard drain-and-rinse screens also report to these tanks. The dilute medium from the two tanks is pumped separately to two sets of classifying cyclones. Separate secondary magnetic separator circuits are employed to recover the remnant magnetite from the classifying cyclone underflow material. The secondary magnetic separator tailings are dewatered on sieve bends fitted with 0.5 mm aperture sieves to yield a final fine coal product and discard.

The results obtained from the plant at Tertre were reported to be good as demonstrated by the data in Table 4.1 below.

<table>
<thead>
<tr>
<th>Grain size</th>
<th>RD of separation</th>
<th>EP-value</th>
</tr>
</thead>
<tbody>
<tr>
<td>10–0.75</td>
<td>1.57</td>
<td>1</td>
</tr>
<tr>
<td>0.75–0.3</td>
<td>1.63</td>
<td>1</td>
</tr>
<tr>
<td>0.3–0.15</td>
<td>1.83</td>
<td>1</td>
</tr>
</tbody>
</table>

Two important facts may be observed in Table 4.1 namely that (a) the cutpoint density becomes progressively higher as the particle size of the coal decreases, and (b) the EP value increases with a decrease in particle size.
It should also be kept in mind that in the above table, the data pertaining to the 10–
0.75 mm size fraction were determined on a 500 mm diameter cyclone whilst the data for the
0.75–0.3 and 0.3–0.15 mm size fractions were determined on a 350 mm cyclone.

The second plant, based on the same flowsheet, was constructed at Winterslag in
Belgium in 1965. This plant reportedly also operated very satisfactorily. Although the plant
remained in operation for about 17 years, no efficiency data were determined.

The magnetite consumption at Tertre and Winterslag was reported to be in the region
of 1 kg per ton for the total plant.

**United States of America: wash-to-zero plants**

Following from the work carried out in Europe, a number of wash-to-zero plants were
built in the USA by Stamicarbon’s licensee, Roberts & Schaefer and also by other companies.

Table 4.2 lists some of the wash-to-zero plants owned by the Island Creek Coal
Company.

<table>
<thead>
<tr>
<th>Plant</th>
<th>Contractor</th>
<th>No. of cyclones</th>
<th>Feed size (inches)</th>
<th>Year commissioned</th>
</tr>
</thead>
<tbody>
<tr>
<td>Pond Fork</td>
<td>Childress</td>
<td>1</td>
<td>¾x0</td>
<td>1976</td>
</tr>
<tr>
<td>Coal Mountain # 12</td>
<td>Childress</td>
<td>1</td>
<td>¾x0</td>
<td>1977</td>
</tr>
<tr>
<td>Providence</td>
<td>J.O. Lively</td>
<td>4</td>
<td>1–¼x0</td>
<td>1979</td>
</tr>
<tr>
<td>Fies # 9</td>
<td>J.O. Lively</td>
<td>4</td>
<td>1–¼x0</td>
<td>1979</td>
</tr>
<tr>
<td>Hamilton # 1</td>
<td>J.O. Lively</td>
<td>6</td>
<td>1–¼x0</td>
<td>1979</td>
</tr>
<tr>
<td>Hamilton # 2</td>
<td>J.O. Lively</td>
<td>4</td>
<td>1x0</td>
<td>1979</td>
</tr>
<tr>
<td>Holden # 22</td>
<td>J.O. Lively</td>
<td>4</td>
<td>1x0</td>
<td>1980</td>
</tr>
<tr>
<td>No. 25 Mine</td>
<td>F.M.C.</td>
<td>3</td>
<td>1x0</td>
<td>1981</td>
</tr>
<tr>
<td>North Branch</td>
<td>Envirotech</td>
<td>2</td>
<td>1–¼x0</td>
<td>1981</td>
</tr>
<tr>
<td>North Branch</td>
<td>Island Creek</td>
<td>2</td>
<td>1–¼x0</td>
<td>1983</td>
</tr>
</tbody>
</table>

Wash-to-zero is the term used to describe the process of beneficiating a whole size
range of coal, typically 25×0 mm, through a heavy-medium cyclone without any desliming of
the feed. The fine coal in the feed is beneficiated together with the coarser coal in the same
cyclone, normally a 600-mm cyclone. A portion of the medium, calculated to prevent a
build-up of fine coal in the circulating medium, which drains through the drain side of the
drain-and-rinse screens, together with the dilute medium from the same screen, is sent to
magnetic separators to recover the magnetite. The tailings from the magnetic separator are
dewatered to recover the beneficiated fine coal.

A typical wash-to-zero circuit is shown in Figure 4.4.
Fig. 4.4. Childress wash-to-zero heavy-medium circuit for 38×0 mm coal

The popularity of wash-to-zero plants is due largely to the fact that they offer a much simplified flowsheet when compared with plant that have desliming circuits and separate fine coal beneficiation circuits. This results in a capital and operating cost saving.

The performance of wash-to-zero plants, as measured on the plus 0.5 mm size fraction, is comparable to that of conventional heavy-medium cyclone plants operating on deslimed feeds. The minus 0.5 mm size fraction, however, is not separated as effectively as was the case for the Belgian plants employing separate smaller cyclones to effect the fine coal beneficiation. In wash-to-zero plants, typical EP-values for the minus 0.5 mm size fraction range from 0.15 to 0.2.

Another factor that must be kept in mind is that the cutpoint density effected on the minus 0.5 mm size fraction is quite a bit higher than that of the coarser coal. This implies that it is difficult to control the cutpoint density and hence also the quality of the final fine coal product produced. Still, a better separation than that obtainable from spirals can be had without the need for a separate spiral installation.

United States of America: Dedicated fine coal circuits

Two plants employing dedicated circuits, purpose-designed to process 1 - 0.1 mm coal, were built in the USA. The plants were built at Marrowbone in West Virginia and at Homer City in Pennsylvania.

The flowsheet for the plant at Marrowbone is shown in Figure 4.5.
The coal preparation plant at Homer City was built to prepare ‘deep cleaned’ coal for burning in the power station. The main purpose of the plant was to reduce the ash content as well as the sulphur content of the coal fed to the power station as an alternative to flue gas desulphurisation.

The washability of the coal at Homer City is such that a very low cutpoint density (1.30) had to be employed to achieve coal of the required quality. Since the minus 1 mm size fraction is beneficiated at a relatively higher cutpoint density than the coarser coal in a wash-to-zero operation and further due to the fact that this density cannot be controlled, it was necessary at Homer City to employ a separate fine coal circuit to beneficiate the 1.0–0.1 mm size fraction.

A pilot plant test circuit was built at the US Bureau of Mines in Bruceton, Pennsylvania, to provide the required parameters for a full-scale fine coal plant at Homer City.

The fine coal circuit was commissioned in July 1978. The performance of the plant during the first year is reported to have been poor.

In February 1979, the utility owners initiated an in-depth investigation to trace the problem areas in the circuit and to rectify such problems. The following major 'flaws' were identified as a result of the investigation:

- The circulating medium was contaminated with fine coal. This was a result of entrapment of fine coal by the magnetic separators. The entrapment of coal by the magnetic separators in turn was caused by too high a percentage of solids in the magnetic separator feed;
− The high percentage of fines in the circulating medium increased the viscosity of the medium, which in turn led to reduced efficiency of separation in the 14” (350 mm) heavy-medium cyclones;
− The high percentage of fine coal in the medium made it very difficult to control the density of the medium;
− The magnetite in the circuit degraded as a result of high losses of particularly the ultra-fine fraction.

A number of modifications were made to the circuit following this investigation. The most significant of these was to modify the size split of coal fed to the fine coal beneficiation plant. Where a minus 3 mm feed was previously fed to the circuit, the screening arrangement was modified so that the 3×0.5 mm raw coal was removed and fed to a separate, conventional, heavy-medium cyclone circuit employing conventional drain-and-rinse screens.

By feeding only the minus 0.5 mm coal to the fine coal heavy-medium circuit, a large reduction in tonnage was achieved. With the smaller top size and reduced tonnage being fed to the circuit, it was possible to change the heavy-medium cyclones from 350 mm to what was considered more efficient 200 mm cyclones.

Additional Derrick screens were installed for desliming the feed. Rapped sieve bends were installed to drain medium from the product and discard following the separating cyclones. This allowed a substantial part of the medium to be re-circulated directly back to the medium tank.

The sieve bend overflow material was diluted with sufficient water before being fed to the magnetic separators to minimise entrapment of fine coal in the recovered medium.

The medium density control was improved by utilising a nuclear density gauge in conjunction with a Ramsey coil. This allowed the contamination of the medium to be “calculated” and the actual density of the medium in the circuit could thus be controlled.

The plant is understood to have operated satisfactorily after these modifications.

**South Africa**

As long ago as 1950, it was found by Van der Walt that the heavy-medium cyclone was capable of making sharp separations on minus 1 mm fine coal. It was, however, only in the early 1970s that heavy-medium cyclone beneficiation of fine coal was considered for implementation in South Africa.

With the establishment of the export low-ash coal project, a means of beneficiating fine coal in order to produce a product containing 7% ash was sought. All the existing techniques such as froth flotation, Deister tables and water-only cyclones were tested on Witbank coals and were found to be incapable of making a sufficiently sharp separation to
yield a low-ash product. As a result of this finding, heavy-medium cyclone processing of fine coal was evaluated.

Despite the success achieved in Belgium and the USA with the heavy-medium cleaning of fine coal, it was realised at the start that South African coals were much more difficult to beneficiate and that sharper separation than that achieved overseas would need to be realised.

Fourie (1987) suggested that an EP-value of better than 0.06 was required to achieve the necessary sharpness of separation in order to produce a 7% ash product with a reasonable yield from the South African fine coal.

In 1975, the Ad-hoc Pilot Plant Advisory Committee, appointed to study the potential of heavy-medium processing of fine coal, reported back to the Fuel Research Institute (FRI). The funds required for the construction of a pilot plant were subsequently voted by the Board of the FRI. Construction of the 5 tonnes per hour pilot plant commenced early in 1976 and the plant was commissioned in 1977. A simplified flowsheet of the plant is shown in Figure 4.6.

\[ \text{Fig. 4.6. FRI heavy-medium pilot plant flowsheet} \]

The work carried out on the pilot plant proved that it was possible to produce low-ash coal from Witbank fines. Excellent results were achieved and EP-values as low as 0.03 were recorded. At the same time, organic efficiency values in excess of 90% were reported.

The test work conducted at the FRI pilot plant pointed to the following critical parameters for efficient separation:
- Small-diameter cyclones (150 mm)
- High cyclone feed pressure (approx. 150 kPa)
- Very fine magnetite (99% minus 45 micron and 50% minus 10 micron)

Based upon the design of the pilot plant, four full-scale plants were then built. These were at Greenside Colliery, at Newcastle Platberg Colliery, at High Carbon Products in Natal and at Rooiberg Tin Mine.

Of these plants, the plant at Greenside remained in operation the longest. There was also very little or no documented proof of existence of the other three plants. The Greenside plant produced a low-ash fine coal product and a steam coal product. A simplified flowsheet of the plant is shown in Figure 4.7.

![Simplified flowsheet of the Greenside heavy-medium plant](image)

**Fig. 4.7.** Simplified flowsheet of the Greenside heavy-medium plant

The plant at Greenside proved to be difficult to operate although it did produce product of the required quality most of the time. Among the problems reported by the plant manager were:
- high magnetite consumption,
- difficulty in maintaining the density of the medium,
- lower-than-expected yields,
- higher-than-specification ash contents of product,
- difficulty in supplying enough superfine magnetite to the circuit,
- blockages on cyclones,
- maintenance of the Bartles screens.
Most of these problems were apparently solved and the plant did succeed in producing low-ash coal and a middling product from the fine coal fraction for a number of years.

**Australia**

The fine coal heavy-medium plant, constructed by the Curragh Queensland Mining Company in Australia, was the most recent development in the technique. As in South Africa, the idea for the plant at Curragh was to produce a product containing 7% ash from the minus 0.5 mm size fraction.

Much of what had previously been learned was incorporated into the design of the Curragh plant, which was carried out in co-operation with CLI Corporation of Pittsburgh, USA. The lessons learned from the Homer City plant in Pittsburgh in particular were taken into account.

The plant was designed around two 500 mm diameter heavy-medium cyclones. Much attention was devoted to the design of the magnetite-recovery circuits, as well as to the desliming of the feed coal.

Vibrating sieve bends were used to assist in both the desliming of the feed and the recovery of medium.

In a 1993 Kempnich et al. report that the plant started up ‘smoothly’ during commissioning. Magnetite consumption seemed low and there was no impact on the magnetite consumption of the overall Curragh operation. Problems were however encountered with the efficiency of the cyclones and it was not possible to produce the required 7% ash content in the product. Test work carried out on site led to the conclusion that the initial cyclone feed pressure of 9D or approximately 45 kPa was too low. The feed pressure was therefore increased to 120 kPa after which, a 7% ash product was consistently achieved. The fine coal plant at Curragh closed down towards the end of 1996. The exact reason for this is still not known.

**4.3.2. Technical suggestions**

**Medium**

From all the accounts of previous experience in the cleaning of fine coal by heavy medium, the magnetite medium emerges as the main parameter. For efficient operation of the separating cyclone, the medium must be kept in a clean, uncontaminated state. A build-up of fine coal or fine clay in the circulating medium increases the viscosity of the medium and negatively influences the separation in the cyclone.
It has also been found that fine pyrite may accumulate in the media. Very fine pyrite, being non-magnetic, reports to the cyclone overflow and contaminates the product coal, increasing the ash content. The coarser pyrite, by virtue of it’s high density, reports to the cyclone spigots and becomes ‘part’ of the circulating medium.

It appears that for low-density separations in small diameter cyclones, very fine medium is required. One reason for this may be the fact that in order to achieve a low relative density cut point, a low differential is needed in the cyclone. Coarser medium classifies according to size in the cyclone and this results in a high density of separation being achieved. The finer the magnetite, the smaller this ‘density shift’ becomes.

It should, however, be kept in mind that the ‘density shift’ is not only a function of magnetite sizing. In tests conducted by Stamicarbon in the Netherlands, using saturated brine as a heavy medium in a 250 mm cyclone, a ‘density shift’ was still observed. When processing fine coal using the brine medium at a relative density of 1.20, an actual density of separation of between 1.32 and 1.41 was obtained.

The same phenomenon was reported by Deurbouck (1973), who used a medium of zinc chloride, in tests conducted at the US Bureau of Mines.

Another significant finding of the work performed using brine and zinc chloride media, is the fact that EP values comparable to those obtained when using magnetite medium were observed.

Magnetic separators are inclined to achieve lower recovery efficiency with very fine magnetite. As a result, the minus10 micron magnetite is lost from the circuit. This results in a gradual coarsening of the magnetite in the circuit which could, according to most accounts, lower the recovery efficiency in the heavy-medium cyclone.

**Cyclones**

The choice of cyclone diameter for the heavy-medium cleaning of fine coal is a moot point. In theory, a smaller cyclone will exhibit a higher centrifugal velocity than a larger cyclone. This in turn, should impart more energy to the coal particles, which should assist the process of density separation. For the latter reason, 150 mm diameter cyclones were chosen for the FRI pilot plant and 200 mm cyclones were installed at Homer City.

The 150 mm cyclones at Greenside are shown in Figure 4.8.
In practice, however, a number of factors moderate the theoretical advantage that a small cyclone may have over larger cyclones. Many small cyclones are normally required to process the same amount of coal that a single larger cyclone may be able to handle. The feeding of a number of small cyclones has to be achieved by some sort of feed distributor. This may introduce problems which could negate the theoretical advantage of small cyclones. Moreover, small cyclones are prone to blockages and ‘roping’ – both factors that could seriously degrade the efficiency of separation in the cyclone.

During test work conducted on behalf of Homer City, Gonos12 found that 14” (350 mm) cyclones resulted in sharper separations than 8” (200 mm) cyclones, particularly as the relative density was reduced and when the quality of the media deteriorated. In work done by the author at the CSIR, the same phenomenon was observed and a 400 mm cyclone yielded better EP-values than a 200 mm cyclone.

It may be of significance that the plant at Curragh, designed after an extensive study of the history of heavy-medium cleaning of fines, installed two 500 mm cyclones rather than a large number of small cyclones.

**Size of coal processed**

When coal in the size range 1.0–0.1 mm is beneficiated in a 600 mm cyclone, together with coarse coal, as is the practice in wash-to-zero plants, EP values of about 0.15 to 0.20 are obtained for this size fraction.
The same size of coal, when beneficiated in the same cyclone, but in the absence of coarser coal, yields EP values of about 0.8 to 0.1, i.e. about half the values obtained in the presence of coarser coal.

One may speculate the separation of the fine coal is negatively affected by the coarser coal through a series of collisions between the coarse coal and the fine coal during the time spent in the cyclone. However, from work done by King (1984) it would seem that the separation of coarse coal particles is not adversely affected by the presence of fine particles or slimes in the cyclone. Nevertheless, the effect of slimes on the viscosity of the magnetite medium may have an influence on the separation. The finer the coal particles, the less precise the separation becomes.

**Magnetite recovery**

The main aspect emerging from the literature regarding the heavy-medium beneficiation of fine coal is that of magnetite recovery.

The recovery of magnetite is critical to ensure acceptable magnetite consumption levels as this has an influence on the economic viability of the process. Even more important is to ensure that the magnetite medium in the circuit is not degraded from a size consist perspective by the preferential loss from the circuit of the superfine, minus 10 micron size fraction.

Another aspect regarding magnetite recovery is that of ensuring that the medium in the circuit does not become excessively contaminated by fine coal. The entrapment of coal in the recovered medium from the magnetic separators proved to be one of the major problems encountered at Homer City. It is estimated that, at Homer City, as much as 50% of the incoming feed was re-circulated back to the cyclone feed tank in the recovered magnetite media. This aspect also made the control of the medium's density extremely difficult.

Extensive test programmes were conducted in the USA by Sehgal and Matoney (1981) and in South Africa by Dardis and Burks (1979) in an attempt to find solutions to the problem of recovering magnetite without entrapping excessive amounts of fine coal in the magnetic separator product.

The results of these tests showed that low solids content and low magnetite/coal ratios in the feed to the magnetic separators were required to minimise the entrapment of fine coal in the magnetite. However, these conditions result in a reduction in the recovery efficiency of magnetic separators. Dardis and Burks reported that a higher proportion of non-magnetic material in the magnetic separator feed related to a lower recovery of magnetics.

It is generally accepted that the very fine magnetite is preferentially lost in magnetic separators.
In secondary, or scavenger, magnetic separators, conditions of low magnetite concentration coupled with high amounts of non-magnetics in the feed are normally encountered. The magnetite also tends to be very fine. As a result, the recovery efficiency of secondary magnetic separators is often very low.

At Homer City, this problem was largely overcome by a novel approach. The magnetite recovered by the secondary magnetic separator was re-circulated to the magnetic separator feed tank. In this way, the magnetic content of the feed to the magnetic separator was increased to levels at which more efficient recovery was obtained. Thereafter, a proportion of the recovered magnetite was removed from the circuit, but enough was still kept in the circuit to ensure effective recovery.

The magnetite recovery was improved by an estimated 65% in this fashion and the overall plant magnetite consumption was reportedly lowered from 3 kg/ton to 1 kg/ton.

Horsfall (1972) made an interesting observation based on the data obtained from the FRI pilot plant. He noticed that the magnetite loss associated with the production of coal was much higher than the loss from the discard side. The heavy-medium cyclone classifies the magnetite by size with the result that the magnetite exiting the cyclone with the product is much finer than the magnetite reporting to the cyclone underflow with the reject coal.

By adding some of the coarser magnetite from the cyclone underflow to the feed of the clean coal magnetic separators, a reduction in magnetite consumption was achieved. The difference induced was quite substantial as shown by the following table (table 4.3).

<table>
<thead>
<tr>
<th>Magnetite loss (grams/litre in non-magnetics)</th>
<th>Variable</th>
</tr>
</thead>
<tbody>
<tr>
<td>3.2</td>
<td>No bleed of coarse magnetite</td>
</tr>
<tr>
<td>2.6</td>
<td>About 25% of coarse magnetite added</td>
</tr>
<tr>
<td>0.022</td>
<td>About 50% of coarse magnetite added</td>
</tr>
</tbody>
</table>

It could be speculated that the improved magnetite recovery may be the result of ‘chaining’ of fine magnetic particles to the coarser magnetite particles. A second contributing factor may be the fact that the magnetite content of the feed to the magnetic separator was increased, thus making conditions more favourable for the magnetic separator.

Horsfall (1974) also proposed the use of cyclones to recover magnetite ahead of the magnetic separators. Up to 50% of the magnetite in the heavy-medium cyclone overflow and underflow is said to be recoverable by using a thickening cyclone. The underflow from this cyclone will contain a clean, high-density magnetite that may be returned to the correct medium.
The overflow from the thickening cyclone will then contain most of the coal plus some magnetite and can be subjected to magnetic separation to recover the balance of the magnetite.

### 4.3.3 Remarks on dense medium cyclone

Heavy-medium cyclone beneficiation of fine coal offers the following benefits:
- sharp, efficient, separation of fine coal is possible,
- low relative density cutpoints can be achieved,
- the density of separation can be controlled,
- oxidised or weathered coal can be processed,
- oversize particles can be handled,
- the process is flexible and suited to almost any type of coal.

Notwithstanding these attributes, no heavy-medium fine coal plant is in operation today as far as can be established. The reasons for this technology not being used may be one or a combination of the following:
- high magnetite consumption,
- problems associated with magnetite recovery,
- the medium density is difficult to control,
- in spite of the sharp separation possible, it is not easy to consistently produce low-ash coal,
- the plant and equipment required are more expensive than spirals,
- the plants previously built were notoriously ‘troublesome’,
- existing equipment (spirals) has up to now sufficed to produce a suitable product.

In recent years, the coal market has become more competitive and demanding in terms of product quality and consistency in quality.

In the past, steam-raising coal with a calorific value of 27.0 MJ/kg was produced for the export market. Spirals could process fine coal and deliver a product of this quality with relative ease. At present, the steam coal market requires a product of 27.6 MJ/kg or even 28.0 MJ/kg to be delivered. This quality is beyond the capability of spirals. Heavy-medium cyclone processing is, however, able to produce a product of this quality from the fine coal fraction.

Heavy-medium cyclone beneficiation of fine coal is unfortunately more expensive, in terms of both capital cost and operating cost, than spiral separation or other techniques for fine coal beneficiation. Judging by its track record, it is also not an easy-to-operate process. However, the increased quality demands being placed on coal producers may necessitate a re-evaluation of the technique for South African conditions.
It seems from the literature that the wash-to-zero approach may be an economical option to evaluate, especially for smaller plants. The fact that a wash-to-zero plant can process the whole size range of coal in a single unit, without the need for an additional fine coal processing section, may well make it viable. The simpler plant design, compared with the plants currently in use that have separate desliming and fine coal processing equipment, may be an added virtue.

To achieve higher recovery efficiency, the route followed at Tertre and Winterslag in Belgium may be worth investigating further. By using a separate cyclone operating at increased pressure for the separation of the fine coal, it becomes possible to improve on the fine coal separation efficiency of what is in fact a wash-to-zero plant.

To be able to produce low-ash coal from the fine fraction, the use of a fines-only heavy-medium plant operating at low relative densities would be necessary. The experience from Homer City and Curragh must, however, be taken into consideration. It seems that extremely good desliming of the feed would have to be achieved and maintained. In this regard, it would be necessary to evaluate some of the ‘newer’ types of screening devices, such as the linear screen and the Pansep screen, as possible devices to fulfil this function. Alternatively, two- or even three-stage hydro-cyclones may prove viable desliming devices.

It may also be worth considering the adoption of a coarser bottom size, say 200 micron, as the initial feed to a heavy-medium fine coal circuit. Once the circuit is fully functional, this bottom size could be extended to 100 microns or smaller.

The recovery of magnetite, being the most critical aspect of the heavy-medium process, would have to be carefully reconsidered. It seems logical, based on past history, to attempt to use a ‘drain-and-rinse’ approach as the initial phase of magnetite recovery. Vibrating sieve bends, Pansep screens or linear screens could be utilised in this regard to drain medium from the product and discard coal prior to magnetic separation.

Magnetite recovery must be carried out on dilute feeds to minimise entrapment of coal in the recovered medium. New magnetic separator technology may prove of value here. The advanced materials used to manufacture magnets for magnetic separators currently result in much improved magnetic separator performance.

The Magmiser, a newer type of magnetic separator, could be gainfully employed as a scavenger unit to reduce magnetite losses. In tests carried out on a number of mines, the Magmiser was found to operate well under conditions of low magnetics and very high non-magnetics in the feed.

The next phase of this work will be aimed at evaluating the potential economic benefit that may result from the use of heavy-medium beneficiation of fine coal. If the technique is found to be economically viable, a re-evaluation of the process for adoption in South Africa will be required.
5. Various models of cyclones

The following part concern is mainly standard hydrocyclones, dense medium cyclones, vorsyl separators, air-sparged hydrocyclone and three-product cyclones.

5.1. Standard cyclones

In a hydrocyclone due to centrifugal force the classification process takes place. The specifics of the process in its application to coal slime are defined in the introduction. The process should not be exclusively identified with classification on the basis of the grain size nor with processing. ‘What is a hydrocyclone’ ask Pytka and Aleksa ‘when it is applied as a concentrator. The best definition is that it is a decaying device’. Declaying should be understood as eliminating the waste material – mainly clay. The latter ones concentrate mainly among the smallest grains of slime, especially of less than 0.04 (0.025) mm size. The size of typical hydrocyclone, not very significant, necessitates small design of the inlet which further on limitates the size of grain of the feed which is to undergo processing (comp. Pytka and Aleksa 2001). The relation between the size of the inlet and the acceptable grains is illustrated in the table underneath.

**Table 5.1. The output of hydrocyclones 350 i 100 mm in different grain classes.**

<table>
<thead>
<tr>
<th>Grain class [mm]</th>
<th>Output for specific grain classes</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>HC-350</td>
</tr>
<tr>
<td></td>
<td>F</td>
</tr>
<tr>
<td>+0.5</td>
<td>16.7</td>
</tr>
<tr>
<td>0.5–0.3</td>
<td>9.8</td>
</tr>
<tr>
<td>0.3–0.2</td>
<td>7.4</td>
</tr>
<tr>
<td>0.2–0.1</td>
<td>8.7</td>
</tr>
<tr>
<td>0.1–0.06</td>
<td>8.3</td>
</tr>
<tr>
<td>0.06–0.045</td>
<td>3.1</td>
</tr>
<tr>
<td>0.045–0.025</td>
<td>4.5</td>
</tr>
<tr>
<td>–0.025</td>
<td>44.9</td>
</tr>
<tr>
<td>Sum</td>
<td>100.0</td>
</tr>
</tbody>
</table>

where F is the feed, O is the output in the overflow and U stands for the output in the underflow. According to Pytka and Aleksa (2001) stable underflow output is achieved in 30-60% of operations. The questions that are considered before the installation of a hydrocyclone
are: the formation of ash in the processing product in the hydrocyclone and the presence of sulfur in the products of classification. In the overflow not only the clay is present but also grains of the smallest size and ash. The technological characteristics of the feed and the output of classification is presented in the table (Table 5.2) underneath.

<table>
<thead>
<tr>
<th>Grain class [mm]</th>
<th>Feed</th>
<th>Underflow</th>
<th>Overflow</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Yield [%]</td>
<td>Ash content A [%]</td>
<td>Yield [%]</td>
</tr>
<tr>
<td>0.5–0.3</td>
<td>14.7</td>
<td>34.66</td>
<td>—</td>
</tr>
<tr>
<td>0.3–0.2</td>
<td>16.3</td>
<td>29.94</td>
<td>—</td>
</tr>
<tr>
<td>0.2–0.1</td>
<td>16.7</td>
<td>28.86</td>
<td>—</td>
</tr>
<tr>
<td>0.1–0.06</td>
<td>8.0</td>
<td>27.36</td>
<td>—</td>
</tr>
<tr>
<td>0.06–0.045</td>
<td>3.7</td>
<td>30.37</td>
<td>—</td>
</tr>
<tr>
<td>0.045–0.025</td>
<td>5.9</td>
<td>31.64</td>
<td>—</td>
</tr>
<tr>
<td>–0.025</td>
<td>34.7</td>
<td>52.56</td>
<td>—</td>
</tr>
<tr>
<td>Sum/average</td>
<td>100.0</td>
<td>38.21</td>
<td>100.0</td>
</tr>
<tr>
<td>Total sulphur content, S [%]</td>
<td>1.48</td>
<td>1.94</td>
<td>0.51</td>
</tr>
</tbody>
</table>

The partition curve is generally monotonic and asymptotic to a value called the bypass. It chapter 6 it will be mentioned that a ‘fish-hook’ curve occurs sometimes when partition values are lower than the bypass are observed. A number of attempts have been made to develop fish-hook partition curve models (Frachon et al. 1999: 53–54). The partition curve is also used to compare the performance of classification in hydrocyclones. In general, the partition curve decreases monotonically with size and is asymptotic to a given value (the so called bypass). The fish-hook is observed for a range of particle sizes. As Franchon et al. (1999) note this phenomenon had received greater attention since the use of laser-based particle size analyzers allowing the distributions in finer particle sizes to be accurately measured. The quoted author also underlines that the mechanism causing the fish-hook had not been explained in a satisfying way. A number of attempts were made to model partition curves that exhibit a fish-hook. It was often assumed that the bypass can be estimated accurately from the fraction of water in the feed that reports to the coarse product stream (Franchon 1999: 54).

Before giving more details on this topic let us describe briefly other models of separators.

5.2. Vorsyl separator and dense medium cyclone

When it comes to dense medium cyclones, one should start from the agents used in separators, that is: dense medium suspensions. These are prepared by adding an aggregation
of solid to a liquid. The liquid can either be a Newtonian liquid or a non-Newtonian liquid. In most industrial processes the liquid used is a Newtonian fluid, usually water. Dense medium suspensions can further be classified as a heterogeneous material comprised of a dispersed phase and a continuous phase. The solid particles represent the disperse phase, while the liquid represents the continuous phase. It should also be mentioned that the solid particles themselves can also be a heterogeneous system comprised of solid particles of different nature and form (Mabuza et al. 2005: 25). This short introduction taken from Mabuza et al. may be summarised by a statement that dense medium cyclones make use of water and accompanying agent that helps to process coal. The dominating dense media particles used in making dense medium suspensions are magnetite and ferrosilicon. Magnetite is a naturally occurring iron oxide ore, while ferrosilicon is a manufactured alloy of iron and silicon. To reduce the corrodibility of ferrosilicon the silicon content should not be less than 15%, but it should not also be greater than 16% so as not to affect the magnetic capture of the particles in the dense medium circuit. Due to the synthetic nature of ferrosilicon its cost is considerably higher. The two most important parameters for dense medium suspensions are their stability and viscosity. The stability of the suspensions is defined in terms of the rate of settling of the solid particles in the media. Suspensions with high settling rates are considered to be unstable, while those with low settling rates are considered as being stable. The viscosity of a suspension may be defined as its resistance to motion. The stability and the viscosity of suspensions are interrelated, both dependant on the density, size and shape of the medium used and its volume concentration. This interrelation of stability and viscosity can be modified through the use of dispersing agents. A spherical shape of the medium reduces the viscosity and stability of the suspension by providing more slippage between adjacent solids in the suspension. (Mabuza et al. 2005: 25–26).

Dense medium cyclone proved to be most efficient for removing the undesirable material from coal (comp. Honaker et al. 2000: 415–416). However, it was due to the fact that though the dense medium cyclone is one of the most popular processing techniques used in India for washing coal of 13 + 0.5 mm size it had been observed that degradation of the feed quality with regards to a high amount of near-gravity material and ash has sometimes resulted in problems of controlling the performance of dense medium circuit.

With regard to the vorseryl separators laboratory studies conducted with the unit established the superiority of the process over dense medium cyclone for treating Indian coals (Rao et al., 1998). The results encouraged to undertake plant scale trials. This separator unit is vertically installed with a tangential feed inlet as shown in (Fig. 5.1) (Banerjee et al. 2003: 102).
As Banerjee et al. (2003) describe ‘a vortex finder is fixed from the base of the separator, which is surrounded by an annular space called the throat. The bottom of the separator is connected to another small cylindrical chamber called the vortex tractor. This small chamber has an outlet vortex-tractor nozzle (spigot). The original unit had a diameter of 610 mm with a capacity of 75 tonnes/h (Abbott et al., 1969). But later, a vorsyl separator of 720 mm was developed with a capacity of 120 tonnes/h. The principle of separation in the vorsyl separator is similar to that of dense media cyclones. The feed material at high pressure enters the top of the separator tangentially’. Material with a relative density less than the separation density moves to the centre of the separator and leaves through the vortex finder. The near-gravity material refuses to move to the wall and reach the throat area. At this place, a strong inward flow is generated by the orifice plate, which reduces the centrifugal force and allows an appropriate portion of near-gravity material to reach the vortex finder opening (Banerjee et al. 2003: 102). The separation in the vorsyl separators effectively handles the near-gravity material. Abbott et al. (1969) had reported a detailed work on a laboratory and plant scale model of the vorsyl separator. Their study had brought out the following interesting features on the flow pattern: (a) the zero axial velocity or dead zone present in the dense medium cyclone is not found in vorsyl separators; (b) there exists a strong recirculation of near-gravity material in the latter ones (Banerjee et al. 2003: 103).

The throat of the VS avoids excessive drainage of medium to the underflow but it does not have an effect on the performance of the VS. What is more, the involute entry gives a higher capacity and better efficiency than the standard entry. As Banerjee et al. (2003),

---

16 Orifice plates utilize the Bernoulli’s principle, which states that when the velocity increases, the pressure decreases and vice versa.
following Cammack (1978), report that the vorsyl separator performs better in accurate separation of gravity-dependent material while the dense medium cyclone produces better results for the separation a material which could be labeled as gravity low-specific. The studies covered data from plants operating in different conditions and for this reason should not be treated as fully satisfactory.

5.3. Air-sparged hydrocyclone

A new type of flotation apparatus, the air-sparged hydrocyclone (ASH), was used in grinding calcification circuit to control the over-gridding of hydrophobic materials after some modifications. In ASH the feed slurry is prepared in conditioning tank. Later it is mixed with collector (e.g. butyl xanthate) and frother (e.g. pine camphor oil) before it is pumped to the ASH. As Liang (1996) reports ‘the overgridding of hydrophobic materials, resulted from the fact that the hydrophobic materials have higher density than the hydrophilic materials, has been known as knotty problem. The modified ASH is designed with a relatively small length/diameter ratio, and a thick vortex finder wall is introduced, and the annular underflow opening between the pedestal and the cylindrical wall is replaced by wide angle coal apex opening and a horizontal baffle that is positioned above the apex opening’ (Liang et al. 1996: 74–75).

![Diagram of ASH](image)

**Fig. 5.2.** Schematic diagram of the ASH: 1 – inlet, 2 – vortex finder with thick wall, 3 – polyethylene porous cylinder, 4 – compressed air, 5 – baffle, 6 – underflow pipe (Liang, 1996)
In the ASH a wide angle cope apex opening and a horizontal baffle that is positioned above the apex opening can, here, replace the underflow opening between a pedestal and the cylindrical wall. Its function is to prevent the froth from entering the underflow. The annular opening, however, is an important feature in the ASH used by various industries. The underflow rate is adjusted by the diameter of the underflow pipe. When it comes to work of ASH we have to note here that the water split to underflow increases when the inner diameter of vortex finder becomes larger. Similar phenomenon connected with dense medium cyclone and hydrocyclones will be commented on in the coming chapters.

5.4. Three-Product Cyclones

The three-product cyclone is used, among many countries, in South Africa where it serves the platinum industry. Mainza et al. (2004) found that a three-product hydrocyclone can produce efficiently three distinct products according to size and density. Later, the flow behaviour in that three-product hydrocyclone was studied by Mainza et al. (2006). They used the computational fluid dynamics to model the three-product cyclone, in particular, the influence of the dual vortex finder arrangement on the flow behaviour. They focused on the overflow zone where the conventional overflow stream was divided into two streams. As Ahmed et al. (2009) note, the three-product hydrocyclone’s axial symmetry makes the separation process similar to that in the conventional hydrocyclone. It is also obvious that the three-product hydrocyclone was developed mainly to overcome specific problems in some applications which use the hydrocyclone as in the paper industry applications and in the classification processes of the platinum industry. This type of three-product hydrocyclone was mainly aiming to treat the overflow product by dividing it into two products.

Both the three-product cyclone and the hydrocyclone have a third output opening in the conical part along the side of the cyclone opposite to the feed opening. The feed slurry is introduced under pressure via the tangential inlet and is constrained by the geometry of the unit to move on a circular path. This creates the opposing outward centrifugal and inwardly acting drag forces which result a spiral flow pattern. An air core develops along the vertical axis which is connected to the atmosphere through the spigot, but the part created by dissolved air is coming out of solution in the low-pressure zone (ibid.: 35–36). The inward drag forces tend to dominate and light particles move towards the vertical axis and join the innermost spiral. Later they are swept up into the overflow opening. The heavy particles experience a greater centrifugal force. They move to the cyclone periphery, join the outermost spiral and move downward toward the middling flow opening and the spigot.

Experiments led to interesting observations when it comes to the operations of three-product cyclone. Mainza (2004a and 2004b) observed the ore containing chromite of a relative density of 4.5. Chromite co-exists with silica (the PGMrich component of the UG2
ore). As one may read in Mainza et al. (2004a) ‘silica has a relative density of 2.8, which is significantly lower than that of chromite. Due to the dense media effect of the high density chromite, the low density silica preferentially reports to the overflow of a conventional hydrocyclone even if it is not fine enough to be recovered in the flotation process. To compound the problem, the high density chromite reports to the underflow of the conventional cyclone when it is sufficiently fine to be discharged through the overflow. The displacement of the coarse silica to the overflow and fine chromite to the underflow in a conventional cyclone results in: a loss in recovery of the PGMs contained in the coarse silica fraction that is displaced to the overflow; the mill losing capacity for fresh feed due to the build-up of fine chromite in the re-circulating load; wasting of energy grinding down the already fine barren chromite’ (Mainza et al. 2004a: 574).

A high proportion of the chromite particles become entrained in the concentrate during the flotation process and are carried over to the smelter. However, there they cause numerous problems in the smelting process. For this reason alternative to conventional hydrocyclones ways of classification where sought. It was discovered that one device could potentially be used to minimise or eliminate the dense media effect (Mainza et al. 2004a: 573). The three-product cyclone has an additional vortex finder inserted concentric to the existing one which helps to obtain third product. The existing vortex finder is termed the outer vortex finder and the additional vortex finder the inner vortex finder. The next figure depicts a comparison of a conventional hydrocyclone and a three-product cyclone. This type of cyclone produces three streams: a finer overflow stream, middlings overflow stream, and a coarse underflow stream. The product from the inner vortex finder is termed the inner overflow and the product from the annulus (gap between the inner and outer vortex finder) is called the outer overflow. Commenting on the application of the device Mainza (2004) says: ‘three-product cyclones were installed at a number of concentrators in South Africa but none of those the authors investigated were operational. This was mainly due to the poor design of the original overflow arrangements. Two major problems were identified in the overflow arrangement designs of these cyclones: firstly, the inner and outer overflow pipes diverged at different heights above the roof of the hydrocyclone resulting in a pressure difference between the two overflow outlet pipes. This resulted in little or no flow in the inner overflow stream whose overflow pipe was much higher than that of the outer overflow. Secondly, the overflow pipes had sharp bends which caused turbulence resulting in poor slurry flow and subsequently poor separation in the hydrocyclone’ (Mainza et al. 2004a: 574). The quoted author reports that an overflow arrangement design for three-product cyclones produced at the Julius Kruttschnitt Mineral

17 More on the phenomenon of medium loss in: Napier-Munn et al. (1995)
Research Centre was adapted and modified slightly for the use in applications with large diameter cyclones. In order to avoid differences in the pressure head between the two overflow discharge pipes, the inner and outer overflow pipes curved out at the same pipe-centre height above the roof of the cyclone. To minimise turbulence, no sharp bends were allowed in the overflow pipes (Mainza et al. 2004a, comp. Mainza et al. 2004b: 575–577).

There was an investigation concerning various designs incorporating interchangeable inner vortex finders without changing the entire overflow arrangement and, for example, the collar-reducer was adopted as the method of mounting the inner vortex finder for the experimental work due to the nature of ore which the plant treats. High tensile strength grub screws were used in attaching the reducer to the collar and the reducer was fitted tightly to the collar to avoid flow of the slurries from one vortex finder to another during operation. The dual vortex finders in a three-product cyclone prevent the short-circuiting of coarse material into the overflow and help to obtain preferential separation of the material reporting to the inner and outer overflow streams. For test purposes, thin walled non-standard pipes were welded to the custom made reducers and used as inner vortex finders. To change the inner and outer vortex finders, the overflow arrangement was hoisted up, see figures 5.3 and 5.4. The inner vortex finder was removed from the overflow arrangement by undoing the grub screws that retained the inner vortex finder on the collar of the overflow arrangement (Mainza et al. 2004: 574).

![Schematic diagram of the main components of the conventional and three-product hydrocyclones](image)

**Fig. 5.3.** Schematic diagram of the main components of the conventional and three-product hydrocyclones (Mainza et al. 2004)
Fig. 5.4. Schematic diagram of the three-product cyclone test rig set-up (Mainza et al. 2004)

Having described the characteristics and questions concerning models of cyclones we shall now devote some time in chapter 6 to the analysis of the operations and processes that go with cyclones’ applications.
6. The analysis of the operations and processes concerning cyclones

It has been proven that dense-medium separators can be the most efficient devices for removing the undesirable material from the mined minerals, such as coal. As it was already mentioned the general description of the work of a cyclone is about applying high-pressure feed injection into dense-medium cyclone to provide a vortex whose centrifugal force will allow an efficient separation of fine coal. This chapter is an overview of all the operations that concern cyclones, values of the feed, ore and media as well as their analysis based on the available literature. We shall begin with a general description of classification cyclones and an outline of the processes that take part within them. Starting from point 6.1 we will present a thorough analysis of the processes inside the cyclones as well as comment on the circumstances of their output.

6.1. Classification cyclones

Classification cyclones separate solid particles according to their differences in size. These devices make use of water or suspension liquids as the separating medium, which – according to data collected by He (2001) – is the most efficient (ibid.: 2991). Usually a mixture of water and magnetite, ferrosilicon or phosphate is employed in dense medium cyclones (Guosheng Li 2012). Rayner and Napier-Munn (2000) and Furman et al. (2005) note that industries had also made use of silica sand (for coal washing) and galena (for metalliferous processing). As we read in Yuling et al. (2011) ‘the use of heavy media introduces a complexity in operational principles: the size and density distributions affect the separation process’ (ibid.:175) From the technological point of view, the partition is the phenomenon of operating conditions such as solid volume concentration and medium viscosity. This surface may become steeper or flatter depending on changes in media and ore characteristics and separator design parameters (comp. Rao et al. 2003: 444, and Xia and Li 2009: 760).

As one may read in the literature, the material carried on the feed in hydrocyclones and dense medium cyclone can be divided into two or more products. Drzymała adds that ‘the separation may provide either identical or dissimilar products as to their quality and quantity. The analysis of the separation process as division of the feed into products (SP) requires two parameters that is the quantity (yield, γ) and the identity (usually name) of the product’ (Drzymała 2002: 42). The separation provides two products: concentrate and tailing. The
quoted author also explains that the yield, which characterizes the division of the material into products, can be expressed either directly in mass units (usually megagrams (Mg) which in equivalent to old unit ton) or in a relative form as per cent in relation to the mass of the feed. It can be also expressed per time (then it is called a stream\textsuperscript{18}). It can be expressed not only per time but also per surface or surface-time unit (comp. Rayner and Napier-Munn 2000: 277–278; Drzymała 2007: 43–44). The figure (Fig. 6.1) underneath is a graphical representation of division of material into products (quantity versus identity).

![Graphical representation of division of material into products (quantity versus identity) (Drzymała 2007)](image)

**Fig. 6.1.** Graphical representation of division of material into products (quantity versus identity) (Drzymała 2007)

The mass balance of the material division into products can be expressed as:

\[ \sum y_{products} = 100\% \]  

(6.1)

and in the case of two products as:

\[ y_{productA} + y_{productB} = 100\% \]  

(6.2)

The mechanism of the operations for classification cyclones is similar to those in dense medium cyclones. However, there are differences in terms of the physical parameter (Svoboda 1998).

The efficiency of a cyclone is based, among many parameters, on the spigot capacity. We should, therefore, define what the spigot capacity is, and establish what determines it. Jull (1972), later Mular and Jull (1978) and recently Magwai and Bosman (2008) suggested that

\textsuperscript{18} Or a ‘flux’
the spigot capacity of a cyclone was reached at the onset of rope discharge at the underflow. Under the normal operating conditions of the cyclone a ‘cone-shaped’ discharge is prevalent at the spigot, and is referred to as spray discharge. During spray discharge the air core extends across the entire length of the cyclone from the overflow to the underflow.

The flow at the underflow changes from a ‘cone-shaped’ discharge with an air core in the middle to a ‘rope-shaped’ flow – it is a consequence of the fact that when a rope discharge commences at the spigot then in this part of the cyclone the air core collapses. The suspension then occupies the entire cross-section of the spigot (with no air core at the underflow). Such phenomenon is referred to as ‘roping’. We should note here that the idea that there is a link between the spigot capacity and roping flow has been implied by such authors as Dahlstrom (1949), Fahlstrom (1963), Abbot (1967a), Plitt (1976), Flintoff et al. (1987), Plitt et al. (1987) and Heiskanen (2000). Types of discharges are illustrated in Figure 6.2.

Commenting on air core and roping Dyakowski and Williams (1995) stated that the collapse of the air core during the onset of roping was a consequence of excessive ‘fluid’ viscosity, which decays the tangential velocity and, consequently, the rotational motion at the underflow. As we may read in Plitt et al. (1987) ‘roping is initiated by the formation of a bed of solids in the apex (spigot) region of cyclone. When the viscosity of the slurry increases to the point where the frictional drag of the cyclone wall stops the rotary motion, roping is initiated’ (Plitt et. al 1987: 24).

![Fig. 6.2. Schematic diagram showing spray, semi-rope and discharge at the underflow of a hydrocyclone](image-url)
Figure 6.3 shows the resistivity images which were taken in a single horizontal plane near the feed inlet.

![Resistivity Images](image)

**Fig. 6.3.** Resistivity images showing the air core below the feed inlet of a 44 mm diameter hydrocyclone

The circular white shapes in the middle represent the air core. This study is taken from Gutiérrez et al. (2000), who illustrated the above-mentioned behaviour with the use of tomographical pictures. Gutiérrez et al. (2000) also studied the effect of the feed flow rate and feed solids concentration on the air core size at the underflow.

The air core size was observed to increase in terms of increasing feed flow-rate up to a certain maximum at various feed solids concentration. As we may read in the literature an increase in the feed solids concentration consistently brings a decrease in the air core size. The decrease in the air core size with increasing solids concentration is a consequence of increased suspension viscosity, which increases with solids concentration as shown in Figure 6.4.
As Gutiérrez et al. (2000) observe once a certain critical underflow solids concentration is reached the rotational motion of the slurry in the cyclone can no longer be sustained. As a result the air core collapses and roping commences. Rao et al. (2003) notes that ‘regardless of density, as particle size decreases, the density-based partition curves flatten, and as particle size approaches zero, the partition curve becomes completely flat with the partition number equal to the fluid recovery in the underflow’ (ibid.: 444; comp. Zhu 2008)

In Figures 6.3 and 6.4 we illustrate what happens within the hydrocyclone in terms of the air core area. These two figures illustrate clearly that during roping, when the angle of discharge is zero (Fig. 6.3), the air core at the underflow collapses (Fig. 6.4).

One should note that particle density does not seem to influence the relationship between volumetric solids concentration and apparent viscosity. The abrupt rise in viscosity with increasing solids concentration was observed to occur at the same concentration of around 37–40% for all three particles types.

6.2. The characteristics of the overloading of sinks. Roping

According to Stas (1957: 161–192) when a rope discharge is prevalent at the sinks the cyclone is operating in an overloaded condition. The author notes that the sinks discharge capacity is exceeded when a rope discharge was encountered at the sinks. In their work Symonds and Malbor (2002) stated that ‘apart from the feed capacity, the cyclone has a limit to how much reject material it can handle. This is due to restriction caused by the apex’ (Symonds and Malbor 2002: 1011). In the quoted work of Stas (1957) it was proposed that the ‘angle of dispersion’ of the sinks slurry was reduced with increasing feeds solids concentration. Stas also reported an increase in the thickness of the sinks slurry with
increasing feed solids concentration, and therefore a reduction in the air core size. The decrease in the air core size is a consequence of excessive solids concentration at the sinks. Additional increase in the feed concentration resulted in the collapse of the spray discharge at the sinks which resulted in the collapse of the air core. The sinks stream was later straightened so that the slurry occupied the entire cross section of the spigot (rope discharge).

An increase in the proportion of ore in the feed that exists at the sinks necessitates a decrease in the recommended feed concentration to avoid spigot overloading. This implies that the sinks ore concentration rather than the feed concentration that is related to overloading.

To support this Stas (1957) proposed a mathematical expression that predicts the onset of spigot overloading by expressing it in terms of the sinks slurry density:

\[ u_U = u_F + (u_F - u_O) \left( \frac{0.921D_U}{D_a} \right)^{\frac{3}{8}} \]  

(6.3)

where:
- \( \varnothing_U \) – density in the sinks stream,
- \( \varnothing_F \) – the relative slurry density in the feed,
- \( \varnothing_O \) – the relative slurry density in the floats stream.

What appears to be interesting in this equation is that it is based on the premise that there is a critical sinks ore concentration beyond which roping takes place. Clarkson and Wood (1993) used a similar criterion, in which they propose that spigot overloading can be avoided by not exceeding the volumetric sinks ore concentration of 40%. Wood (1990) also reported that only a ‘little disruption of efficient separation’ was encountered.

6.3. Cyclone’s operational variables. Feed inlet pressure

The available literature on the topic of cyclones’ application deals with various aspects influencing these devices’ efficiency. These aspects cover geometrical data as well as operational data. As it is the CFD modelling that concerns us most, from all the selected analyses we have chosen these descriptions which touch upon the media’s and water behaviour.

As one may read in the theoretical studies, the efficiency of a cyclone can be enhanced by the feed inlet pressure (Mukherjee et al. 2002: 259). Following Vallebuona et al. (1995) the authors of the former work report that slurry flow rate to the cyclone can be measured by increasing such variables as vortex finder, spigot diameter as well as feed inlet pressure. These however have to also consider the characteristics of the raw coal (Mukherjee et al. 2002: 259). They also conclude what follows: ‘Consequent to increase in feed inlet pressure
and feed rate, the VF diameter of the dense medium cyclone should be increased in proportion to the additional slurry flow rate towards the overflow. This modification would reduce the pressure drop along the dense medium cyclone axis, and it would also reduce the axial velocity of the particles. The additional slurry flow rate in the dense medium cyclone would increase the tangential velocity of the particles which would help in a better separation of the fine-sized coal particles (...) and, hence, it would improve clean coal yield in the dense medium cyclone’ (Mukharejee et al. 2002: 272–273). This conclusion was based on the studies of laboratory experiments as well as plant trials.

In 1950s, De Gelder et al. (1952) and Dhalstrom et al. (1954) have shown that the pressure drop in hydrocyclone is linearly related to its throughput. Later the empirical model of a hydrocyclone, as proposed by Lynch et al. (1975), showed that throughput to the hydrocyclone increases linearly with increase in feed inlet pressure and VF diameter. Mathematical model developed on the basis of regression analysis (Plitt, 1976; Flintoff et al., 1987) reveals that at higher feed inlet pressure hydrocyclone capacity increases. The mathematical model developed for a hydrocyclone by Nageaswararao et al. (1978) is incorporated in the JKSimMet, a simulator developed by Julius Kruttschnitt Mineral Research Centre. In the late 1990s Asomah and Napier-Munn (1997) and Napier-Munn et al. (1999) carried out detailed work at Julius Kruttschnitt Mineral Research Centre, Australia, for developing an empirical model of a hydrocyclone. The pressure-throughput equation of the model suggested that throughput to the hydrocyclone increases with increase in feed inlet pressure.

Mukarjee et al. (2002) say that ‘over the years, studies on cyclone are mostly confined within hydrocyclone operation. Similar types of detail studies were not carried out for dense medium cyclone since the operation of dense medium cyclone is more complicated and its applications in the mineral-processing field is less as compared to the hydrocyclone’.

Apart from changing the inlet pressure, it was suggested that flow rate will not affect the sharpness of separation. In fact, there would be a tendency for lower interrelationship of the states at higher flow rates. Although high flow rate ensures better sharpness of separation (low $E_p$), clean coal yield deteriorates due to the shift of the relative density of separation to a lower value (Mukherjee 2002: 261). He and Laskowski (1994) have shown that the performance of dense medium cyclone largely depends on medium stability. They also found that at high flow rates, the medium stability would not be affected by fine-sized magnetite.

The quoted authors have explained the importance of feed inlet pressure on the efficiency of dense medium cyclone basing on the data acquired from laboratory and industrial experiments on Jamadoba washery of Jahiva group supplies.

\[19\] or $E_p$ value for short.
In this section units as in original research are used. Laboratory experiments were carried out with coal and magnetite sample received from the Jamadoba washery. Coal particles of size less than 2.0 mm were used as feed, and the tests were carried out on 4-in. diameter dense medium cyclone. The magnetite-to-coal ratio was maintained at 4:1, and the relative density of the feed media was kept at 1.35 following the existing plant condition. Dense medium cyclones of the plant are gravity-fed, and the vertical feed inlet is connected to feed distributor. In the plant, the height of the vertical feed inlet is nine times the diameter of dense medium cyclone, and it provides a feed inlet pressure of 5.2 psi (59.5 kPa). In the laboratory tests, positive displacement pump was used for feeding the slurry to dense medium cyclone in place of the gravity-fed system. The feed inlet pressure was achieved by controlling the by-pass valve attached to the feed inlet pipe, and the feed inlet pressure was monitored using a pressure gauge. Laboratory experiments were carried out by varying feed inlet pressure from 5.2 psi (59.5 kPa) to 11 psi (125.7 kPa). At the Jamadoba washery, coarse coal is treated on four dense medium cyclones. These dense medium cyclones are equally distributed in to two streams where each stream contains two dense medium cyclones. On the basis of the findings from the laboratory tests, feed inlet pressure was increased in dense medium cyclones of the second stream. Feed inlet pressure for dense medium cyclones of the second stream was enhanced by increasing the static pressure head and in this process, the height difference between the feed box and dense medium cyclone was increased. With the available space in the plant, feed inlet pressure could be increased up to 6.4 psi (11d) from the present level of 5.2 psi (9d), and the VF diameter of the dense medium cyclone was increased by machining the orifice surface. Plant trial was carried out at high feed rate (35 tph/dense medium cyclone), with high feed inlet pressure (6.4 psi) in the second stream and normal feed rate (30 tph/dense medium cyclone) with normal feed inlet pressure (5.2 psi) in the first stream. The data generated from dense medium cyclones of streams 1 and 2 were evaluated to compare the process performance (Mukherjee 2002: 262).
The formula for calculation of approximate capacity of cyclone based on the simple hydraulic theory and the study of published literature:

\[ Q = K_d \cdot \sqrt{AF} \]  \hspace{1cm} (6.4)

The formula indicates that by increasing feed inlet pressure, slurry flow rate to the cyclone can be increased. Ideally, with increase in flow rate, velocity of the slurry will increase (Sriririya 2001). Following general studies by Van der Welt (1950), Upadrashta and Venkateswarlu (1982) and Napier-Munn (1999) we need to note that feed pressure it has been reported to influence \( Q \) as follows:

\[ Q \propto H^n \]  \hspace{1cm} (6.5)

As expected, an increase in the feed pressure results in an increase in \( Q \). Upadrashta and Venkateswarlu (1982) reported \( Q \) to be slightly more sensitive to changes in pressure during roping than when a spray discharge was prevalent at the sinks. An increase at the entrance appears to bring about a decrease in the proportion of the feed slurry that exits at the sinks.

Van der Walt (1950) reported that the ‘maximum permissible spigot loading’. The spigot capacity increased with increasing feed pressure. According to Cohen and Isherwood (1960), the effect of pressure on the rate of flow through the spigot is restricted by the high viscosity of the sinks pulp. Napier-Munn (1986) reported that the pressure drop across a dense medium cyclone appears to be a function of the total medium rheology. He proposed that the
relationship between flow through cyclone and pressure at low Reynolds numbers is as follows:

\[ Q \propto H^{1.89} \]  

(6.6)

and medium viscosity plays little or no part in determining the pressure drop. At high Reynolds numbers the relationship between flow through cyclone and pressure can be again represented as follows:

\[ Q \propto H^n \]  

(6.7)

with \( n < 0.5 \). The value of the exponent \( n \) is related to the prevailing medium viscosity\(^{20}\).

In laminar flow regime, with increase in slurry velocity, the radial velocity of the particle will also increase, and this would reduce the drag force on each particle as per the equation given below:

\[ F_D = 0.5 \cdot C_D \cdot (v_p - u_r)^2 \cdot \rho_f \cdot A_c \]  

(6.8)

As reported by Mukherjee (2002) at low drag force, particles will move towards the cyclone wall, i.e., the hypothetical surface of no axial velocity would shift towards the cyclone wall; as a result, volume of the separation zone would increase, and this would facilitate in increasing slurry flow rate to the dense medium cyclone. At high flow rate, tangential velocity of particles will increase and, hence, centrifugal force on feed particles would also increase. Therefore, particles of specific size and density will follow the equilibrium orbit having larger radius at high feed inlet pressure than that of the normal feed inlet pressure. Following Monredon et al. (1992) Mukherjee et al. (2002) observe that feed particles report to hydrocyclone overflow and underflow depending on the slip velocity of the particles along the radius and axis of the cyclone. The radial \( (U_S) \) and axial \( (W_S) \) slip velocities of the feed particles are defined as:

\[ U_S = \frac{4}{3} \cdot \frac{\rho_p - \rho_f}{\rho_f} \cdot \frac{V^2 \cdot D_p}{R \cdot C_D} \]  

(6.9)

\[ W_S = \frac{4}{3} \cdot \frac{\rho_p - \rho_f}{\rho_f} \cdot g \cdot \frac{D_p}{C_D} \]  

(6.10)

\(^{20}\) Comp. Wang et al. (2011: 21)
For good-quality feed coal, proportion of overflow would be much more than the underflow. Due to the increase of the inlet pressure and the development of the free vortex along dense medium cyclone axis, a strong axial velocity will carry magnetite particles to overflow. Therefore, pressure drop would increase relative density difference between overflow and underflow and also the relative density offset between the relative density of the feed media and relative density of separation. This situation is deterrent to efficient operation; it would initially reduce the clean coal yield and then the clean coal quality due to the choking and misplacement from the underflow discharge end. Moreover, fine-sized coal would report to the overflow stream without any separation (Mukherjee 2002: 263).21

On the other hand Bradley (1965) found that at very high feed inlet pressure, the pressure drop along the dense medium cyclone axis is high, and this develops an air core along the dense medium cyclone axis. The air core inside the dense medium cyclone initially reduces the rate of underflow discharge and thereafter reduces the flow rate of the feed slurry in the dense medium cyclone. Therefore, feed slurry flow rate to the dense medium cyclone cannot be increased to a great extent by increasing the feed inlet pressure (Mukherjee 2002: 264).

For each laboratory test, separation efficiency ($S_E$) is calculated following the method as given below:

$$S_E = \frac{(\alpha - \nu) \cdot (\nu - u)}{\alpha \cdot (1 - u - \nu)}$$  (6.11)

In the plant, the $V_F$ diameter was less than the $V_F$ diameter required at increased feed inlet pressure (6.4 psi). Therefore, one can observe pressure drop along the axis of the dense medium cyclone. This pressure drop would adversely affect the separation process and with increase in feed inlet pressure the $V_F$ diameter should be increased.

The separation efficiency decreases with increase in spigot diameter. According to Mukherjee et al. (2002: 264) larger spigot diameter would neither effect magnitude nor the shape of the tangential velocity profile in the cyclone; it only allows more slurry to flow through the spigot. Using the authors’ own words ‘at larger spigot diameter, effect of pressure drop on separation would be less and there would be less flow reversal. Therefore, at larger spigot diameter (15 mm), solid flow rate to the dense medium cyclone underflow increased and, hence, separation efficiency was less as compared to the optimum (10 mm) spigot diameter’ (ibid.).

As high feed inlet pressure had enhanced the classification of magnetite particles in the dense medium cyclone, high feed inlet pressure increases the pressure drop inside the

---

21 ‘At higher slurry flow rate, the tangential velocity of the slurry in the dense medium cyclone will improve separation of fine coal and NGM, thereby clean coal yield would increase’ (Mukharjee 2002: 263).
dense medium cyclone, and it facilitates the segregation of media. Thus, the relative-density
differential of the media increases with an increase in feed inlet pressure (Fig. 6.6). In
industry scale conditions, similar changes in media behavior would initially effect coal output
and then the ash content in the clean coal due to the misplacement of coal particles (comp.

![Fig. 6.6. Effect of feed inlet pressure on relative density differential (Mukherjee, 2003)](image)

As reported in the literature, at high feed inlet pressure, tangential velocity of the coal
particles increased, and this had improved sharpness of separation significantly for fine (-3.0
+ 0.5 mm) size coal particles. Near-gravity present in the feed is high.  

6.4. Geometry of cyclones and their efficiency

As pointed out in the paragraphs above, another factor influencing the cyclones
productivity is the spigot diameter.

---

22 Mukherjee et al. (2002) shortly comment on the statistical techniques for the plant trial data analysis. For the
rather rough and concise form of the comment we would like to allow ourselves to quote it in full. We are placing
the fragment in the footnote as it does not have a direct input to the subject of our paper. The authors say: 'process
performance of streams 1 and 2 were compared on the basis of a few samples collected from the plant. In the
plant, process variables are many and their fluctuations are common. Considering the effect of fluctuation in
the process variables on the clean coal yield, the performance of the streams 1 and 2 should not be compared only
on the basis of few data points. Therefore, a robust statistical method was used for comparing the performance of the
dense medium cyclones from streams 1 and 2. Percentage ash in the feed, clean coal and rejects of the dense
medium cyclones were collected against each data points from streams 1 and 2. The coarse clean coal yield
corresponding to each data point was calculated through ash balance method. The process performance of the
dense medium cyclones from streams 1 and 2 are compared on the basis of clean coal yield through detailed plant
data analysis using advanced statistical techniques. Noisy data sets were removed using the outlier analysis. Data
sets were then compared on the basis of comparison of regression line method to find out the difference in clean
coal yield between the dense medium cyclone from streams 1 and 2. The method involves three major steps:
establishing simple linear relationship between clean coal yield and feed ash using data sets collected from
streams 1and 2, fitting of the regression lines using a pooled/mean gradient and then comparing the regression
lines from the mean separation value' (Mukherjee, 2002: 268)
Upadrashta and Venkateswarlu (1982) related slurry flow rate with spigot diameter and vortex finder diameter as follows:

$$Q \propto (D_s^3 + D_v^3)^n$$  \hspace{1cm} (6.12)

with \( n = 0.30 \) during spray discharge and \( n = 0.26 \) during rope discharge. According to Upadrashta and Venkateswarlu the nature of the influence of \( D_s \) and \( D_v \) on \( Q \) changes slightly with the change in flow type from spray to rope discharge.

Further, the volume split is related to the spigot and vortex diameters as follows:

$$\frac{Q_{sp}}{Q_{v}} \propto \left( \frac{D_s}{D_v} \right)^n$$  \hspace{1cm} (6.13)

The values of the exponent \( n \) obtained by various sources in the literature are given in the table underneath. Upadrashta and Venkateswarlu (1982) reported the effect of \( D(u) \) on \( S \) to be the same during spray and rope discharges. It should be noted that it is quite surprising that considering the fact that there is an air core at the sinks during spray discharge and none during roping discharge.

**Table. 6.1.** The values of the exponent \( n \) obtained by various authors.

<table>
<thead>
<tr>
<th>Source</th>
<th>Exponent n</th>
</tr>
</thead>
<tbody>
<tr>
<td>Stas (1957)</td>
<td>3.5 (the cyclone operated with water only)</td>
</tr>
<tr>
<td>Upadrashta and Venkateswarlu (1982)</td>
<td>2.32 (valid for spraying and roping conditions)</td>
</tr>
</tbody>
</table>

Spigot diameter generally has a small but significant influence on \( Q \). However, \( D(u) \) does have a very strong influence on the slurry split to the sinks as shown in the table. In fact, it is expected that the spigot capacity should be determined mainly by the dimensions of the spigot diameter.

England et al. (2002) elaborates on spigot capacity basing, among many, on the data from Dutch State Mines. The relationship between spigot diameter and spigot capacity obtained by the original developers of the dense medium cyclone in Dutch State Mines is illustrated in Figure 6.7. This figure shows the spigot capacities for dense medium cyclones beneficiating coal whereby the medium was magnetite. The cyclone was operated at a head of 9D.
A regression of the data in this figure gives the following relationship:

\[
Q_{UM} = (0.9 \cdot 10^{-3}) \cdot D_u^{1.556}
\]  

(6.14)

with \( D_u \) in mm and \( Q_{UM} \) (ore only) in m\(^3\)/h \((R^2 = 0.9944)\). The spigot capacity for dense medium cyclones appears to depend mainly on the cross-sectional area of the spigot. As previously illustrated, it is thought that spigot capacities were determined under conditions in which roping was not prevalent at the sinks stream. England et al. (2002) reported spigot capacities that were equivalent to those specified by Dutch State Mines; these are shown in the next figure 6.8.

The relationship between spigot diameter and spigot capacity for individual minerals and ores at various feed pressures are shown in the figure underneath. Regression analysis of the data in this figure yielded the following expression:
\[ Q_{\text{UN}} = (9.9 \cdot 10^{-8})D_{H}^{1.64} \]  

(6.15)

with \( H \) being the head diameter and \( D \) the cyclone diameter and \( R^2 = 0.9911^{23} \).

As far as the vortex finder is concerned it has been reported that it also influences \( Q \) in accordance with the following relationship:

\[ Q = Q_{s2} + Q_{w2} \propto D_{o}^{n} \]  

(6.16)

As mentioned previously, Upadrashta and Venkateswarlu (1982) related \( Q \) with spigot diameter and vortex finder diameter. From the theoretical point of view this relation can be described as follows:

\[ Q \propto (D_{s2}^{n} + D_{w2}^{n})^{n} \]  

(6.17)

with \( n = 0.30 \) during spray discharge and \( n = 0.26 \) during roping discharge.

It is, therefore, expected that \( Q \) should increase with decreasing \( D_{o} \). With all other conditions being the same, \( D_{o} \) strongly increases the slurry flow through the vortex finder at the expense of slurry flow at the sinks. Like Vergese and Rao (1994), Restarick and Krnic

\[ ^{23} \text{The expression is consistent with the equation above.} \]
(1991) observed $Q$ to increase with the vortex finder diameter, although they did not quantify the effect.

6.5. Medium and feed grade

Finally, we would like to briefly report what is said about the performance of such values as feed, ore and media for the efficiency of a dense medium cyclone. First of all, as far as the particle size is concerned, finer medium grades tend to be more viscous than coarser grades. He and Laskowski (1995) reported the cyclone flow-rate, when operated with medium only, to be the highest when operating with the coarsest medium. The split was unaffected by changes in the medium grade. Assuming that these trends would still hold at low medium to ore ratios, then the spigot capacity should increase when the medium particles are relatively coarse.

Medium segregation within dense medium cyclone is dependent on medium grade, and coarser grades tend to segregate more. Little is said in the literature, however, about the relationship between cyclone flow-rate and particle size. Cohen and Isherwood (1960) reported that the sinks slurry density obtained when the spigot was overloaded is dependent on the specific gravity and packing characteristics of the sinks solids. And the packing characteristics of solids are dependent on the particle size and size distribution.
7. **Computational Fluid Dynamics (CFD).**

**Description and examples of application**

The described devices make use of fluids as the medium for classification of the raw material. Such application requires considering the behaviour of the material in contact with various fluids and also the way the latter are introduced into a device. As an example we may refer to what Drzymała (2002) writes about fluidizing classification. The author says that ‘in fluidizing classifiers (elutriators) (...) the rule of separation (...) is simple. For ideal separation the particles whose sedimentation velocity is higher than the velocity of up going medium flow are carried by the medium to the overflow, while a particle having lower sedimentation velocity is directed to the overflow’ (Drzymała 2002: 175). It has been stated that our aim is to trace the application of fluids in dense medium cyclones with the aid of CFD. The topic finds its value in the fact that describing the behaviour of fluids in cyclones may help to manipulate the hydrodynamics to achieve the desired classification. As it has been already described, in a typical dense-medium cyclone mixture of medium and raw coal enters tangentially near the top of the cylindrical section, thus forming a strong swirling flow. According to Narasimha et al. (2006) there is an extensive literature on the performance of dense medium separation processes though very few have successfully introduced effective mathematical models of processes for simulation, other than the trivial option of using partition curves with arbitrary parameter selection, which for the authors appeared to be trivial (comp. Narasimha 2006: 42).

Computational study has been carried by various institutions and described by many authors, among whom Wang et al. (2011) have lately stated that, when it comes to dense medium cyclones, the analysis ‘is very complicated with the presence of swirling turbulence, an air-core and segregation of the medium and coal particles. It involves multiple phases: air, water, coal and magnetite particles of different sizes, densities and other properties’ (ibid.: 20; comp. Chu 2012). In recent years CFD simulations were successfully applied for the analysis of dense-medium cyclones’ operations in various plants (Suresh et al. 2010). Successful analyses have been conducted resulting in practical applications of CFD modelling, which this chapter tries to exemplify.

At this moment we would like to present a description of the model before moving on to its application. Chu (2009) relates CFD with another type of analysis which is **Discrete Element Method**. The Discrete Element Method is used to model the motion of discrete
particles by applying Newton’s laws of motion, while CFD is used to model the motion of suspension medium by numerically solving the local-averaged Navier–Stokes equations facilitated with the Volume of Fluid and mixture multiphase flow models. The used nomenclature is as presented on the beginning of this work.

7.1. Computational fluid dynamics. Description

Slack et al. (2003) claim that CFD would provide the capability to develop ‘designer cyclones’ optimized for the specific operating conditions of a particular process. Thus, as other writers estimate, it is readily apparent that the CFD tool is much easier to use today than in the past (Degadillo and Rahjamani 2007: 253). The authors report that in the description of the fluid dynamics of a cyclone, the key component is the turbulence closure model.

A number of models including the \( \kappa-\varepsilon \) model (Hargreves and Silvesters, 1990; Dyakowski and Williams, 1992; Malhotra et al., 1994; He et al., 1999) and the Reynolds stress model (Lu et al., 1999; Slack et al., 2000; Cullivan et al., 2004; comp. Furman and Stęgowski 2011) have been tried in the recent days. Now, with the aid of three-dimensional computations, the mining industry can avoid assumptions provided by a two-dimensional analysis. According to scholars (e.g. Derksen and Van den Akker, 2000; Slack et al., 2000; Delgadillo and Rajamani, 2005 and 2007) large-eddy simulation is a leading candidate for the description of turbulence.

Using words of Narasimha et al. (2006) ‘Computational Fluid Dynamics (CFD) is a versatile means to predict velocity profiles under a wide range of design and operating conditions’ (ibid.: 42). Delgadillo and Rajamani (2005a) compared renormalization group (RNG) \( \kappa-\varepsilon \) model, the Reynolds stress model and the large-eddy simulation model (henceforth LES). They showed that LES leads in the description of mass balance, velocity profile, air-core dimension and even root-mean-square velocity prediction. Later the same authors showed that LES predicts measured tangential and axial velocity profiles in 75-mm and 250-mm hydrocyclones. In LES, velocity profiles are resolved by a filtering operation of the velocity field, and the smaller scales are modelled with the eddy viscosity model. In the authors’ own words ‘the eddies up to the size of the computational mesh are solved and the eddies below the mesh size are modelled’ (Degadillo and Rajamani 2007: 253). Therefore, the velocity field is defined as the sum of the filtered velocity \( \bar{u}_i \) and the residual component \( u'_i \).

\[
\bar{u}_i = \bar{u}_{i\text{f}} + u'_{i\text{f}}
\]  

(7.1)

From this transformation, a new term arises, containing the stress tensor of the residual motions \( \tau_{ij}^{\text{res}} \) (Mainza et al., 2006: 1050) The first equation in this chapter is applied to the governing Navier–Stokes equations, which, as scholars observe, are the ‘back-bone’ of CFD
modelling (Narasimha et al., 2006: 43). The filtered Navier–Stokes equations are given in Equations. (7.2) and (7.3).

\[
\frac{\partial \boldsymbol{\Pi}}{\partial t} + \mathbf{g} \cdot \frac{\partial \boldsymbol{\Pi}}{\partial \mathbf{x}} = 0
\]  
\hspace{1cm} (7.2)

\[
\frac{\partial \mathbf{u}_t}{\partial t} + \frac{\partial (\mathbf{u}_t \cdot \mathbf{u}_t)}{\partial x} = -\frac{1}{\rho} \frac{\partial p}{\partial x} + \frac{\partial}{\partial x_j} \left( \mu \frac{\partial \mathbf{u}_t}{\partial x_j} \right) - \frac{\partial \tau_{ij}^{SS}}{\partial x_j} + \mathbf{g}
\]  
\hspace{1cm} (7.3)

The residual stress tensor \( \tau_{ij}^{SS} \) is modeled as the product of the eddy viscosity and the strain rate which is illustrated further on in Equation (7.4). The eddy viscosity is resolved using the RNG model. According to Degadillo and Rajamani (2007), who report after Yakhot et al. (1989), the RNG model is very effective to model the transitional flows and near wall regions where the molecular viscosity has more significance:

\[
\tau_{ij}^{SS} = -\mu_e \cdot \left( \frac{\partial \mathbf{u}_t}{\partial x_j} + \frac{\partial \mathbf{u}_t}{\partial x_i} \right)
\]  
\hspace{1cm} (7.4)

The turbulent viscosity \( \mu_t \) is defined as the effective turbulent viscosity \( \mu_e \) or molecular viscosity \( \mu \) as shown underneath:

\[
\mu_t = \mu \cdot \sqrt{1 + H(x) \cdot \left( \frac{\mu_e^2 \cdot \mu_{eff}}{\mu^2} - 100 \right)}
\]  
\hspace{1cm} (7.5)

where the effective viscosity \( (\mu_{eff}) \) is defined as:

\[
\mu_{eff} = \mu_s \cdot \sqrt[3]{1 + H(x)}
\]  
\hspace{1cm} (7.6)

\( H(x) \) is the Heaviside function, defined as \( H(x) = x \) for \( x \geq 0 \) and 0 for \( x \leq 0 \). The turbulent viscosity \( (\mu_t) \) depends on the strain rate defined as:

\[
\mu_t = \left( \mathbf{C}_{\text{RNG}} \cdot \mathbf{\bar{S}}^t \cdot \mathbf{\bar{S}}^t \right) \cdot \sqrt{2 \cdot \mathbf{S}^t \cdot \mathbf{\bar{S}}^t}
\]  
\hspace{1cm} (7.7)

In the last equation, provided again by Degadillo and Rajamani (2007), \( V \) is the volume of the computational cell and \( \mathbf{\bar{S}}^t \) is the filtered strain rate. As the authors report, the tangential acceleration applied to the flow creates a high centrifugal force that pushes the fluid to the wall, creating a low pressure in the central axis. The low pressure in the core of the
cyclone provides conditions to force the air into the system, forming a cone of air. In the quoted literature the surface is modelled with the volume-of-fluid model whose model equations are shown below. It tracks the location of such a surface at every time step.

\[
\frac{\partial \alpha_p}{\partial t} + \alpha \cdot \frac{\partial \mathbf{u}_f}{\partial x} = 0
\]  

\[ (7.8) \]

\[
\frac{\partial (\rho \mathbf{u}_p)}{\partial t} + \frac{\partial (\rho \mathbf{u}_p \mathbf{u}_f)}{\partial x} = -\frac{\partial \rho_f}{\partial x} \left( \mathbf{u} \cdot \left( \frac{\partial \mathbf{u}_f}{\partial x} + \frac{\partial \mathbf{u}_p}{\partial x} \right) \right) + \mathbf{g} \cdot \mathbf{g}_f
\]  

\[ (7.9) \]

In the air-core region, the volume fraction \( \alpha_p \) (which ranges from 0 to 1) is nearly 0. Particles are classified due to the drag force and diffusion of particles due to the fluid turbulence. Therefore, additional model used in the prediction of the hydrocyclone operations is the particle trajectory prediction. The trajectory of particles is computed by Lagrangian tracking of particles upon the Eulerian continuous-phase predictions. The balance between the drag, gravity, buoyancy and centrifugal forces determines the trajectory of the particles in the continuous phase. This fact is depicted in the equations underneath (Degadillo and Rajamani 2007: 253–254):

\[
\frac{\partial \mathbf{u}_p}{\partial t} = F_D \cdot (\mathbf{u} - \mathbf{u}_p) + \frac{\rho_f}{\rho_p} \cdot \mathbf{g} \cdot \mathbf{g}_p
\]  

\[ (7.10) \]

\[
F_D = \frac{18 \mu}{Q_p \cdot d_p^2} \cdot \frac{C_D \cdot R \rho}{24}
\]  

\[ (7.11) \]

\[
R \rho = \frac{\rho \cdot d_p \cdot |\mathbf{u}_p - \mathbf{u}|}{\mu}
\]  

\[ (7.12) \]

where:
- \( \mathbf{u}_p \) – the particle velocity,
- \( \mathbf{u} \) – the fluid velocity,
- \( \rho_f \) – the fluid density,
- \( \rho_p \) – the density of the particle,
- \( F_D \) – the drag force,
- \( d_p \) – the particle diameter,
- \( C_D \) – the drag coefficient.

### 7.2. CFD and other models
Chu et al. (2009), as we said, related CFD with DEM, while Furman and Stęgowski (2011) have recently compared the use of CFD with residence time distribution method (henceforth RTD). The concept of residence time distribution arose from tracer experiments and was to be used to quantify mixing. The RTD concept was described in a many scientific papers in the past century (e.g. Danckverts, 1953; Levenspiel, 1999). The list of methods for the exploitation of experimental RTD’s includes:

- analysis in terms of statistical moments that yields some global parameters: mean residence time (MRT) and standard deviation,
- reactor flaws detection such as dead zones and bypasses,
- adjustment of a model for flow parameters.

Furman and Stęgowski (2011) note that more fundamental information about mixing can be obtained through numerical simulation tools (such as CFD) as it can provide efficient means to obtain the RTD. However, factors such as inappropriate residence time distribution sampling and numerical diffusions can lead to considerable variations in the residence time distribution prediction. Therefore, the authors say that to ensure acceptable accuracy of the residence time distribution prediction by CFD, comparison with experimental residence time distribution is essential.

CFD software uses physical laws which have to be supplemented by models of turbulence and approximations. Simulations require detailed knowledge of the process itself and of fluid dynamics to decide on the suitable CFD model and to analyse results24. The authors state that for a deep investigation of the flow structure, experimental validation by residence time distribution is not sufficient, because of the averaged nature of this characteristic (different systems can produce the same residence time distribution). Experimental validation of the numerical residence time distribution is more convenient than experimental validation of the complete flow obtained numerically, hence – as Furman and Stęgowski (2011) say – it is pretty popular. In the paper often quoted here, *CFD models of jet mixing and their validation by tracer experiments*, Furman and Stęgowski (2011) add a comment that ‘by now, however, no framework has been established in the literature for evaluating the flow introduced by CFD. Numerous experimental and CFD studies on residence time distribution in tanks have been carried out and published over the years. The papers frequently conclude with the sentence: Experimental residence time distribution study has allowed us to validate CFD simulations’ (Furman and Stęgowski, 2011: 301).

---

24 According to Furman and Stęgowski (2011) experiments are still necessary in order to validate simulations. There are attempts to employ gamma ray computer tomography (CT) or Computer Assisted Radioactive Particle Tracking (CARPT), at least on lab-scale. The role of experimental validation is not questioned. However, there are different views as far as the RTD–CFD relation is concerned. It is said for instance that ‘the RTD predictions allow a rather painless way to validate a complex flow simulation’ (Furman and Stęgowski 2011: 301).
The figure underneath shows comparison of experimental and calculated residence time distribution functions for the models studied. The results of residence time distribution analysis showing the calculated mean residence time and variance are presented in Table 7.1.

![Fig. 7.1. Experimental and simulated RTD functions (Furman, Stęgowski, 2011)](image)

<table>
<thead>
<tr>
<th>Model name</th>
<th>Mean residence time MRT [s]</th>
<th>Standard deviation $\sigma$ [s]</th>
</tr>
</thead>
<tbody>
<tr>
<td>Standard $k-\varepsilon$</td>
<td>329</td>
<td>190</td>
</tr>
<tr>
<td>RSM</td>
<td>327</td>
<td>228</td>
</tr>
<tr>
<td>RNG $k-\varepsilon$</td>
<td>360</td>
<td>214</td>
</tr>
<tr>
<td>Experimental RTD</td>
<td>363</td>
<td>216</td>
</tr>
<tr>
<td>V/Q</td>
<td>356</td>
<td>—</td>
</tr>
</tbody>
</table>

The most popular in literature standard $k-\varepsilon$ turbulence model produces the results that can be accepted.

The following paragraphs refer what Furaman and Stęgowski (2011) noted in their study. According to the authors the question arises which model of the proposed ones – CFD or residence time distribution – reproduces the flow field more accurately. Flow visualization studies (Fig. 7.2) showed phenomena related to a ‘long tail’ on the residence time distribution.
The four frames show the progress of the dye at 8, 16, 24 and 100 s after injection. Two flow structures are formed in the tank: a recirculation flow in the lower part of the tank induced by the water jet, and low velocity and turbulence fields in the upper section of the tank. Also a small stagnant zone at the bottom of the reactor can be seen. The predicted flow fields in the reactor are presented in Figures 7.3 and 7.4.
Figure 7.3 shows the trajectories of tracers released at the inlet. Four time frames were generated by RNG $k$–$\varepsilon$ model. Qualitative analysis reveals the presence of back-mixing streams at the bottom and generally the acceptable convergence between the simulation and the flow visualization pictures (Fig. 7.2). Figure 7.4 shows the velocity vectors on three horizontal planes located 10, 25 and 40 cm from the bottom of the tank, for all the models. The examination of the velocity vectors reveals two, symmetric, down-flow patterns in the lower part of the reactor. If one assumes that a flow field that properly accounts for residence time distribution function has been established then the detailed comparison of velocity profiles can be performed. Figure 7.5 shows velocity profiles (vertical velocity components) in the reactor at different levels on the plane perpendicular to the flow visualization plane.

One should note that negative values indicate the change in the direction of the flow, from upward to downward in the axi-symmetrical region depicted in Figure 7.4. By studying these results, one can draw the following three conclusions about the flow structure:

1. The flow field is dominated by circulation loops – the main macro mixing mechanism. This phenomenon is observed basically in the lower part of the reactor;
2. RSM model overpredicts the velocity in the middle of the tank. Moreover, it indicates a downward stream also in the upper part that was not confirmed experimentally. Note that even qualitative analysis of residence time distribution function (Fig. 7.1) might exclude RSM model from further consideration;
3. Both $k$–$\varepsilon$ models predict mixing fairly well. They were successfully fitted with the experimental residence time distribution. However, the maximum difference in the predicted velocities of almost 25% in the upper plane of the reactor may be unsatisfactory considering reactor optimization. Furman (2011) notes that such discrepancy may have significant impact on chemical reactions.
Fig 7.5. Velocity profiles in the reactor at different levels on the plane perpendicular to the flow visualization plane (along the diameter of the tank). Average vertical components of velocity vs. distance from axis, the lower plot at 0.1 m, the middle plot at 0.25 m and the upper plot at 0.4 m from the bottom of the reactor (Furman, Stęgowski, 2011)

The CFD models predicted different flow patterns. It is evident, that the experimental residence time distribution can be used to identify unacceptable predictions. One should add that successful application of CFD models with experimental data requires some experience And that experienced researchers may take the risk and confirm the validity of CFD simulation results by analytical residence time distribution. In this case Furman and Stęgowski (2011) suggested what follows in verifying the CFD results:

1. Choose appropriate CFD models\(^{25}\);
2. Carry out the flow field and residence time distribution computation with CFD;

\(^{25}\) ‘Appropriate’ from the physical point of view.
3. Evaluate the convergence between the predicted residence time distribution and the experimental data. If poor, return to step 1 and change the model;
4. Inspect the flow fields (e.g. velocity profiles). If different, design an experiment to resolve.

The authors are of the opinion that one should look for the reactors and processes with the best flow pattern for implementation of eco-friendly chemistry. Unfortunately, common flow systems are opaque and the presented optical flow visualization does not work. As experimentation on larger size systems and application of new technologies (such as CARPT and CT) is avoided as too costly the authors say that ‘additional experiment is recommended to resolve the validation problem’ (*ibid*: 303).

Furman and Stęgowski in their publications remind that numerical models are used also in radiotracers. By means of a radiotracer test one may determine the mixing intensity, flow rate and separation coefficients. The experimental study (analysed as an illustration of the method) is a tracer experiment carried out on a dewatering system. The experimental results as well as the models and their parameters obtained from the advanced data analysis enable a detailed analysis of the solid state behaviour inside the dewatering subsystems (comp. Stęgowski et al., 2004: 43).

Apart from the work of the mentioned scholars CFD studies on cyclone separators have been also reviewed in details by Brennan et al. (2003), Suasnabar (2000) and also by Slack et al. (2000).

The CFD study was carried out by Hsieh (1988) with further publications by Hsieh and Rajamani (1991) and Monredon et al. (1992). These studies used 2D-axisymmetric grids with an imposed air-core position. The authors demonstrated the effect of swirl and flow reversal on turbulence behaviour. It should be noted here that there are extensive studies on how to apply the data and calculations to re-design dense medium cyclones and such trials were collected, among many, by Napeir-Munn et al. (1995), Rayner and Napier-Munn (2000, 2003), Rao et al. (2003: 447–452), Pianko-Oprych and Jaworski (2009). What we are interested in are those of the calculations that were based on the application of FLUENT and made use of CFD modelling.

7.3. Mathematical preassumptions of CFD and its application

The more complex Reynolds Stress Turbulence model (Launder et al., 1975), which solves a set of transport equations for each individual component in the Reynolds Stress tensor, is recognized as being more suitable and has been used successfully in recent CFD studies of cyclones (Brennan et al., 2003, Suasnabar, 2000 and Slack et al., 2000). Narasimha et al. (2006) refer to these studies, plus add their own investigation on the data from Dutch
State Mines where numerical simulations based on using FLUENT 6.2 multi-phase models with RSM and LES turbulence models.

According to the mentioned studies in multi-phase simulations the predicted air-core shape and diameter were found to be close to the experimental results measured by gamma ray tomography. As Narasimha et al. (2006) estimate the LES turbulence model together with ASM multi-phase model can be used to predict the air-suspension surface accurately (within 2% error). Multi-phase simulations (air-water-medium) showed appropriate medium segregation effects but over-predicted the level of segregation compared to that measured by gamma ray tomography near the wall. The predictions of accurate axial segregation of magnetite medium investigated using the LES turbulence model together with the multi-phase mixture model and viscosity corrections according to the feed particle loading factor. The effect of size distribution of the magnetite has been fully studied. The ultrafine magnetite sizes (i.e. 2 and 7 microns) are distributed uniformly throughout the cyclone. As the size of magnetite increases, more segregation of magnetite occurs close to the wall. Narasimha et al. (2006) concluded that the excessive underflow volumetric flow rates are responsible for under prediction of the overflow density26.

The computational cost of the simulations is considerably high. Most authors note that the standard k-ε turbulence model is unsuitable for the highly swirling flows that exist in DMCs.

Hargreves and Silvesters (1990), Dyakowski and Williams (1996) and Malhotra et al. (1994) applied the k-ε model for modelling turbulences. Others have modified the dissipation equation in the standard k-ε model with a swirl correction (Fraser et al., 1997; He et al., 1999; Suasnabar, 2000; Schuetz et al., 2004; Narasimha et al., 2005). This modified k-ε model is implemented in Fluent as an adjustable constant in the RNG turbulence model (Narasimha 2006: 44). The standard k-ε model is a semi-empirical model based on model transport equations for the turbulent kinetic energy (k) and its dissipation rate (ε), and are given by

\[
\frac{\partial (\rho u_i k)}{\partial t} + \frac{\partial (\rho u_i u_j k \delta_{ij})}{\partial x_j} = \frac{\partial}{\partial x_j} \left[ \left( \mu + \frac{\mu_t}{k} \right) \frac{\partial k}{\partial x_j} \right] + G_k - \rho \cdot S
\]

\[
\frac{\partial (\rho u_i \varepsilon)}{\partial t} + \frac{\partial (\rho u_i u_j \varepsilon \delta_{ij})}{\partial x_j} = \frac{\partial}{\partial x_j} \left[ \left( \mu + \frac{\mu_t}{k} \right) \frac{\partial \varepsilon}{\partial x_j} \right] + C_1 \frac{\varepsilon}{k} \left( \frac{\partial k}{\partial x_j} \right) - C_2 \varepsilon \frac{\rho \cdot S}{k} + \frac{\partial}{\partial x_j} \left( \frac{\varepsilon \rho u_j}{k} \right)
\]

\[
G_k = -\rho \cdot u_i \cdot \frac{\partial u_j}{\partial x_j}
\]

26 The authors are further investigating modifications to the basic ASM model to incorporate turbulent mixing of the medium (ibid.: 54)

112
As Narasimha et al. (2005) explain, in these equations, $G_k$ represents the generation of turbulent kinetic energy due to the mean velocity gradients, $C_{1\varepsilon}$ and $C_{2\varepsilon}$ are constants. $\sigma_k$ and $\sigma_\varepsilon$ are the turbulent Prandtl numbers for $k$ and $\varepsilon$, respectively. The ‘eddy’ or turbulent viscosity, is defined in Equations (7.16) and (7.17):

$$\frac{\partial(g \cdot \bar{v})}{\partial t} + \nabla \cdot (g \cdot \bar{v}) = -\nabla p + \nabla \cdot (\tau) + g \cdot \bar{g}$$  \hspace{1cm} (7.16)

$$\tau = \mu_{\text{eff}} \left[ (\bar{v} \cdot \nabla) - \frac{2}{3} \nabla \cdot \bar{v} \right]$$  \hspace{1cm} (7.17)

can be computed by combining $k$ and $\varepsilon$ as follows:

$$\frac{\partial \Pi_k}{\partial t} + \frac{\partial \Pi_{\langle v \rangle}}{\partial x_f} = -\frac{1}{g} \frac{\partial \Pi}{\partial x_g} + \frac{\partial}{\partial x_f} \left( \mu_{\text{eff}} \frac{\partial \Pi_k}{\partial x_g} \right) - \frac{\partial \tau_{\text{eff}}^{\text{eddy}}}{\partial x_f} + g_t$$  \hspace{1cm} (7.18)

As Narasimha (2006) mentions, Suasnabar (2000) subsequently used a parametrically modified $k-\varepsilon$ model, though this model had some limitations in application of high swirling flows. Suasnabar (2000) used the RSM model developed by Launder et al. (1975) as well Hanjelic and Launder (1972)\(^{27}\). Slack et al. (2000) demonstrated the DSM to perform well in comparison to LES without high computational costs.

According to Narasimha et al. (2007) the computationally more expensive LES provides the best solution for capture of time dependent vortex oscillations and non-equilibrium turbulence, which potentially will affect the separation efficiency. Separate models can be formulated for the medium and coal particles. However, the models must interact with each other to produce an overall model for dense-medium cyclones.

The segregation of magnetite media in dense-medium cyclones can be predicted by two different approaches of multi-phase models (comp. Wang et al., 2011: 20–21). In simplest case, the magnetite can be modelled using ASM multi-phase model considering magnetite with average size (30 microns) as a separate phase apart from water and air. While in a complex case, the magnetite can be modelled with feed size distribution – six size fractions (Narasimha et al., 2007: 44). The continuity and momentum equations in their general form are (these equations are analogous to presented in previous subparagraph):

$$\frac{\partial(g \cdot \bar{v})}{\partial t} + \nabla \cdot (g \cdot \bar{v}) = 0$$  \hspace{1cm} (7.19)

\(^{27}\) The RSM model is recognized as being more suited to swirling flows, however the six additional transport equations makes this model computationally expensive (Wang et al., 2011: 20).
The turbulence modelling is being done here by both RSM and LES in order to capture the high swirling flow patterns in dense medium cyclones.

LES turbulence model was used to predict the cone of air and water splits in cyclone. The air-core surface has been modelled using volume of fluid method. In this model the velocity is decomposed in a different manner than in the $k–\varepsilon$ and the Reynolds stress model. In LES it is resolved by a filtering operation of the velocity field and the smaller scales or residuals are modelled in a particular manner. A definition for the velocity, defined as the sum of a resolved component $u_{\bar{}}_i$ and a residual component $u_{\bar{}}_i$ is given by

$$u_i = \bar{u}_i + \bar{u}'_i$$

(7.23)

In order to get the resolved component a filtering operation is applied to the governing Equations (7.20) and (7.21). These equations are of the standard form, containing the residual stress tensor $\tau_{ij}^{\text{res}}$ that arises from the residual motions. The final LES model equations would be:

$$\frac{\partial \bar{u}}{\partial t} + \bar{u} \cdot \frac{\partial \bar{u}}{\partial x_j} = - \frac{1}{\rho} \frac{\partial p}{\partial x_j} + \frac{1}{2} \frac{\partial \tau_{ij}^{\text{res}}}{\partial x_j}$$

(7.24)

$$\frac{\partial \tau_{ij}^{\text{res}}}{\partial x_j} = - \frac{1}{\rho} \frac{\partial \tau_{ij}^{\text{res}}}{\partial x_j} - \frac{1}{\rho} \frac{\partial \tau_{ij}^{\text{res}}}{\partial x_j} + \bar{u} \cdot \frac{\partial \bar{u}}{\partial x_j}$$

(7.25)

The residual stress tensor $\tau_{ij}^{\text{res}}$ contains all information of the sub-grid scales or residual and is defined as:

$$\tau_{ij}^{\text{res}} = \bar{u}_i \cdot \bar{u}_j - \bar{u}_i \cdot \bar{u}_j$$

(7.26)

The stress tensor is defined as the product of the eddy viscosity and the strain rate as:

$$\tau_{ij}^{\text{eddy}} = - \bar{u}_i \cdot \left( \frac{\partial \bar{u}_i}{\partial x_j} + \frac{\partial \bar{u}_j}{\partial x_i} \right)$$

(7.27)
The turbulent viscosity is defined as the difference between the effective viscosity and the molecular viscosity:

\[ \mu_{eq} = \mu_{eff} - \mu \]  

(7.28)

And the effective viscosity is defined as follows:

\[ \mu_{eff} = \mu \cdot \left( 1 + H \cdot \left( \frac{\mu_{eq}^{2} \cdot \rho_{eq}^{3}}{\mu^{3}} - 100 \right) \right) \]  

(7.29)

The turbulent viscosity \( \mu_s \) in the sub-grid scale is defined as:

\[ \mu_s = \left( C_{RNG} \cdot \nabla \cdot \right)^{2} \cdot \sqrt{2 \cdot \frac{\gamma_{ij}}{\psi} \cdot \frac{\psi_{ij}}{\psi}} \]  

(7.30)

\( H(x) \) is defined in this way:

\[ H(x) = \{ \begin{array}{ll} x & x \geq 0 \\ 0 & x < 0 \end{array} \} \]  

(7.31)

Initially, the magnetite in the feed was represented by a single average size of 30 micron and modelled using the ASM (Algebraic Slip Mixture) model, and the Raynolds stress model (quadratic pressure–strain correlations) turbulence model, along with water and air as separate phases.

The table 7.2 shows the predicted and the measured medium densities for feed with relative density of 1.245 as the preliminary studies of 350 mm dense medium cyclone.

**Table 7.2.** Flow densities predicted for 350 mm DSM cyclone compared to measured densities (kg/m³) (Narasimha, 2007)

<table>
<thead>
<tr>
<th>Density</th>
<th>At feed RD 1.245</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>measured</td>
</tr>
<tr>
<td>Overflow</td>
<td>1 151</td>
</tr>
<tr>
<td>Underflow</td>
<td>1 836</td>
</tr>
</tbody>
</table>

As Narasimha et al. (2007) estimate these results show that the CFD is predicting the general features of the medium segregation qualitatively with average single magnetite size in the feed with RSM turbulence mixture models (comp. Chu, 2009: 238–244). Using Narasimha et al. (2006) own words, it is expected that there will be an improvement with consideration of complete size distribution of magnetite in the feed using mixture model. In
dense-medium cyclone’s operations, it is believed that the rheology of magnetite is a predominant factor in the stability of media. ‘To account for the particle packing effect on the segregation of magnetite in the cyclone, the viscosity of the suspension is modified with magnetite feed particle loading factor’ (Narasimha et al., 2007: 49). Two viscosity correction models were identified in the literature based on the feed particle loading range for the magnetite medium suspension.

The Ishii and Mishima (1984) viscosity correction model is straightforward, for the general suspension viscosity, calculated from

\[
\frac{\mu_{\text{mix}}}{\mu_s} = \left(1 - \frac{\alpha_d}{0.62}\right)^{-L_d}\tag{7.32}
\]

where \(\alpha_d\) is dispersed phase void fraction.

Another one is the Rajamani and Milin (1992) viscosity correction model, which is based on Thomas expression for the suspension viscosity. The suspension viscosity in this case is calculated from

\[
\frac{\mu_{\text{mix}}}{\mu_s} = 1 + 2.5 \cdot \alpha_d + 10.05 \cdot \alpha_d^2 + 0.00273 \cdot e^{466 - \alpha_d}\tag{7.33}
\]

Simulations have been carried out with these two viscosity corrections using a user define interpreted function with magnetite size distribution consideration in **ASM model**. According to Narasimha et al. (2006) from the Raynolds stress model simulations it is observed that by modelling the medium, as separate phases (size distribution) instead of single average phase in the ASM multi-phase model, the medium segregation predictions are improved.

When it comes to prediction of magnetite segregation using LES turbulence and ASM multi-phase models, one obvious improvement that can be expected in air-core predictions is that the correct air-core shape and diameter predictions should improve the medium segregation with LES turbulence model compared to RSM turbulence model. Simulations were carried out with the above two viscosity corrections using user defined interpreted function plugged in to FLUENT solver with magnetite size distribution consideration of LES and ASM models. The results are presented in Figures 7.6 and 7.7 underneath:
Fig. 7.6. Comparison between predicted (LES-ASM) medium densities (left) and those measured by gamma ray tomography (right) for feed RD of 1.2; in elevation (Narasimha, 2007)

Fig. 7.7. Comparison between contours predicted (LES-ASM) by CFD (left) and those measured by gamma ray tomography (right) at 0.21 m from roof of cyclone for feed RD of 1.2 (Narasimha, 2007)

Based on LES turbulence model simulated results there appears to be a considerable improvement in the predicted axial segregation of magnetite in the cyclone medium densities. Fig. 7.6 shows a comparison between the contours of density as predicted by CFD and those measured at different levels. Fig. 7.7 shows a comparison at the level of 10 mm below the conical section.

The CFD is only a simulation, therefore its predictions follow the right trends but show some differences to those measured. Narasimha et al. (2007) highlight three features of CFD modelling:

- the CFD predicts that the medium is concentrated near the wall at the bottom of the conical section in a way that is not observed experimentally (Fig. 7.6),
the CFD predicts a ring of high medium density (Fig. 7.7) below the vortex finder\textsuperscript{28},

the CFD predicts that the density drops to that of water near the air/water surface; the medium segregates such that a film of nearly pure water forms around the air-cone\textsuperscript{29}.

The radial segregation of magnetite down the cyclone axis is reported in Figures 7.8 and 7.9.

Fig. 7.8. Comparison between contours predicted (LES-ASM) by CFD (left) and those measured by gamma ray tomography (right) at 0.47 m from roof of cyclone for feed RD of 1.2 (Narasimha, 2007)

Fig. 7.9. Comparison between contours predicted (LES-ASM) by CFD (left) and those measured by gamma ray tomography (right) at 0.67 m from roof of cyclone for feed RD of 1.2 (Narasimha, 2007)

\textsuperscript{28}‘This clearly occurs but the density predicted by CFD is a little high, being around 1400 kg/m\textsuperscript{3} compared to a measured peak of around 1300 kg/m\textsuperscript{3}’ (Narasimha et al., 2007: 50)

\textsuperscript{29}‘However, the tomography measurements show that the density drops to only 1100 kg/m\textsuperscript{3}. This in particular explains why the overflow density is under-predicted because of the overflow fluid passing through this region’ (ibid.).
Basing on Figures 7.8 and 7.9 Narasimha et al. (2007) observe that ‘though there is a good agreement in medium densities with the obtained data before the air/water surface, the CFD predicted results consistently predict higher medium densities near the wall, where as the measured values are showing density fall near the wall. This may be the shear thinning effect of the medium (magnetite) due to the dominance of lift force on magnetite particles near the wall’ (Narasimha et al. 2007: 50). The effect of size distribution of magnetite is shown in Figure 7.10.

Fig. 7.10. Distribution of different magnetite sizes inside the cyclone at 9D inlet pressure and RD at 1.243 feed density (Narasimha, 2007)
As one can learn from the data collected by Narasimha et al. (2007) as the size of magnetite increases, the segregation of magnetite near the wall side also increases. The $d_{50}$ of the magnetite segregation is 32 microns, which is expected of this feed size distribution.

The CFD predictions were compared with available literature of dense medium cyclone models and it could be seen that the CFD predicted underflow densities closely match the experimental results, measured by gamma ray tomography, where as the overflow densities are consistently under-predicted. According to Narasimha et al. (2007) it is believed that the over reported underflow volumetric flow rates are responsible for overflow densities under predictions. The authors also conclude: ‘the agreement between the correlations and the CFD are also reasonably good, but still the underflow density is lower than the model predictions. This is believed to be because of unaccounted back-mixing of the dispersed phase due to turbulence in the basic Algebraic Slip Mixture model. Though the Wood (1990) model predictions are reasonably close to experimental density predictions, the underflow density predictions are consistently lower than the experimental values as the Wood model is not accounting any medium viscosity effect in its calculations’ (Narasimha et al., 2007: 52).

![Fig. 7.11. Comparison between contours predicted (LES-ASM) by CFD and those measured by gamma ray tomography: (a) at 0.23 m and (b) at 0.47 m from roof of cyclone for feed RD of 1.467 (Narasimha, 2007)](image)

### 7.4. Devices’ designs and CFD
The application of CFD to the performance of hydrocyclones inspired inventors to improve the shapes of the devices and in India six new potentially more efficient designs were introduced. Changes considered, among many, widening of the cylinder. The expectation was that the widening cylinder would distance the coarse particles from the vortex finder (Degadillo and Rajamani 2007: 255). The marks and numbers provided follow those from the quoted text.

Novel design #1 and #2

As reported by Degadillo and Rajamani (2007) design #2 was a slight modification of a previous design #1 (both illustrated underneath). In design #2 the vortex finder length was reduced from 50 mm to 37.5 mm. This modification creates an increase in the tangential velocity at the top section. In this model particles entering the hydrocyclone are immediately classified. The expected result was an improvement in the classification process.

Novel design #3

As reported by the mentioned authors, the logic behind design #3 is to create the opposite effect created in designs #1 and #2. Design #3 is an exploration of the cone-angle effect on the classification process. The cylindrical section is replaced with a tapered cone of angle 30°. This modification changes the flow field and the classification process, creating a smooth classification and probably creating a lower turbulence in the fluid. The reduction of fluid turbulence improves the classification efficiency. The angle of the lower cone was kept at 20°.

Novel design #4

The double-cone configuration used in design #3 depends on the combination of cone angles. Design #4, is an improvement of design #3. In this design there are no abrupt changes in the outer walls, and hence the velocity field transitions smoothly. The angle of the cone tip was changed from 30° to 20° so that the main body is made up of a single cone. The angle of the lower cone was kept constant at 20°.

Novel design #5

Design #5 was based on modification in the cylindrical and conical sections. The cylinder length was reduced from 75 mm to 50 mm. A cone angle of 30° was placed followed by a second cone of 10° angle. This was a mix of standard and #3 models. The modification was introduced in order to create a more aggressive classification process and increase the residence time.

Novel design #6

30 See also Narasimha et al. (2005)
The last design was a further modification of design #5. The angles of the two cones were changed respectively from 30° to 20° and from 10° to 6° in order to produce a sharp classification process and an increase in the residence time. All designs, described in more details in Degadillo and Rajamani (2007) we present here:

![Diagram of six new designs](image)

Fig. 7.12. Dimension for six new designs (Degadillo, Rajamani, 2007)
The optimization of the classification performance through the modification of the standard geometry is easily achieved with the CFD methodology. The application was not more efficient in all cases. As we can read in the available papers the modification in the geometry in designs #1 and #2 did not improve the classification performance. The weak point in these designs was the lower tangential velocity in the vortex finder region. Designs #3 and #4 were a success in the sense that there was a small increase in the by-pass of the very fine sizes and the classification of coarse particles greatly improved. As Degadillo and Rajamani (2007) report ‘it should be noted that the pressure drop increased over the standard design, which means more energy per unit mass of suspension is spent’ (Degadillo and Rajamani 2007: 260). The authors also report that the fine cut size allows a better classification for mineral processing circuits with low mineral liberation sizes.

In designs #5 and #6 a slightly better efficiency was achieved. The slope of the classification curve is steeper than that of the standard classification curve. Since the computed pressure drop is lower, the flow rate can be increased to match the pressure drop of the standard hydrocyclone. Thus, when the pressure drop is increased, one expects more improvement in size classification. These designs showed that a double cone is the solution to improve classification. Designs #3 and #6 appeared to be promising configurations to create a sharper particle classification.
8. Literature review on CFD models of dense medium cyclones

Osborne in his paper (Osborne, 1986) wrote: effective beneficiation of fine coal; i.e., coal sized below 1 mm, either by means of deashing or dewatering, or a combination of both, is mainly dependent upon the following factors:

- favourable coal selling price and a growth situation in coal demand;
- improved performance and favourable cost of fine coal preparation equipment;
- degree of optimized beneficiation possible with respect to the coarser coal and the required product quality.

Further motivation has emerged as a result of a substantial increase in fines generation caused by expansion in mine mechanization coupled by sharply rising mining and transportation costs. With such incentives, research and development and progressive commercialization of new treatment devices has gathered momentum during the past ten years. The result has been that the demand for new treatment equipment, especially in the dewatering field, has tended to speed up development from the prototype machines to commercial application.

Almost all techniques of cleaning and dewatering fine coal have undergone some form of development, and new equipment for controlling or monitoring performance has emerged to create significant improvement in the performance of the more traditional, original methods.

In fine coal cleaning by means of jigs, tables, spirals, dense medium and autogenous (water) cyclones, collectively known as the gravity methods, all such methods have received attention as have froth flotation, oil agglomeration and certain magnetic and electrical separation and chemical techniques. In fine coal dewatering, vacuum and pressure filters, centrifuges of numerous types and thermal drying techniques have become widely used.

This paper reviews developments that have occurred in fine coal cleaning by gravity methods, concentrating on the past ten-year period. It also includes a brief glimpse into the immediate future in an attempt to try and forecast what may occur as a result of current research and development efforts.

Extensive separation tests were carried out on a 6” dense medium cyclone loop with the use of density tracers to study the effects of medium stability and rheology on the separation of fine particles (He, Laskowski, 1994). The medium properties were modified by changing the magnetic particle size and medium density. Four magnetite samples with RRB
size modulus d 63.2 ranging from 4.7 to 33 μm were used, and the medium densities were varied from 1.2 to 1.7 s.g. It was found that, while the separation efficiency and cutpoint shift for coarse particles (> 2.0 mm) were mainly determined by the medium stability, the separation performance of fine particles (< 0.5 mm) was more sensitive to the change in medium rheology.

Dense-medium separators have proved to be the most efficient processes for removing the undesirable gangue material from run-of-mine coal (Honaker et al., 2000). The application of high-pressure feed injection into dense-medium cyclones to provide an elevated centrifugal force has recently been found to allow efficient separation performances for the treatment of fine coal (i.e., < 1000 μm). However, high-pressure injection requires specialized pumps and results in relatively high maintenance requirements.

A test program has been conducted to evaluate the potential cleaning of 1000×44 μm fine coal using dense-medium in an enhanced gravity separator (EGS), which mechanically generates enhanced gravity field. Test results indicate that the use of dense-medium in an EGS resulted in an 8% weight unit increase in mass yield compared to the use of water-only for the treatment of an easy-to-clean Illinois No. 6 fine coal sample. For a difficult-to-clean coal, the mass yield improvement was significantly greater at nearly 20% by weight. From the treatment of four different coal samples, organic efficiency values greater than 90% were obtained over the entire range of product quality values. These findings are reflective of the highly efficient, low density separations provided by the dense-medium as indicated by probable error values below 0.05.

The partition function has been determined by (King, Juckes, 1988) for fine coal in a dense-medium cyclone. The partition curve is characterised by four parameters; the cut point, the separation efficiency, the short circuit to underflow and the short circuit to overflow. Each of these parameters is a strong function of particle size for fine coal. The effect of slimes in the feed is shown to have only a small effect on the measured partition curves.

A mathematical model is proposed by (Wang et al., 2009) to describe the multiphase flow in a dense-medium cyclone (DMC). In this model, the volume of fluid multiphase model is first used to determine the shape and position of the air core, and then the mixture multiphase model is employed to describe the flow of the dense medium (comprising finely ground magnetite in water) and the air core, where the turbulence is described by the Reynolds stress model. The results of fluid flow are finally used in the simulation of coal particle flow described by the stochastic Lagrangian particle tracking model. The validity of the proposed approach is verified by the reasonably good agreement between the measured and predicted results under different conditions. The flow features in a DMC are then examined in terms of factors such as flow field, pressure drop, particle trajectories, and separation efficiency. The results are used to explain the key characteristics of flow in DMCs,
such as the origin of a short-circuit flow, the flow pattern, and the motion of coal particles. Moreover, the so-called surging phenomenon is examined in relation to the instability of fluid flow. The model offers a convenient method to investigate the effects of variables related to geometrical and operational conditions on the performance of DMCs.

Numerical simulations of changes in feed medium solids on dense medium cyclone performance were performed (Mangadoddy, 2007), using a multi-phase mixture CFD (Computational Fluid Dynamics) model for medium and air-core coupled with Lagrangian model for coal particles for a 350mm DSM cyclone. The turbulence was resolved using Large Eddy Simulation (LES). The mixture model considered the interactions between water and solid phases in terms of hindered settling, lift and Bagnold forces at high feed medium solid loadings. The medium properties were modified by changing the particle size distribution and concentration. Three different medium sizes (ultrafine, superfine and fine) were used in the simulations. The effect of medium stability and rheology on DMC performance is related to feed medium size in terms of density differential and medium segregation. The simulations predicted low Ep (Ecort probability) values with finer medium which gives high separation efficiency on density. A reduction in cyclone efficiency observed for a given feed medium solids distribution at higher feed medium concentrations due to an increase in slurry viscosity.

The effect of medium composition on dense medium cyclone performance is investigated by (He, Laskowski, 1995). It is found that a change in composition affects medium stability and rheology, and may lead to different responses in separation efficiency and cutpoint shift. The final result depends on whether the medium stability or rheology is the dominant factor, and also on the feed particle size. An ideal medium composition is found to be one with a bimodal particle size distribution. Such a medium exhibits low viscosity and improved stability. Optimum dense medium cyclone performance is achieved with the magnetite dense medium characterized by a bimodal particle size distribution when the medium contains approximately 25% fine and 75% coarse magnetite. With a bimodal dense medium, separation efficiency is more closely related to medium rheology, while the cutpoint shift is more closely dependent on medium stability.

Paper (Wang et al., 2011) presents a numerical study of the gas–liquid–solid flow in 1000 mm dense medium cyclones (DMCs) with different body dimensions, which includes the spigot diameter, cylinder length, cone length and inlet size by means of a computer model which we recently proposed. In this model, mixture multiphase model is used to describe the flow of the dense medium (comprising finely ground magnetite contaminated with non-magnetic material in water) and the air core, where the turbulence is described by the well-established Reynolds Stress Model. The stochastic Lagrangian Particle Tracking method is used to simulate the flow of coal particles. It is found that the spigot size is very sensitive to the performance. The operational head and medium split reporting to overflow, decrease
dramatically as the spigot diameter increases. The density differential decreases rapidly, and then passes through a minimum and increases slowly. The long body including cylinder and cone is helpful to particle separation, particularly for fine and heavy particles. The inlet size plays a remarkable role on the performance on DMCs. The operational head, the density differential and the medium split increase dramatically as the inlet size decreases. Both the upward flow and the downward flow become very strong in the DMC with a small inlet when medium feed rate is constant, which results in a very low $E_p$.

The gas–liquid–solid flow and separation performance of 1000 mm DMC with different body dimensions are numerically studied, focused on variables such as the spigot diameter, cylinder length, cone length and inlet size. The results are useful for better design and control of the DMC operation.

The multiphase flow in DMC is numerically studied. The effects of spigot, cylinder, cone and inlet dimensions on DMC flow and performance are quantified.

The motion of solid particles and the “fish-hook” phenomenon in an industrial classifying hydrocyclone of body diameter 355 mm is studied by a computational fluid dynamics model, by (Wang et al., 2010). In the model, the turbulent flow of gas and liquid is modeled using the Reynolds Stress Model, and the interface between the liquid and air core is modeled using the volume of fluid multiphase model. The outcomes are then applied in the simulation of particle flow described by the stochastic Lagrangian model. The results are analysed in terms of velocity and force field in the cyclone. It is shown that the pressure gradient force plays an important role in particle separation, and it balances the centrifugal force on particles in the radial direction in hydrocyclones. As particle size decreases, the effect of drag force whose direction varies increases sharply. As a result, particles have an apparent fluctuating velocity. Some particles pass the locus of zero vertical velocity (LZVV) and join the upward flow and have a certain moving orbit. The moving orbit of particles in the upward flow becomes wider as their size decreases. When the size is below a critical value, the moving orbit is even beyond the LZVV. Some fine particles would recircuit between the downward and upward flows, resulting in relatively high separation efficiency and the “fish-hook” effect. Numerical experiments were also extended to study the effects of cyclone size and liquid viscosity. The results suggest that the mechanisms identified are valid, although they are quantitatively different.

An analytical study of the flow and pressure fields inside a small-diameter dense-media cyclone is presented by (Shen et al., 2009). The simulations were done with the help of the CFD software FLUENT. The following conclusions were reached: the tangential velocity tends to increase when moving from the centre toward the exterior. The velocity then begins to decrease when the maximum velocity point is reached. The velocity field divides into two different sections; an inner swirling zone and an outer swirling zone. The axial velocity points
down at the wall and gradually decreases toward the bottom. Continuing toward the bottom, the axial velocity passes through zero and then gradually increases in the opposite direction. In the cyclone's central zone, the pressure is negative and the suction of air allows an air column to be formed therein. At the centre of the radial negative zone the pressure drops to its lowest value—phenomenon that has been verified by theoretical analysis. Some discrepancies between the observed data and the simulated data are noted when an analysis is made on a cyclone operating with either fresh water only or with water with added heavy particles.

Dense-medium separators have proven to be the most efficient processes for removing the undesirable material from run-of-mine coal (Parate et al., 2010). The application of high-pressure feed injection into dense-medium cyclones to provide an elevated centrifugal force has recently been found to allow efficient separation performances for the treatment of fine coal (i.e., <1000 µm). However, high-pressure injection requires specialized pumps and results in relatively high maintenance requirements. Current study involves experimental investigation of separation performance characteristics of the dense media hydrocyclone (DMC). A pilot plant DMC has been designed and fabricated for performance characterization. Experiments have been conducted on 300 mm dense medium cyclone treating coal in the size range of −6 to +2 mm using magnetite as the medium under operating conditions. The operating variable was the specific gravity of the medium, feed inlet pressure and feed inlet flow rate. The ash contents of the feed coal reporting to the overflow and underflow have been analysed qualitatively. The result indicates that the use of magnetite as dense medium in DMC resulted in the yield of clean coal, which is 5% more when the air core is suppressed as compared to the same conditions when the air core remains. A 3-D geometry is created in Gambit to support the experimental findings by using CFD simulation. It is interesting to observe that experimental findings agree well with the simulation results.
9. Research

9.1. Sampling of mechanical processing plant in KWK Zofiówka

The copy of original results of sampling of mechanical processing plant in KWK Zofiówka in previous years is presented in Appendix B.

In the following chapters the following measured values are presented:

− content of solid phase and ash content in feed, overflow and underflow of samples taken at different times,
− yield and ash content for different size grades for averaged sample.

There are presented values for both 33" (inch) cyclone and 26" (inch) cyclone. However for CFD modeling only values obtained for 33" cyclone were used.

9.1.1. Feed of 33" cyclone sampling results

The results of measured solid phase and ash content in samples are presented in Table 9.1. The content of solid phase is between 93.8 and 177.0 g/dm³. The ash content is in the range of 16.2 to 20.7%. Results of grain-ash content analysis are presented in Table 9.2.

<table>
<thead>
<tr>
<th>Date and time</th>
<th>Content of solid phase [g/dm³]</th>
<th>Ash content [%]</th>
</tr>
</thead>
<tbody>
<tr>
<td>16.05.2007, 10:20</td>
<td>106.3</td>
<td>16.28</td>
</tr>
<tr>
<td>16.05.2007, 12:00</td>
<td>137.3</td>
<td>20.72</td>
</tr>
<tr>
<td>16.05.2007, 13:00</td>
<td>138.7</td>
<td>17.27</td>
</tr>
<tr>
<td>30.05.2007, 10:00</td>
<td>93.8</td>
<td>17.87</td>
</tr>
<tr>
<td>30.05.2007, 11:00</td>
<td>177.0</td>
<td>16.19</td>
</tr>
<tr>
<td><strong>Average</strong></td>
<td><strong>130.6</strong></td>
<td><strong>17.67</strong></td>
</tr>
</tbody>
</table>
Table 9.2. Grain size analysis of the average sample of feed for 33” cyclone

<table>
<thead>
<tr>
<th>Size grade [mm]</th>
<th>Yield [%]</th>
<th>Sum of yield [%]</th>
<th>Ash content [%]</th>
<th>Average ash content [%]</th>
</tr>
</thead>
<tbody>
<tr>
<td>+3</td>
<td>0.4</td>
<td>0.4</td>
<td>6.53</td>
<td>6.53</td>
</tr>
<tr>
<td>3–2</td>
<td>2.3</td>
<td>2.7</td>
<td>6.68</td>
<td>6.66</td>
</tr>
<tr>
<td>2–1</td>
<td>10.2</td>
<td>12.9</td>
<td>5.42</td>
<td>5.68</td>
</tr>
<tr>
<td>1–0.5</td>
<td>17.8</td>
<td>30.8</td>
<td>6.67</td>
<td>6.25</td>
</tr>
<tr>
<td>0.5–0.3</td>
<td>8.8</td>
<td>39.6</td>
<td>9.68</td>
<td>7.02</td>
</tr>
<tr>
<td>0.3–0.2</td>
<td>9.5</td>
<td>49.1</td>
<td>12.15</td>
<td>8.01</td>
</tr>
<tr>
<td>0.2–0.1</td>
<td>9.9</td>
<td>59.0</td>
<td>13.54</td>
<td>8.94</td>
</tr>
<tr>
<td>0.1–0.045</td>
<td>9.1</td>
<td>68.1</td>
<td>16.52</td>
<td>9.95</td>
</tr>
<tr>
<td>0.045–0.025</td>
<td>5.3</td>
<td>73.4</td>
<td>15.22</td>
<td>10.33</td>
</tr>
<tr>
<td>-0.0025</td>
<td>26.6</td>
<td>100.0</td>
<td>36.18</td>
<td>17.20</td>
</tr>
<tr>
<td>Sum</td>
<td>100.0</td>
<td></td>
<td>17.20</td>
<td></td>
</tr>
</tbody>
</table>

9.1.2. Overflow of 33” cyclone sampling results

The results of measured solid phase and ash content in samples are presented in Table 9.3. The content of solid phase is between 34.8 and 59.7 g/dm³. The ash content is in the range of 19.3 to 21.3%. Results of grain-ash content analysis are presented in Table 9.4.

Table 9.3. Characteristics of samples of the overflow for 33” cyclone

<table>
<thead>
<tr>
<th>Date and time</th>
<th>Content of solid phase [g/dm³]</th>
<th>Ash content [%]</th>
</tr>
</thead>
<tbody>
<tr>
<td>16.05.2007, 10:20</td>
<td>34.8</td>
<td>19.32</td>
</tr>
<tr>
<td>16.05.2007, 12:00</td>
<td>50.8</td>
<td>19.56</td>
</tr>
<tr>
<td>16.05.2007, 13:00</td>
<td>43.0</td>
<td>21.25</td>
</tr>
<tr>
<td>30.05.2007, 10:00</td>
<td>47.6</td>
<td>20.13</td>
</tr>
<tr>
<td>30.05.2007, 11:00</td>
<td>59.7</td>
<td>20.65</td>
</tr>
<tr>
<td>Average</td>
<td>47.2</td>
<td>20.18</td>
</tr>
</tbody>
</table>

Table 9.4. Grain size analysis of the average sample of overflow for 33” cyclone

<table>
<thead>
<tr>
<th>Size grade [mm]</th>
<th>Yield [%]</th>
<th>Sum of yield [%]</th>
<th>Ash content [%]</th>
<th>Average ash content [%]</th>
</tr>
</thead>
<tbody>
<tr>
<td>+0.5</td>
<td>0.3</td>
<td>0.3</td>
<td>1.99</td>
<td>1.99</td>
</tr>
<tr>
<td>0.5–0.3</td>
<td>1.9</td>
<td>2.2</td>
<td>2.02</td>
<td>2.02</td>
</tr>
<tr>
<td>0.3–0.2</td>
<td>6.2</td>
<td>8.4</td>
<td>1.18</td>
<td>1.40</td>
</tr>
<tr>
<td>0.2–0.1</td>
<td>14.9</td>
<td>23.3</td>
<td>1.77</td>
<td>1.64</td>
</tr>
<tr>
<td>0.1–0.045</td>
<td>17.1</td>
<td>40.5</td>
<td>5.50</td>
<td>3.27</td>
</tr>
<tr>
<td>0.045–0.025</td>
<td>10.0</td>
<td>50.4</td>
<td>11.58</td>
<td>4.91</td>
</tr>
</tbody>
</table>
9.1.3. Underflow of 33” cyclone sampling results

The results of measured solid phase and ash content in samples are presented in Table 9.5. The content of solid phase is between 606.2 and 747.4 g/dm$^3$. The ash content is in the range of 13.4 to 16.5%. Results of grain-ash content analysis are presented in Table 9.6.

Table 9.5. Characteristics of samples of the underflow for 33” cyclone

<table>
<thead>
<tr>
<th>Date and time</th>
<th>Content of solid phase [g/dm$^3$]</th>
<th>Ash content [%]</th>
</tr>
</thead>
<tbody>
<tr>
<td>16.05.2007, 10:20</td>
<td>657.9</td>
<td>13.35</td>
</tr>
<tr>
<td>16.05.2007, 12:00</td>
<td>747.4</td>
<td>15.02</td>
</tr>
<tr>
<td>16.05.2007, 13:00</td>
<td>606.2</td>
<td>16.52</td>
</tr>
<tr>
<td>30.05.2007, 10:00</td>
<td>716.1</td>
<td>13.52</td>
</tr>
<tr>
<td>30.05.2007, 11:00</td>
<td>744.9</td>
<td>15.22</td>
</tr>
<tr>
<td>Average</td>
<td>694.5</td>
<td>14.73</td>
</tr>
</tbody>
</table>

Table 9.6. Grain size analysis of the average sample of underflow for 33” cyclone

<table>
<thead>
<tr>
<th>Size grade [mm]</th>
<th>Yield [%]</th>
<th>Sum of yield [%]</th>
<th>Ash content [%]</th>
<th>Average ash content [%]</th>
</tr>
</thead>
<tbody>
<tr>
<td>+3</td>
<td>0.2</td>
<td>0.2</td>
<td>12.84</td>
<td>12.84</td>
</tr>
<tr>
<td>3–2</td>
<td>5.1</td>
<td>5.3</td>
<td>8.55</td>
<td>8.73</td>
</tr>
<tr>
<td>2–1</td>
<td>22.3</td>
<td>27.6</td>
<td>7.37</td>
<td>7.63</td>
</tr>
<tr>
<td>1–0.5</td>
<td>44.5</td>
<td>72.1</td>
<td>10.77</td>
<td>9.57</td>
</tr>
<tr>
<td>0.5–0.3</td>
<td>4.5</td>
<td>76.5</td>
<td>11.80</td>
<td>9.70</td>
</tr>
<tr>
<td>0.3–0.2</td>
<td>10.8</td>
<td>87.3</td>
<td>19.36</td>
<td>10.89</td>
</tr>
<tr>
<td>0.2–0.1</td>
<td>7.0</td>
<td>94.3</td>
<td>41.44</td>
<td>13.16</td>
</tr>
<tr>
<td>0.1–0.045</td>
<td>3.4</td>
<td>97.7</td>
<td>59.08</td>
<td>14.75</td>
</tr>
<tr>
<td>0.045–0.025</td>
<td>0.9</td>
<td>98.6</td>
<td>55.47</td>
<td>15.12</td>
</tr>
<tr>
<td>-0.0025</td>
<td>1.4</td>
<td>100.0</td>
<td>43.68</td>
<td>15.51</td>
</tr>
<tr>
<td>Sum</td>
<td>100.0</td>
<td></td>
<td>15.51</td>
<td></td>
</tr>
</tbody>
</table>

9.1.4. Feed of 26” cyclone sampling results

The results of measured solid phase and ash content in samples are presented in Table 9.7. The content of solid phase is between 58.6 and 86.6 g/dm$^3$. The ash content is in the range of 19.6 to 28.5%. Results of grain-ash content analysis are presented in Table 9.8.
<table>
<thead>
<tr>
<th>Date and time</th>
<th>Content of solid phase [g/dm³]</th>
<th>Ash content [%]</th>
</tr>
</thead>
<tbody>
<tr>
<td>16.05.2007, 10:20</td>
<td>86.6</td>
<td>22.00</td>
</tr>
<tr>
<td>16.05.2007, 12:00</td>
<td>76.4</td>
<td>19.56</td>
</tr>
<tr>
<td>16.05.2007, 13:00</td>
<td>75.2</td>
<td>21.65</td>
</tr>
<tr>
<td>30.05.2007, 10:00</td>
<td>58.6</td>
<td>24.12</td>
</tr>
<tr>
<td>30.05.2007, 11:00</td>
<td>72.5</td>
<td>28.47</td>
</tr>
<tr>
<td><strong>Average</strong></td>
<td><strong>73.9</strong></td>
<td><strong>23.16</strong></td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Size grade [mm]</th>
<th>Yield [%]</th>
<th>Sum of yield [%]</th>
<th>Ash content [%]</th>
<th>Average ash content [%]</th>
</tr>
</thead>
<tbody>
<tr>
<td>+1.0</td>
<td>2.1</td>
<td>2.1</td>
<td>2.50</td>
<td>2.50</td>
</tr>
<tr>
<td>1–0.5</td>
<td>11.2</td>
<td>13.3</td>
<td>4.15</td>
<td>3.89</td>
</tr>
<tr>
<td>0.5–0.3</td>
<td>10.6</td>
<td>23.9</td>
<td>4.60</td>
<td>4.21</td>
</tr>
<tr>
<td>0.3–0.2</td>
<td>9.4</td>
<td>33.3</td>
<td>10.63</td>
<td>6.02</td>
</tr>
<tr>
<td>0.2–0.1</td>
<td>12.3</td>
<td>45.6</td>
<td>29.11</td>
<td>12.26</td>
</tr>
<tr>
<td>0.1–0.045</td>
<td>12.7</td>
<td>58.3</td>
<td>27.98</td>
<td>15.68</td>
</tr>
<tr>
<td>-0.045</td>
<td>41.7</td>
<td>100.0</td>
<td>36.55</td>
<td>24.38</td>
</tr>
<tr>
<td><strong>Sum</strong></td>
<td><strong>100.0</strong></td>
<td></td>
<td><strong>24.38</strong></td>
<td></td>
</tr>
</tbody>
</table>

**9.1.5. Overflow of 26” cyclone sampling results**

The results of measured solid phase and ash content in samples are presented in Table 9.9. The content of solid phase is between 31.9 and 41.4 g/dm³. The ash content is in the range of 23.5 to 25.6%. Results of grain-ash content analysis are presented in Table 9.10.

<table>
<thead>
<tr>
<th>Date and time</th>
<th>Content of solid phase [g/dm³]</th>
<th>Ash content [%]</th>
</tr>
</thead>
<tbody>
<tr>
<td>16.05.2007, 10:20</td>
<td>33.3</td>
<td>24.99</td>
</tr>
<tr>
<td>16.05.2007, 12:00</td>
<td>37.3</td>
<td>25.63</td>
</tr>
<tr>
<td>16.05.2007, 13:00</td>
<td>41.4</td>
<td>25.56</td>
</tr>
<tr>
<td>30.05.2007, 10:00</td>
<td>35.7</td>
<td>24.20</td>
</tr>
<tr>
<td>30.05.2007, 11:00</td>
<td>31.9</td>
<td>23.48</td>
</tr>
<tr>
<td><strong>Average</strong></td>
<td><strong>35.9</strong></td>
<td><strong>24.77</strong></td>
</tr>
</tbody>
</table>
Table 9.10. Grain size analysis of the average sample of overflow for 26" cyclone

<table>
<thead>
<tr>
<th>Size grade [mm]</th>
<th>Yield [%]</th>
<th>Sum of yield [%]</th>
<th>Ash content [%]</th>
<th>Average ash content [%]</th>
</tr>
</thead>
<tbody>
<tr>
<td>+0.3</td>
<td>0.6</td>
<td>0.6</td>
<td>2.02</td>
<td>2.02</td>
</tr>
<tr>
<td>0.3–0.2</td>
<td>0.6</td>
<td>1.2</td>
<td>2.13</td>
<td>2.08</td>
</tr>
<tr>
<td>0.2–0.1</td>
<td>7.3</td>
<td>8.5</td>
<td>2.23</td>
<td>2.21</td>
</tr>
<tr>
<td>0.1–0.045</td>
<td>11.3</td>
<td>19.8</td>
<td>4.16</td>
<td>3.32</td>
</tr>
<tr>
<td>-0.045</td>
<td>80.2</td>
<td>100.0</td>
<td>31.94</td>
<td>26.28</td>
</tr>
<tr>
<td>Sum</td>
<td>100.0</td>
<td></td>
<td>26.28</td>
<td></td>
</tr>
</tbody>
</table>

9.1.6. Underflow of 26" cyclone sampling results

The results of measured solid phase and ash content in samples are presented in Table 9.11. The content of solid phase is between 445.5 and 717.6 g/dm³. The ash content is in the range of 12.8 to 26.2%. Results of grain-ash content analysis are presented in Table 9.12.

Table 9.11. Characteristics of samples of the underflow for 26" cyclone

<table>
<thead>
<tr>
<th>Date and time</th>
<th>Content of solid phase [g/dm³]</th>
<th>Ash content [%]</th>
</tr>
</thead>
<tbody>
<tr>
<td>16.05.2007, 10:20</td>
<td>635.3</td>
<td>15.84</td>
</tr>
<tr>
<td>16.05.2007, 12:00</td>
<td>445.5</td>
<td>13.83</td>
</tr>
<tr>
<td>16.05.2007, 13:00</td>
<td>629.9</td>
<td>20.69</td>
</tr>
<tr>
<td>30.05.2007, 10:00</td>
<td>537.2</td>
<td>12.80</td>
</tr>
<tr>
<td>30.05.2007, 11:00</td>
<td>717.6</td>
<td>26.23</td>
</tr>
<tr>
<td>Average</td>
<td>593.1</td>
<td>17.88</td>
</tr>
</tbody>
</table>

Table 9.12. Grain size analysis of the average sample of underflow for 26" cyclone

<table>
<thead>
<tr>
<th>Size grade [mm]</th>
<th>Yield [%]</th>
<th>Sum of yield [%]</th>
<th>Ash content [%]</th>
<th>Average ash content [%]</th>
</tr>
</thead>
<tbody>
<tr>
<td>+3</td>
<td>0.3</td>
<td>0.3</td>
<td>6.46</td>
<td>6.46</td>
</tr>
<tr>
<td>3–2</td>
<td>1.0</td>
<td>1.3</td>
<td>4.34</td>
<td>4.88</td>
</tr>
<tr>
<td>2–1</td>
<td>6.7</td>
<td>8.0</td>
<td>4.63</td>
<td>4.67</td>
</tr>
<tr>
<td>1–0.5</td>
<td>37.7</td>
<td>45.6</td>
<td>4.54</td>
<td>4.56</td>
</tr>
<tr>
<td>0.5–0.3</td>
<td>7.1</td>
<td>52.7</td>
<td>10.50</td>
<td>5.36</td>
</tr>
<tr>
<td>0.3–0.2</td>
<td>17.6</td>
<td>70.3</td>
<td>17.89</td>
<td>8.49</td>
</tr>
<tr>
<td>0.2–0.1</td>
<td>16.2</td>
<td>86.5</td>
<td>31.47</td>
<td>12.80</td>
</tr>
<tr>
<td>0.1–0.045</td>
<td>8.1</td>
<td>94.6</td>
<td>62.48</td>
<td>17.04</td>
</tr>
<tr>
<td>0.045–0.025</td>
<td>2.2</td>
<td>96.8</td>
<td>64.15</td>
<td>18.12</td>
</tr>
<tr>
<td>-0.0025</td>
<td>3.2</td>
<td>100.0</td>
<td>42.26</td>
<td>18.89</td>
</tr>
</tbody>
</table>
9.2. CFD model of cyclone in KWK Zofiówka

9.2.1. Flow model used

In our research we used analogues set up of CFD model as in work of Nowak et al. (2006). The $k–\varepsilon$ turbulent flow model has been used for calculations. The standard model is a semi-empirical model based on model transport equations for the turbulence kinetic energy ($k$) and its dissipation rate ($\varepsilon$).

In Eulerian frame, the turbulent flow field may by specified by velocity vector $\mathbf{U}$ with three orthogonal components ($u_1$, $u_2$, $u_3$). In Reynolds averaging, each velocity component may be split into the mean (ensemble-averaged or time-averaged) and fluctuating (turbulence) components (eq. (9.1)).

$$u_i = \bar{u}_i + u_i'$$

(9.1)

where:
- $\bar{u}_i$ – the mean velocity components ($i = 1, 2, 3$),
- $u_i'$ – the fluctuating velocity components ($i = 1, 2, 3$).

Besides the velocity $\mathbf{U}$, the turbulent flow field may by specified by pressure $p$, density $\rho$ and temperature $T$. For the determination of these quantities the following differential equations are requisite:

- conservation of momentum

$$\frac{\partial}{\partial t}(\rho u_i) + \frac{\partial}{\partial x_j} (\rho u_i u_j) = \frac{\partial p}{\partial x_i} + \frac{\partial}{\partial x_j} \left[ \mu \left( \frac{\partial u_i}{\partial x_j} + \frac{\partial u_j}{\partial x_i} - \frac{2}{3} \frac{\partial}{\partial x_j} \frac{\partial}{\partial x_i} \right) \right] + \frac{\partial}{\partial x_j} (-u_i'u_i')$$

(9.2)

where:
- $\mu$ – the laminar viscosity,
- $\delta_{ij}$ – the Kronecker delta function,

and the last element of this equation represents the Reynolds stress which are relate to the mean velocity gradients:

$$-\bar{u}_i'u_i' = \mu_t \left( \frac{\partial u_i}{\partial x_j} + \frac{\partial u_j}{\partial x_i} \right) - \frac{2}{3} \left( \nu k + \mu_{ij} \frac{\partial}{\partial x_i} \right) \delta_{ij}$$

(9.2a)

where $\mu_t$ is the turbulent viscosity.
- continuity of flow

\[
\frac{\partial \rho}{\partial t} + \frac{\partial}{\partial x_i} (\rho u_i) = 0
\]  

(9.3)

- energy equation

\[
\frac{\partial}{\partial t} (\rho E) + \frac{\partial}{\partial x_i} (\rho u_i E) = \frac{\partial}{\partial x_j} \left( \lambda \frac{\partial T}{\partial x_j} + \rho u_i \frac{\partial T}{\partial x_j} \right)
\]

(9.4)

where:

\[E\] the total energy,
\[\lambda\] the thermal conductivity.

- transport equations for the turbulence kinetic energy \( k \)

\[
\frac{\partial}{\partial t} (\rho k) + \frac{\partial}{\partial x_i} (\rho u_i k) = \frac{\partial}{\partial x_j} \left[ \left( \mu + \frac{\mu_k}{\sigma_k} \right) \frac{\partial k}{\partial x_j} \right] + C_{1\varepsilon} \frac{\varepsilon}{k} + C_{2\varepsilon} \varepsilon - \frac{\varepsilon}{M}
\]

(9.5)

- transport equations for the turbulence energy dissipation \( \varepsilon \)

\[
\frac{\partial}{\partial t} (\rho \varepsilon) + \frac{\partial}{\partial x_i} (\rho u_i \varepsilon) = \frac{\partial}{\partial x_j} \left[ \left( \mu + \frac{\mu_k}{\sigma_k} \right) \frac{\partial \varepsilon}{\partial x_j} \right] + C_{1\varepsilon} \frac{\varepsilon}{k} + C_{2\varepsilon} \varepsilon - C_{3\varepsilon} \frac{\varepsilon^2}{k}
\]

(9.6)

where:

\[G_k\] the generation of turbulence kinetic energy due to the mean velocity gradients,
\[G_b\] the generation of turbulence kinetic energy due to buoyancy,
\[Y_M\] the contribution of the fluctuating dilatation in compressible turbulence to the overall dissipation rate,
\[C_{1\varepsilon}, C_{2\varepsilon}, C_{3\varepsilon}\] constants,
\[\sigma_k, \sigma_\varepsilon\] the turbulent Prandtl numbers for \( k \) and \( \varepsilon \), respectively.

- equation for the turbulent (or eddy) viscosity \( \mu_t \)

\[
\mu_t = \frac{\mu_k}{\sigma_\varepsilon}
\]

(9.7)

where \( C_{\mu} \) is a constant.

The model constants \( C_{1\varepsilon}, C_{2\varepsilon}, C_{\mu}, \sigma_k \) and \( \sigma_\varepsilon \) have the following default values:
These default values have been determined from experiments with air and water for fundamental turbulent shear flows including homogeneous shear flows and decaying isotropic grid turbulence. They have been found to work fairly well for a wide range of wall-bounded and free shear flows.

9.2.2. Software used

Software GAMBIT and FLUENT has been used for CFD simulation. The GAMBIT software has been used for the cyclone’s geometry and the mesh preparation. The three-dimensional tetrahedral-hybrid mesh was created by the software and a total of ca. 0.5 million mesh cells were created. Simulation of water and solid particles flow inside the cyclone was carried using FLUENT software. The particles flowed into the cyclone’s inlet with the same velocity as water. The particles trajectory is calculated by force balance equations and turbulent dispersion of particles where the Discrete Random Walk model is used.

9.2.3. Parameters of the model

19 phases were modelled:
- Phase no 1: heavy medium (suspension of water and magnetite)
- Phases no 2–10: 9 phases of coal,
- Phases no 11–19: 9 phases of ash.

For coal and ash two most coarse phases from cyclone sampling (there were 10 phases separated) were combined.

Assumed parameters:
- density of heavy medium 1320 kg/m³,
- viscosity of heavy medium 0.003 kg·(m·s)^{-1},
- density of coal 1200 kg/m³,
- density of ash 2600 kg/m³.

After steady flow was accomplished (two weeks of calculations were required to obtain steady parameters of the flow) following values of flow were obtained:
- feed 0.246 m³/s,
- overflow 0.187 m³/s,
- underflow 0.0586 m³/s.
For the first attempt as the starting (boundary) condition a cyclone filled homogeneously with only heavy medium was assumed. This attempt was not good from the perspective of numerical stability. The second attempt was to begin with cyclone filled homogeneously with heavy medium, ash and coal. Average values of ash and coal content were used. This attempt was successful.

In the interior of cyclone an area of relative negative pressure of the water was observed. This addresses creation of air core described in previous chapters. This was modelled by adding artificial cone inside cyclone with boundaries with no shear. This artificial con can be seen on figures presenting results of simulation.

9.3. Discussion of results

The polish coal industry has been progressively adopting dense medium cyclone (DMC) processing technology for the last 40 years. There is little readily available information on the operation of these types of units. Access to such information will allow the optimization of the design and operation of plants that include these units. As a result, Author has begun studies in this area.

This work has created a framework for the development of a fundamentally based mathematical description of dense medium cyclones (DMC). The dynamics of multiphase flow through dense medium cyclones has been derived from basic principles and utilising Computational Fluid Dynamics (CFD) approach. Discrete element modelling (DEM) methodology has then been applied to incorporate the dynamics of particle flow.

The methodology and concepts used in this project are rather new and are becoming more popular among researchers and engineers.

Once validated, the CFD-DEM model developed in this work gives possibility of full understanding of how a dense medium cyclone operates.

For example, it was used to understand:

− pressure distributions inside an operating cyclone,
− density distributions inside an operating cyclone,
− tangential, axial and radial velocity distributions inside an operating dense medium cyclone.
− formation of air core that was continually in motion through which air passed from spigot to vortex finder outlets.

Industrial trials of two cyclones – a 33 inch and a 26 inch diameter dense medium cyclone were conducted at Coal Beneficiating Plant in KWK “Zofiówka”. The industrial studies have already improved the understanding of the relationship between the cyclone
performance and the operating conditions, in particular medium flow rates and feed pressures. Set of industrial trials is presented in Appendix B.

For the purpose of this work five sampling tests were chosen. The coal and ash flow rate of DMC feed, overflow and underflow was measured for every test.

The use of DMCs has become widespread in the polish coal industry and, in the future, they are likely to be the dominant processing unit. The aim of this work was to increase the knowledge of performance of large dense medium cyclones. It is to provide a better understanding of the operational efficiency of the large cyclones and allow the industry to optimise their plant designs and operations.

This work was essentially a scoping study to identify key operating traits of dense medium cyclones, and to recommend project areas that would yield benefits to the industry if further R&D were carried out.

Readily available information from the literature has been collated to provide a firm basis for the consideration of the observed operational results.

This work has shown that it is possible to make useful predictions of dense medium cyclone performance with a CFD model of the DMC running in the real plant.

The model reads operating data from the plant (coal and ash content in the feed, densities, feed pressure) and predicts output of cyclone. All important parameters of overflow and underflow (e.g. coal and ash content in given size grades, densities) can be predicted with good accuracy.

Figure 9.1 shows pressure on a vertical plane inside cyclone.
Figure 9.2 shows volume fraction of Phase 4 (coal) in the mixture on 3 horizontal planes.

**Fig. 9.2.** Volume fraction of Phase 4 (coal) in the mixture on 3 horizontal planes

Figure 9.3 shows volume fraction of Phase 15 (ash) in the mixture on one vertical plane.

**Fig. 9.3.** Volume fraction of Phase 15 (ash) in the mixture on one vertical plane
Figure 9.4 shows vectors of velocity on three horizontal planes, coloured by magnitude of velocity.

![Figure 9.4. Vectors of velocity on three horizontal planes, colored by magnitude of velocity.](image)

Figures 9.5 and 9.6 show value of velocity in vertical direction only. On the first figure only points with velocity vectors directed upwards are shown, on the second one downwards.

![Fig. 9.5. Value of velocity in vertical upwards direction only](image)
Figure 9.6 shows values of turbulent energy on a vertical plane.

Appendix C shows several dozen graphic presentations of results obtained by using presented model for existing industrial scale working device. Presented approach allows us to realistically model flow of both medium and solid particles.
Figure 9.8 presents comparison of particle size distribution of average sample and results obtained by the model.

![Comparison of particle size distribution](image)

**Fig. 9.8.** Comparison of particle size distribution of sample and model

Important feature of elaborated model is its foundation (assumptions). Earlier cyclones were modelled using empirical (statistical) data. These models were based on regression analysis and assumption of similar conditions in real plants comparing to model conditions. Selection of parameters of cyclone was dependent on state of art knowledge and experience of engineers. Due to stochastic parameters of flow of medium and solid particles, continuous changes of feed parameters may result in not-continuous changes in over and underflow parameters. This weakness of empirical models can be overcome using CFD approach.

The foundations of CFD model are basic laws of physics. The model of turbulences has to be assumed. The geometry of the device is an input parameter, which can be easily altered using modern software as GAMBIT. The other inputs to the model are feed parameters (density, pressure, coal and ash content in size grade viscosity etc.).

Based only on these assumptions and input data the CFD model can produce output data as all required parameters of over and underflow (e.g. flow rate, coal and ash content in each size grades), distribution of pressure and density. It is even possible to trace single particle.

Due to complicity and amount of calculations to be done, only nowadays one is able to conduct required calculations (i.e. solving of differential equations numerically) using supercomputers.
References


van der Walt R., 1993: *Magnetite recovery tests conducted at Middelburg Mining Services Colliery*. Report ENER-I 93003, Division of Energy Technology, CSIR.


Publications of the Author


Appendix A. Scheme of Processing Plant in KWK Zofiówka

Fig. A.1. Scheme of Processing Plant in KWK Zofiówka
Appendix B. Sampling of mechanical processing plant in KWK Zofiówka

Opróbowanie hydrocykłonów Krebsa w dniu 09.12.2004r.
Płuczka poz. 29m, godz. 8:30

<table>
<thead>
<tr>
<th></th>
<th>Przelew 3.71.1</th>
<th>Wylew</th>
<th>Przelew 3.71.2</th>
<th>Wylew</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Dysza Ø 350mm</td>
<td></td>
<td>Dysza Ø 300mm</td>
<td></td>
</tr>
<tr>
<td>Zagęszczenie (g/l)</td>
<td>77</td>
<td>765</td>
<td>72</td>
<td>750</td>
</tr>
<tr>
<td>Ad</td>
<td></td>
<td></td>
<td>10,1</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Analiza sítowa (%)</td>
<td>1,9</td>
<td>9,9</td>
<td>2,1</td>
<td>9,8</td>
</tr>
<tr>
<td>+ 1,8</td>
<td>0,0</td>
<td></td>
<td>0,0</td>
<td>9,8</td>
</tr>
<tr>
<td>1,8 - 1,0</td>
<td>0,0</td>
<td>9,9</td>
<td>0,0</td>
<td>24,0</td>
</tr>
<tr>
<td>1,0 - 0,75</td>
<td>0,5</td>
<td>12,3</td>
<td>0,0</td>
<td>9,8</td>
</tr>
<tr>
<td>0,75 - 0,50</td>
<td>1,4</td>
<td>20,6</td>
<td>0,0</td>
<td>20,3</td>
</tr>
<tr>
<td>0,50 - 0,315</td>
<td>5,3</td>
<td>18,1</td>
<td>9,3</td>
<td>34,0</td>
</tr>
<tr>
<td>0,315 - 0,0</td>
<td>92,8</td>
<td>37,2</td>
<td>90,7</td>
<td>100,0</td>
</tr>
<tr>
<td>Razem</td>
<td>100,0</td>
<td>100,0</td>
<td>100,0</td>
<td>100,0</td>
</tr>
</tbody>
</table>

Fig. B.1. Report of measurement
Opróbowanie hydrocyklonów Krebsa w dniu 10.12.2004r.
Płużka poz. 29m, godz. 9:00

<table>
<thead>
<tr>
<th>Zanieczyszczenie (g/l)</th>
<th>Przelew 3.71.1 Dyszsa Ø 350mm</th>
<th>Wylew</th>
<th>Przelew 3.71.2 Dyszsa Ø 300mm</th>
<th>Wylew</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>96</td>
<td>707</td>
<td>92</td>
<td>741</td>
</tr>
<tr>
<td>Ad</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

**Analiza sitowa (%)**

<table>
<thead>
<tr>
<th></th>
<th>Przelew 3.71.1</th>
<th>Wylew</th>
<th>Przelew 3.71.2</th>
<th>Wylew</th>
</tr>
</thead>
<tbody>
<tr>
<td>+ 1,8</td>
<td>0,0</td>
<td>2,1</td>
<td>0,0</td>
<td>2,1</td>
</tr>
<tr>
<td>1,8 – 1,0</td>
<td>0,0</td>
<td>10,3</td>
<td>0,0</td>
<td>10,7</td>
</tr>
<tr>
<td>1,0 – 0,75</td>
<td>0,0</td>
<td>11,6</td>
<td>0,0</td>
<td>11,0</td>
</tr>
<tr>
<td>0,75 – 0,50</td>
<td>0,8</td>
<td>26,5</td>
<td>0,5</td>
<td>26,0</td>
</tr>
<tr>
<td>0,50 – 0,315</td>
<td>3,9</td>
<td>20,0</td>
<td>3,4</td>
<td>28,4</td>
</tr>
<tr>
<td>0,315 – 0,0</td>
<td>95,3</td>
<td>29,5</td>
<td>96,1</td>
<td>21,8</td>
</tr>
<tr>
<td>Razem</td>
<td>100,0</td>
<td>100,0</td>
<td>100,0</td>
<td>100,0</td>
</tr>
</tbody>
</table>

**Fig. B.2. Report of measurement**
Fig. B.3. Report of measurement

Opróbowanie hydrocykliów Krebsa w dniu 20.12.2004r. Płuczka poz. 29m, godz. 9:05

<table>
<thead>
<tr>
<th>Zagęszczenie (g/l)</th>
<th>3.72 B32 system I wlew</th>
<th>3.72 B32 system I przelew</th>
<th>3.72 B31 system II wlew</th>
<th>3.72 B31 system II przelew</th>
</tr>
</thead>
<tbody>
<tr>
<td>Ad</td>
<td>727</td>
<td>41,2</td>
<td>698</td>
<td>76</td>
</tr>
</tbody>
</table>

### Analiza sitowa (%)

<table>
<thead>
<tr>
<th></th>
<th>3.72 B32 system I</th>
<th>3.72 B32 system I</th>
<th>3.72 B31 system II</th>
<th>3.72 B31 system II</th>
</tr>
</thead>
<tbody>
<tr>
<td>+ 1,8</td>
<td>3,5</td>
<td>0</td>
<td>1,2</td>
<td>0</td>
</tr>
<tr>
<td>1,8 – 1,0</td>
<td>20,2</td>
<td>1,0</td>
<td>12,7</td>
<td>0</td>
</tr>
<tr>
<td>1,0 – 0,75</td>
<td>11,3</td>
<td>1,9</td>
<td>7,2</td>
<td>0</td>
</tr>
<tr>
<td>0,75 – 0,50</td>
<td>25,0</td>
<td>3,4</td>
<td>16,8</td>
<td>0,5</td>
</tr>
<tr>
<td>0,50 – 0,315</td>
<td>21,2</td>
<td>15,5</td>
<td>19,9</td>
<td>3,2</td>
</tr>
<tr>
<td>0,315 – 0,0</td>
<td>18,8</td>
<td>78,2</td>
<td>42,2</td>
<td>96,3</td>
</tr>
<tr>
<td>Razem</td>
<td>100,0</td>
<td>100,0</td>
<td>100,0</td>
<td>100,0</td>
</tr>
</tbody>
</table>
Opróbowanie hydrocyklonów Krebsa w dniu 08.12.2004 r.
Płuczka poz. 29m, godz. 10:40 <br>Fig. B.4. Report of measurement

<table>
<thead>
<tr>
<th>Zagęszczenie (g/l)</th>
<th>Przelew 3.71.1 Dysza Ø 350mm</th>
<th>Wylew</th>
<th>Przelew 3.71.2 Dysza Ø 300mm</th>
<th>Wylew</th>
</tr>
</thead>
<tbody>
<tr>
<td>Ad</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>80</td>
<td>760</td>
<td>66</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td>719</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Analiza sitowa (%)</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>+ 1,8</td>
<td>0,0</td>
<td>2,0</td>
<td>0,0</td>
<td>2,2</td>
</tr>
<tr>
<td>1,8 - 1,0</td>
<td>0,0</td>
<td>9,2</td>
<td>0,0</td>
<td>10,6</td>
</tr>
<tr>
<td>1,0 - 0,75</td>
<td>1,0</td>
<td>10,1</td>
<td>0,1</td>
<td>11,1</td>
</tr>
<tr>
<td>0,75 - 0,50</td>
<td>2,1</td>
<td>29,8</td>
<td>0,3</td>
<td>26,1</td>
</tr>
<tr>
<td>0,50 - 0,315</td>
<td>9,6</td>
<td>21,1</td>
<td>2,0</td>
<td>26,5</td>
</tr>
<tr>
<td>0,315 - 0,0</td>
<td>87,3</td>
<td>27,8</td>
<td>97,6</td>
<td>23,3</td>
</tr>
<tr>
<td>Razem</td>
<td>100,0</td>
<td>100,0</td>
<td>100,0</td>
<td>100,0</td>
</tr>
</tbody>
</table>

Jastrzębie, 09.12.2004 r.

[Signature]

mgr inż. Zbigniew Danielewska
Pluczka poz. 29m, godz. 8.10
Opróbowanie hydrocyklonów Krebsa w dniu 22.12.2004r.

**Fig. B.5. Report of measurement**

<table>
<thead>
<tr>
<th>Zageśczenie (g/l)</th>
<th>3.72 B32 system I wylew</th>
<th>3.72 B32 system I przelew</th>
</tr>
</thead>
<tbody>
<tr>
<td>905</td>
<td>75</td>
<td></td>
</tr>
<tr>
<td>Ad</td>
<td>7.1</td>
<td>11.7</td>
</tr>
<tr>
<td>Analiza sitowa (%)</td>
<td></td>
<td></td>
</tr>
<tr>
<td>+ 1,8</td>
<td>8.6</td>
<td>0.0</td>
</tr>
<tr>
<td>1,8 – 1,0</td>
<td>12.3</td>
<td>0.0</td>
</tr>
<tr>
<td>1,0 – 0,75</td>
<td>10.4</td>
<td>0.0</td>
</tr>
<tr>
<td>0,75 – 0,50</td>
<td>20.3</td>
<td>0.9</td>
</tr>
<tr>
<td>0,50 – 0,315</td>
<td>18.5</td>
<td>3.9</td>
</tr>
<tr>
<td>0,315 – 0,0</td>
<td>29.9</td>
<td>95.2</td>
</tr>
<tr>
<td>Razem</td>
<td>100.0</td>
<td>100.0</td>
</tr>
</tbody>
</table>

Proce tylko jednego systemu hydrocyklonów
Opróbkowanie hydrocykłonów Krebsa w dniu 7.12.2004r.
Plucza poz. 29m, godz. 8:30

<table>
<thead>
<tr>
<th>Zgęszczenie (g/l)</th>
<th>Przelew 3.71.1 Dusaha Ø 350mm</th>
<th>Wylew Ø 250mm</th>
<th>Przelew 3.71.2 Dusaha Ø 300mm</th>
<th>Wylew Ø 250mm</th>
</tr>
</thead>
<tbody>
<tr>
<td>Ad</td>
<td>10,6</td>
<td>8,3</td>
<td>11,8</td>
<td>7,9</td>
</tr>
</tbody>
</table>

**Analiza sitowa (%)**

<table>
<thead>
<tr>
<th></th>
<th>Przelew 3.71.1 Dusaha Ø 350mm</th>
<th>Wylew Ø 250mm</th>
<th>Przelew 3.71.2 Dusaha Ø 300mm</th>
<th>Wylew Ø 250mm</th>
</tr>
</thead>
<tbody>
<tr>
<td>+ 1,8</td>
<td>0,0</td>
<td>2,2</td>
<td>0,0</td>
<td>1,6</td>
</tr>
<tr>
<td>1,8 – 1,0</td>
<td>3,0</td>
<td>12,0</td>
<td>0,0</td>
<td>10,9</td>
</tr>
<tr>
<td>1,0 – 0,75</td>
<td>3,9</td>
<td>14,3</td>
<td>0,1</td>
<td>12,5</td>
</tr>
<tr>
<td>0,75 – 0,50</td>
<td>4,1</td>
<td>29,5</td>
<td>0,4</td>
<td>24,6</td>
</tr>
<tr>
<td>0,50 – 0,315</td>
<td>9,7</td>
<td>25,9</td>
<td>2,6</td>
<td>24,5</td>
</tr>
<tr>
<td>0,315 – 0,0</td>
<td>79,3</td>
<td>16,1</td>
<td>96,9</td>
<td>25,9</td>
</tr>
</tbody>
</table>

**Razem**

| Przelew 3.71.1 Dusaha Ø 350mm | 100,0 | 100,0 |
| Przelew 3.71.2 Dusaha Ø 300mm | 100,0 | 100,0 |

Fig. B.6. Report of measurement
1. Zag (g/l) - 577
2. Analiza sitowa:

<table>
<thead>
<tr>
<th>Wymiary oczka (mm)</th>
<th>Udział (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1,2 – 1,0</td>
<td>7,3</td>
</tr>
<tr>
<td>1,0 – 0,75</td>
<td>10,5</td>
</tr>
<tr>
<td>0,75 – 0,50</td>
<td>27,4</td>
</tr>
<tr>
<td>0,50 – 0,315</td>
<td>25,9</td>
</tr>
<tr>
<td>0,315 – 0</td>
<td>28,9</td>
</tr>
<tr>
<td>Razem</td>
<td>100,0</td>
</tr>
</tbody>
</table>

Fig. B.7. Report of measurement
Opróbowanie hydrocyklonu Krebsa w dniu 6.12.2004r.  
Pluczka poz. 29m, godz. 11:00  — 33°C

<table>
<thead>
<tr>
<th></th>
<th>Przelew 3.71.1</th>
<th>Wylew 230μm</th>
<th>Przelew 3.71.2</th>
<th>Wylew 250μm</th>
</tr>
</thead>
<tbody>
<tr>
<td>Zagęszczenie (g/l)</td>
<td>90</td>
<td>780</td>
<td>89</td>
<td>675</td>
</tr>
<tr>
<td>Ad</td>
<td>44,7</td>
<td>9,3</td>
<td>12,0</td>
<td>44,6</td>
</tr>
<tr>
<td>Analiza sitowa (%)</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>+ 1,8</td>
<td>0,0</td>
<td>2,5</td>
<td>0,0</td>
<td>2,1</td>
</tr>
<tr>
<td>1,8 – 1,0</td>
<td>0,0</td>
<td>11,5</td>
<td>0,0</td>
<td>10,1</td>
</tr>
<tr>
<td>1,0 – 0,75</td>
<td>0,4</td>
<td>16,0</td>
<td>0,2</td>
<td>10,3</td>
</tr>
<tr>
<td>0,75 – 0,50</td>
<td>5,0</td>
<td>29,1</td>
<td>0,8</td>
<td>18,3</td>
</tr>
<tr>
<td>0,50 – 0,315</td>
<td>5,7</td>
<td>19,5</td>
<td>3,9</td>
<td>23,2</td>
</tr>
<tr>
<td>0,315 – 0,0</td>
<td>88,9</td>
<td>21,4</td>
<td>95,1</td>
<td>36,0</td>
</tr>
<tr>
<td>Razem</td>
<td><strong>100,0</strong></td>
<td><strong>100,0</strong></td>
<td><strong>100,0</strong></td>
<td><strong>100,0</strong></td>
</tr>
</tbody>
</table>

Fig. B.8. Report of measurement
Zestawienie wyników wylewów z hydrocyclonów KREBSA 3.72 B31 i 3.72 B32
PLUCZKA poz. 29 m

<table>
<thead>
<tr>
<th>Data pobrania</th>
<th>Zag (g/l)</th>
<th>Analiza sitowa (mm)</th>
<th>Razem</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>1,8</td>
<td>1,8–1,0</td>
</tr>
<tr>
<td>6.12.2004</td>
<td>780</td>
<td>2,5</td>
<td>11,5</td>
</tr>
<tr>
<td>6.12.2004</td>
<td>675</td>
<td>2,1</td>
<td>10,1</td>
</tr>
<tr>
<td>7.12.2004</td>
<td>673</td>
<td>2,2</td>
<td>12,0</td>
</tr>
<tr>
<td>7.12.2004</td>
<td>719</td>
<td>1,6</td>
<td>10,9</td>
</tr>
<tr>
<td>8.12.2004</td>
<td>760</td>
<td>2,0</td>
<td>9,2</td>
</tr>
<tr>
<td>8.12.2004</td>
<td>719</td>
<td>2,2</td>
<td>10,6</td>
</tr>
<tr>
<td>9.12.2004</td>
<td>765</td>
<td>1,9</td>
<td>9,9</td>
</tr>
<tr>
<td>9.12.2004</td>
<td>750</td>
<td>2,1</td>
<td>9,8</td>
</tr>
<tr>
<td>10.12.2004</td>
<td>707</td>
<td>2,1</td>
<td>10,3</td>
</tr>
<tr>
<td>10.12.2004</td>
<td>741</td>
<td>2,1</td>
<td>10,7</td>
</tr>
<tr>
<td>20.12.2004</td>
<td>727</td>
<td>3,5</td>
<td>20,2</td>
</tr>
<tr>
<td>20.12.2004</td>
<td>698</td>
<td>1,2</td>
<td>12,7</td>
</tr>
<tr>
<td>Średnia</td>
<td>725</td>
<td>2,1</td>
<td>11,5</td>
</tr>
<tr>
<td>*22.12.2004</td>
<td>905</td>
<td>8,6</td>
<td>12,3</td>
</tr>
</tbody>
</table>

* - praca tylko jednego systemu hydrocyclonów KREBSA

Wyniki wylewów z hydrocyclonów KREBSA 3.72 B31 i 3.72 B32
Próbę pobrano dnia 1.02.2005 r.
PLUCZKA poz. 29 m

1. Zag (g/l) - 577
2. Analiza sitowa:

<table>
<thead>
<tr>
<th>Wymiar oczka (mm)</th>
<th>Udział (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>+1,8</td>
<td>4,1</td>
</tr>
<tr>
<td>1,8–1,2</td>
<td>3,5</td>
</tr>
<tr>
<td>1,2–1,0</td>
<td>6,7</td>
</tr>
<tr>
<td>1,0–0,75</td>
<td>9,8</td>
</tr>
<tr>
<td>0,75–0,50</td>
<td>25,4</td>
</tr>
<tr>
<td>0,50–0,315</td>
<td>25,9</td>
</tr>
<tr>
<td>0,315–0</td>
<td>26,6</td>
</tr>
<tr>
<td>Razem</td>
<td>100</td>
</tr>
</tbody>
</table>

Fig. B.9. Report of measurement
Zestawienie wyników wylewów z hydrocyklioonów KREBSA 3.72 B31 i 3.72 B32
PLUCZKA poz. 29 m

<table>
<thead>
<tr>
<th>Data pobrania</th>
<th>Zag (g/l)</th>
<th>Analiza sitowa (mm)</th>
<th>Razem</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>1.8</td>
<td>1.8 – 1.0</td>
<td>1.0 – 0.75</td>
</tr>
<tr>
<td>6.12.2004</td>
<td>780</td>
<td>2.5</td>
<td>11.5</td>
</tr>
<tr>
<td>6.12.2004</td>
<td>675</td>
<td>2.1</td>
<td>10.1</td>
</tr>
<tr>
<td>7.12.2004</td>
<td>673</td>
<td>2.2</td>
<td>12.0</td>
</tr>
<tr>
<td>7.12.2004</td>
<td>710</td>
<td>1.6</td>
<td>10.9</td>
</tr>
<tr>
<td>8.12.2004</td>
<td>760</td>
<td>2.0</td>
<td>9.2</td>
</tr>
<tr>
<td>8.12.2004</td>
<td>719</td>
<td>2.2</td>
<td>10.6</td>
</tr>
<tr>
<td>9.12.2004</td>
<td>765</td>
<td>1.9</td>
<td>9.9</td>
</tr>
<tr>
<td>9.12.2004</td>
<td>750</td>
<td>2.1</td>
<td>9.8</td>
</tr>
<tr>
<td>10.12.2004</td>
<td>707</td>
<td>2.1</td>
<td>10.3</td>
</tr>
<tr>
<td>10.12.2004</td>
<td>741</td>
<td>2.1</td>
<td>10.7</td>
</tr>
<tr>
<td>20.12.2004</td>
<td>727</td>
<td>3.5</td>
<td>20.2</td>
</tr>
<tr>
<td>20.12.2004</td>
<td>698</td>
<td>1.2</td>
<td>12.7</td>
</tr>
<tr>
<td>Średnia</td>
<td>725</td>
<td>2.1</td>
<td>11.5</td>
</tr>
</tbody>
</table>
*a22.12.2004   | 905      | 8.6                 | 12.3    | 10.4     | 20.3      | 18.5     | 29.9     | 100 |

*a - praca tylko jednego systemu hydrocyklioonów KREBSA

Wyniki wylewów z hydrocyklioonów KREBSA 3.72 B31 i 3.72 B32
Próbę pobrano dnia 1.02.2005 r.
PLUCZKA poz. 29 m

1. Zag (g/l) - 577
2. Analiza sitowa:

<table>
<thead>
<tr>
<th>Wymiar oczka (mm)</th>
<th>Udział (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>+1.8</td>
<td>4.1</td>
</tr>
<tr>
<td>1.8 – 1.2</td>
<td>3.5</td>
</tr>
<tr>
<td>1.2 – 1.0</td>
<td>6.7</td>
</tr>
<tr>
<td>1.0 – 0.75</td>
<td>9.8</td>
</tr>
<tr>
<td>0.75 – 0.50</td>
<td>25.4</td>
</tr>
<tr>
<td>0.50 – 0.315</td>
<td>23.9</td>
</tr>
<tr>
<td>0.315 – 0</td>
<td>26.6</td>
</tr>
<tr>
<td>Razem</td>
<td>100</td>
</tr>
</tbody>
</table>

Fig. B.10. Report of measurement
Płuczka poz. 29 m. Nowe hydrocyklony KREBSA.
Dysza przelewowa Ø 350 mm.
Próby pobrano 15.11.2006 r.

<table>
<thead>
<tr>
<th>Wymiar ziaren w mm</th>
<th>Przelew</th>
<th>Wylew</th>
<th>Ad</th>
<th>Ad</th>
</tr>
</thead>
<tbody>
<tr>
<td>3,15</td>
<td>0,0</td>
<td>2,4</td>
<td></td>
<td></td>
</tr>
<tr>
<td>3,15 – 1,8</td>
<td>0,0</td>
<td>3,9</td>
<td></td>
<td></td>
</tr>
<tr>
<td>1,8 – 1,0</td>
<td>0,3</td>
<td>4,7</td>
<td></td>
<td></td>
</tr>
<tr>
<td>1,0 – 0,75</td>
<td>0,2</td>
<td>3,3</td>
<td></td>
<td></td>
</tr>
<tr>
<td>0,75 – 0,5</td>
<td>2,3</td>
<td>7,9</td>
<td></td>
<td></td>
</tr>
<tr>
<td>0,5 – 0,315</td>
<td>0,6</td>
<td>8,0</td>
<td></td>
<td></td>
</tr>
<tr>
<td>0,315 – 0,0</td>
<td>97,1</td>
<td>69,8</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Razem</td>
<td>100,0</td>
<td>100,0</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

Zag. (g/l) 84   851

Fig. B.11. Report of measurement
Płuczka poz. 29 m. Nowe hydrocyklony KREBSA.
Dysza przelewowa Ø 350 mm.
Próby pobrano 14.11.2006 r.

<table>
<thead>
<tr>
<th>Wymiary ziaren w mm</th>
<th>Przelew</th>
<th>Wylew</th>
</tr>
</thead>
<tbody>
<tr>
<td>3,15</td>
<td>0,0</td>
<td>0,5</td>
</tr>
<tr>
<td>3,15 – 1,8</td>
<td>0,0</td>
<td>2,0</td>
</tr>
<tr>
<td>1,8 – 1,0</td>
<td>0,3</td>
<td>10,4</td>
</tr>
<tr>
<td>1,0 – 0,75</td>
<td>0,2</td>
<td>10,0</td>
</tr>
<tr>
<td>0,75 – 0,5</td>
<td>0,3</td>
<td>8,1</td>
</tr>
<tr>
<td>0,5 – 0,315</td>
<td>1,6</td>
<td>6,1</td>
</tr>
<tr>
<td>0,315 – 0,0</td>
<td>97,6</td>
<td>16,4</td>
</tr>
<tr>
<td>Razem</td>
<td>100,0</td>
<td>100,0</td>
</tr>
</tbody>
</table>

Zag. (g/l) 58 684

Fig. B.12. Report of measurement
Opróbowanie hydrocyklonów Krebsa na Pluczce
Próby pobrano 22.06.2006r.

<table>
<thead>
<tr>
<th>Wymiar ziarna [mm]</th>
<th>Wyłew</th>
<th>Przelew</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Ø 33&quot;</td>
<td>Ø 26&quot;</td>
</tr>
<tr>
<td>+ 2,5</td>
<td>2,2</td>
<td>1,5</td>
</tr>
<tr>
<td>2,5 – 1,8</td>
<td>2,8</td>
<td>1,4</td>
</tr>
<tr>
<td>1,8 – 1,0</td>
<td>15,4</td>
<td>14,8</td>
</tr>
<tr>
<td>1,0 – 0,75</td>
<td>10,8</td>
<td>5,9</td>
</tr>
<tr>
<td>0,75 – 0,5</td>
<td>25,6</td>
<td>16,1</td>
</tr>
<tr>
<td>0,5 – 0,315</td>
<td>27,4</td>
<td>20,1</td>
</tr>
<tr>
<td>0,315 – 0,0</td>
<td>15,8</td>
<td>40,2</td>
</tr>
<tr>
<td>Razem</td>
<td>100,0 %</td>
<td>100,0%</td>
</tr>
</tbody>
</table>

Zag. g/1/Na dl 898/13,6 946/13,5 144/13,9 140/13,7

Opróbowanie nadawy na odmulnik DORR’A
ob.8 ( PLUCZKA-przelew z rzapia 3.710)
próby pobrano 22.06.2006r.

<table>
<thead>
<tr>
<th>Wymiar ziarna [mm]</th>
<th>S I</th>
<th>S II</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>%</td>
<td>%</td>
</tr>
<tr>
<td>+ 2,5</td>
<td>0,0</td>
<td>0,0</td>
</tr>
<tr>
<td>2,5 – 1,8</td>
<td>0,0</td>
<td>0,0</td>
</tr>
<tr>
<td>1,8 – 1,0</td>
<td>0,6</td>
<td>0,7</td>
</tr>
<tr>
<td>1,0 – 0,75</td>
<td>0,5</td>
<td>0,5</td>
</tr>
<tr>
<td>0,75 – 0,5</td>
<td>1,8</td>
<td>2,0</td>
</tr>
<tr>
<td>0,5 – 0,315</td>
<td>5,2</td>
<td>6,2</td>
</tr>
<tr>
<td>0,315 – 0,0</td>
<td>91,9</td>
<td>90,6</td>
</tr>
<tr>
<td>Razem</td>
<td>100,0</td>
<td>100,0</td>
</tr>
</tbody>
</table>

Zag. g/1/Na dl 156/13,2 164/13,6

Fig. B.13. Report of measurement
Zestawienie wyników opróbkowania hydrocyclonów Ø 33" - dysza przelewowa Ø 400 mm.
Pluczka, poz. 29 m. Próby pobrano 24-27.04.2006 r.

<table>
<thead>
<tr>
<th>wymiar oczka, mm</th>
<th>wylew 3.72 / B31</th>
<th>śr.</th>
<th>wylew 3.72 / B32</th>
<th>śr.</th>
<th>przelew</th>
<th>śr.</th>
</tr>
</thead>
<tbody>
<tr>
<td>+ 2,5</td>
<td>1,0</td>
<td>0,9</td>
<td>0,6</td>
<td>0,4</td>
<td>0,8</td>
<td>0,7</td>
</tr>
<tr>
<td>2,5 – 1,8</td>
<td>1,8</td>
<td>0,9</td>
<td>1,5</td>
<td>1,0</td>
<td>1,3</td>
<td>0,9</td>
</tr>
<tr>
<td>1,8 – 1,0</td>
<td>10,1</td>
<td>7,6</td>
<td>10,4</td>
<td>8,7</td>
<td>9,2</td>
<td>8,9</td>
</tr>
<tr>
<td>1,0 – 0,75</td>
<td>6,2</td>
<td>6,1</td>
<td>5,3</td>
<td>5,6</td>
<td>6,1</td>
<td>6,2</td>
</tr>
<tr>
<td>0,75 – 0,5</td>
<td>22,8</td>
<td>23,9</td>
<td>17,3</td>
<td>20,0</td>
<td>20,2</td>
<td>23,9</td>
</tr>
<tr>
<td>0,5 – 0,315</td>
<td>21,7</td>
<td>23,9</td>
<td>20,3</td>
<td>21,9</td>
<td>21,9</td>
<td>18,9</td>
</tr>
<tr>
<td>0,315 – 0,0</td>
<td>36,4</td>
<td>39,7</td>
<td>43,6</td>
<td>42,4</td>
<td>40,5</td>
<td>30,5</td>
</tr>
<tr>
<td>razem</td>
<td>100,0</td>
<td>100,0</td>
<td>120,0</td>
<td>100,0</td>
<td>100,0</td>
<td>100,0</td>
</tr>
<tr>
<td>Zag. (g/t)</td>
<td>650</td>
<td>788</td>
<td>523</td>
<td>823</td>
<td>696</td>
<td>904</td>
</tr>
<tr>
<td>Adi</td>
<td>–</td>
<td>11,8</td>
<td>14,3</td>
<td>15,3</td>
<td>16,8</td>
<td>16,9</td>
</tr>
</tbody>
</table>

Jastrzębie 28.04.2006 r.

Kierowniczy
Kontrola jakości produkcji
mgr inż. Zbigniew Danielowski
Płuczka, poz. 29 m. - Hydrocyklon Ø 33" - dysza przelewowa Ø 400 mm
Próby pobrano 26.04.2004 r.

**Table: Report of measurement**

<table>
<thead>
<tr>
<th>Wymiar oczka, mm</th>
<th>Wylew 3.72 / B31</th>
<th>Wylew 3.72 / B32</th>
<th>Przelew</th>
</tr>
</thead>
<tbody>
<tr>
<td>+ 2.5</td>
<td>0.6</td>
<td>0.8</td>
<td>0.0</td>
</tr>
<tr>
<td>2.5 – 1.8</td>
<td>1.5</td>
<td>1.3</td>
<td>0.0</td>
</tr>
<tr>
<td>1.8 – 1.0</td>
<td>10.4</td>
<td>8.3</td>
<td>0.0</td>
</tr>
<tr>
<td>1.0 – 0.75</td>
<td>6.3</td>
<td>5.6</td>
<td>0.0</td>
</tr>
<tr>
<td>0.75 – 0.5</td>
<td>17.3</td>
<td>17.4</td>
<td>1.0</td>
</tr>
<tr>
<td>0.5 – 0.315</td>
<td>20.3</td>
<td>24.3</td>
<td>3.3</td>
</tr>
<tr>
<td>0.315 – 0.0</td>
<td>43.6</td>
<td>42.3</td>
<td>95.7</td>
</tr>
<tr>
<td>razem</td>
<td>100.0</td>
<td>100.0</td>
<td>100.0</td>
</tr>
<tr>
<td>Zag. (g/l)</td>
<td>523</td>
<td>460</td>
<td>60</td>
</tr>
</tbody>
</table>

Fig. B.15. Report of measurement
Płuczka, poz. 29 m. - Hydrocyklon Ø 33” - dysza przelewowa Ø 400 mm  
Próby pobrano 25.04.2004 r.

**Fig. B.16. Report of measurement**

<table>
<thead>
<tr>
<th>Wymiar oczka, mm</th>
<th>Wylew 3.72 / B31</th>
<th>Wylew 3.72 / B32</th>
<th>Przelew</th>
</tr>
</thead>
<tbody>
<tr>
<td>+ 2,5</td>
<td>0,9</td>
<td>0,8</td>
<td>0,0</td>
</tr>
<tr>
<td>2,5 – 1,8</td>
<td>0,9</td>
<td>1,1</td>
<td>0,0</td>
</tr>
<tr>
<td>1,8 – 1,0</td>
<td>7,6</td>
<td>8,7</td>
<td>0,0</td>
</tr>
<tr>
<td>1,0 – 0,75</td>
<td>6,1</td>
<td>6,0</td>
<td>0,0</td>
</tr>
<tr>
<td>0,75 – 0,5</td>
<td>20,9</td>
<td>21,3</td>
<td>1,2</td>
</tr>
<tr>
<td>0,5 – 0,315</td>
<td>23,9</td>
<td>21,8</td>
<td>6,1</td>
</tr>
<tr>
<td>0,315 – 0,0</td>
<td>39,7</td>
<td>40,3</td>
<td>92,7</td>
</tr>
<tr>
<td>razem</td>
<td>100,0</td>
<td>100,0</td>
<td>100,0</td>
</tr>
<tr>
<td>Zag. (g/l)</td>
<td>788</td>
<td>800</td>
<td>122</td>
</tr>
</tbody>
</table>
Fig. B.17. Report of measurement

Wyniki prób pobranych w dniu 04.11.2009 r. po starcie zakładu przerobczego.

<table>
<thead>
<tr>
<th>Taśma 5.32 godz. 13:07</th>
<th>Analizator</th>
<th>Suszarka</th>
</tr>
</thead>
<tbody>
<tr>
<td>Ad – 7,7% Wtr – 22,1%</td>
<td>Wtr – 22,5%</td>
<td></td>
</tr>
<tr>
<td>Taśma 5.32 godz. 13:09</td>
<td>Ad – 7,8% Wtr – 22,4%</td>
<td>Wtr – 24,0%</td>
</tr>
<tr>
<td>Taśma 5.32 godz. 13:16</td>
<td>Ad – 12,1% Wtr – 21,1%</td>
<td>Wtr – 21,7%</td>
</tr>
<tr>
<td>Taśma 5.32 godz. 13:19</td>
<td>Ad – 12,2% Wtr – 17,5%</td>
<td>Wtr – 16,9%</td>
</tr>
<tr>
<td>Taśma 5.32 godz. 13:22</td>
<td>Ad – 14,7% Wtr – 21,7%</td>
<td>Wtr – 19,7%</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Wymiar oczek (mm)</th>
<th>Udział (%)</th>
<th>Udział (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Ob. 3 Hydr. D-33 wylew φ 15 ″</td>
<td>Ob. 3 Hydr. D-33 przelew</td>
</tr>
<tr>
<td></td>
<td>zag. 510 g/l</td>
<td>zag. 40 g/l</td>
</tr>
<tr>
<td>+ 3,15</td>
<td>0,8</td>
<td>-</td>
</tr>
<tr>
<td>3,15 – 2,5</td>
<td>1,2</td>
<td>-</td>
</tr>
<tr>
<td>2,5 – 1,8</td>
<td>2,7</td>
<td>-</td>
</tr>
<tr>
<td>1,8 – 1,0</td>
<td>5,9</td>
<td>-</td>
</tr>
<tr>
<td>1,0 – 0,5</td>
<td>35,3</td>
<td>0,5</td>
</tr>
<tr>
<td>0,5 – 0,2</td>
<td>27,5</td>
<td>1,8</td>
</tr>
<tr>
<td>0,2 – 0,071</td>
<td>19,2</td>
<td>9,2</td>
</tr>
<tr>
<td>0,071 – 0,0</td>
<td>7,4</td>
<td>88,5</td>
</tr>
<tr>
<td>Razem</td>
<td>100,0</td>
<td>100,0</td>
</tr>
</tbody>
</table>
Wyniki opróbkowania hydrocyklonów Krebs, Płucza poz. 22 m i 29 m.

**Próby pobrano 13 i 14.10.2009 r.**

<table>
<thead>
<tr>
<th></th>
<th></th>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Nr HC</td>
<td>1 2 3 4</td>
<td>1 2 3 4</td>
<td>1 2 3 4</td>
</tr>
<tr>
<td>Zag., g/l</td>
<td>900 810 622 776</td>
<td>850 880 900 900</td>
<td>730 835 768 712</td>
</tr>
<tr>
<td>Ad. %</td>
<td>17.4 17.3 13.7 13.6</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

**An. sitowa**

<table>
<thead>
<tr>
<th></th>
<th>+2.0</th>
<th>2.0 – 1.0</th>
<th>1.0 – 0.5</th>
<th>&lt;0.5</th>
</tr>
</thead>
<tbody>
<tr>
<td>Gromadziwa</td>
<td>3.3  2.1  2.0  1.9</td>
<td>-  -  -  -</td>
<td>1.5  1.6  2.3  1.7</td>
<td></td>
</tr>
<tr>
<td>2.0 – 1.0</td>
<td>8.9  9.4  11.1 11.2</td>
<td>-  -  -  -</td>
<td>10.7 15.7 11.6 14.0</td>
<td></td>
</tr>
<tr>
<td>1.0 – 0.5</td>
<td>35.6 35.2 38.3 37.8</td>
<td>-  -  -  -</td>
<td>43.6 43.8 39.8 41.7</td>
<td></td>
</tr>
<tr>
<td>&lt;0.5</td>
<td>52.2 53.3 48.6 49.1</td>
<td>-  -  -  -</td>
<td>44.2 38.9 46.3 42.6</td>
<td></td>
</tr>
<tr>
<td>Razem</td>
<td>100.0 100.0 100.0 100.0</td>
<td>-  -  -  -</td>
<td>100.0 100.0 100.0 100.0</td>
<td></td>
</tr>
</tbody>
</table>

**PRZELEW**

<table>
<thead>
<tr>
<th></th>
<th></th>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Nr HC</td>
<td>1+2 3+4</td>
<td>1+2 3+4</td>
<td>1+2 3+4</td>
</tr>
<tr>
<td>Zag., g/l</td>
<td>58 68 63 47</td>
<td>96 66</td>
<td></td>
</tr>
<tr>
<td>Ad. %</td>
<td>17.0 17.7</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

**An. sitowa**

<table>
<thead>
<tr>
<th></th>
<th>+2.0</th>
<th>2.0 – 1.0</th>
<th>1.0 – 0.5</th>
<th>0.5 - 0.315</th>
<th>&lt;0.315</th>
</tr>
</thead>
<tbody>
<tr>
<td>Gromadziwa</td>
<td>0.0  0.0  -  -  -  -  97.8  99.1</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>2.0 – 1.0</td>
<td>0.3  0.1  0.2  -  0.4  -  7.5  5.7</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>1.0 – 0.5</td>
<td>1.9  0.8  2.1  0.9  1.3  0.9  7.5  5.7</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>0.5 – 0.315</td>
<td>-  -  -  -  -  -  90.2  93.4</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>&lt;0.315</td>
<td>99.1 94.5 96.1</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Razem</td>
<td>100.0 100.0 100.0 100.0 100.0 100.0</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

Fig. B.18. Report of measurement
Płuczka, poz. 29 m. - Hydrocyklon Ø 33” - dysza przelewowa Ø 400 mm
Próby pobrano 24.04.2004 r.

![Graph and Table](image)

Fig. B.19. Report of measurement
### Wyniki opróbowania Płuczki w dn. 21.03.2006r. zm A

Jastrzębie 23.03.2006r.

<table>
<thead>
<tr>
<th>Miejsce pobrania</th>
<th>Przestawacz 3.52.1 pr górny</th>
<th>Przestawacz 3.52.3 pr górny</th>
<th>Wylot z hydrocykul, 26m³</th>
<th>Wylot z krebsów 32m³</th>
<th>Odsącz z przestawaczy 3.52.1 i 3.52.3</th>
<th>Odsącz zbiorczy z odwadniarza poz. 12m</th>
</tr>
</thead>
<tbody>
<tr>
<td>zagęszczenia g/l</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>wyniarka oczka</td>
<td>%</td>
<td>%</td>
<td>%</td>
<td>%</td>
<td>%</td>
<td>%</td>
</tr>
<tr>
<td>[mm]</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>&gt; 2,5</td>
<td>0,5</td>
<td>9,4</td>
<td>0,5</td>
<td>11,2</td>
<td>0,0</td>
<td>2,2</td>
</tr>
<tr>
<td>2,5 - 1,8</td>
<td>1,9</td>
<td>6,4</td>
<td>2,4</td>
<td>6,2</td>
<td>0,0</td>
<td>11,7</td>
</tr>
<tr>
<td>1,8 - 1,0</td>
<td>15,1</td>
<td>7,2</td>
<td>18,2</td>
<td>7,4</td>
<td>0,7</td>
<td>22,3</td>
</tr>
<tr>
<td>1,0 - 0,75</td>
<td>11,0</td>
<td>7,3</td>
<td>11,8</td>
<td>8,4</td>
<td>3,2</td>
<td>13,3</td>
</tr>
<tr>
<td>0,75 - 0,5</td>
<td>22,6</td>
<td>10,3</td>
<td>23,2</td>
<td>10,1</td>
<td>14,7</td>
<td>16,0</td>
</tr>
<tr>
<td>0,5 - 0,315</td>
<td>22,9</td>
<td>21,5</td>
<td>22,0</td>
<td>16,0</td>
<td>27,8</td>
<td>8,6</td>
</tr>
<tr>
<td>0,315 - 0</td>
<td>26,0</td>
<td>14,4</td>
<td>21,9</td>
<td>27,3</td>
<td>53,6</td>
<td>25,9</td>
</tr>
<tr>
<td>razem</td>
<td>100,0</td>
<td>13,1</td>
<td>100,0</td>
<td>14,4</td>
<td>100,0</td>
<td>100,0</td>
</tr>
</tbody>
</table>
Wyniki opróbowania hydrocyclonów Krebsa, ob. 3, poz. 29m  

Próby pobrano 05.11.2008 r.

<table>
<thead>
<tr>
<th>Nr HC</th>
<th>WYLEW</th>
<th>PRZELEW</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>1</td>
<td>2</td>
</tr>
<tr>
<td>Zag. g/l</td>
<td>642</td>
<td>292</td>
</tr>
<tr>
<td>Ad. %</td>
<td>10,0</td>
<td>19,0</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Wymiar ziem, mm</th>
<th>Udział %</th>
<th>Ad. %</th>
<th>Udział %</th>
<th>Ad. %</th>
<th>Udział %</th>
<th>Ad. %</th>
<th>Udział %</th>
<th>Ad. %</th>
</tr>
</thead>
<tbody>
<tr>
<td>&gt; 3,15</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>3,15 – 1,0</td>
<td>30,2</td>
<td>6,0</td>
<td>16,4</td>
<td>11,1</td>
<td>12,1</td>
<td>5,5</td>
<td>3,1</td>
<td>13,6</td>
</tr>
<tr>
<td>1,0 – 0,5</td>
<td>46,3</td>
<td>7,0</td>
<td>32,2</td>
<td>9,9</td>
<td>28,2</td>
<td>4,9</td>
<td>9,2</td>
<td>12,6</td>
</tr>
<tr>
<td>&lt; 0,5</td>
<td>23,5</td>
<td>22,8</td>
<td>51,4</td>
<td>28,1</td>
<td>59,7</td>
<td>15,3</td>
<td>87,7</td>
<td>19,5</td>
</tr>
<tr>
<td>RAZEM</td>
<td>100,0</td>
<td>-</td>
<td>100,0</td>
<td>-</td>
<td>100,0</td>
<td>-</td>
<td>100,0</td>
<td>-</td>
</tr>
</tbody>
</table>

Fig. B.21. Report of measurement
### Table 1: Measurement Report

<table>
<thead>
<tr>
<th>Wymiar oczek (mm)</th>
<th>Udział (%)</th>
<th>Udział (%)</th>
<th>Udział (%)</th>
<th>Udział (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Ob. 3 Hydr. D-33 wylew d HEC</td>
<td>Ob. 5 Hydr. D-15 wylew</td>
<td>zag. 510 g/l Ad-10,1%</td>
<td>zag. 340 g/l Ad-10,4%</td>
</tr>
<tr>
<td>+ 3,15</td>
<td>0,02</td>
<td>0,03</td>
<td></td>
<td></td>
</tr>
<tr>
<td>3,15–2,5</td>
<td>0,02</td>
<td>0,03</td>
<td></td>
<td></td>
</tr>
<tr>
<td>2,5–1,8</td>
<td>0,02</td>
<td>0,18</td>
<td></td>
<td></td>
</tr>
<tr>
<td>1,8–1,0</td>
<td>1,33</td>
<td>2,18</td>
<td></td>
<td></td>
</tr>
<tr>
<td>1,0–0,5</td>
<td>59,26</td>
<td>41,53</td>
<td></td>
<td></td>
</tr>
<tr>
<td>0,5–0,2</td>
<td>20,90</td>
<td>24,82</td>
<td></td>
<td></td>
</tr>
<tr>
<td>0,2–0,071</td>
<td>12,20</td>
<td>27,76</td>
<td></td>
<td></td>
</tr>
<tr>
<td>0,071–0,0</td>
<td>6,25</td>
<td>3,47</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Razem</td>
<td>100,0</td>
<td>100,0</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Wymiar oczek (mm)</th>
<th>Udział (%) 5.11 godz. 8:02</th>
<th>Udział (%) 5.11 godz. 8:07</th>
<th>Udział (%) 5.11 godz. 8:12</th>
<th>Udział (%) 5.11 godz. 8:17</th>
</tr>
</thead>
<tbody>
<tr>
<td>+ 3,15</td>
<td>2,7</td>
<td>30,7</td>
<td>9,5</td>
<td>5,4</td>
</tr>
<tr>
<td>3,15–2,5</td>
<td>1,1</td>
<td>3,7</td>
<td>1,6</td>
<td>1,7</td>
</tr>
<tr>
<td>2,5–1,0</td>
<td>12,9</td>
<td>13,8</td>
<td>13,6</td>
<td>21,0</td>
</tr>
<tr>
<td>1,0–0,75</td>
<td>12,3</td>
<td>7,8</td>
<td>11,3</td>
<td>15,7</td>
</tr>
<tr>
<td>0,75–0,5</td>
<td>32,3</td>
<td>13,2</td>
<td>21,5</td>
<td>26,5</td>
</tr>
<tr>
<td>0,5–0,315</td>
<td>22,1</td>
<td>15,7</td>
<td>24,3</td>
<td>16,8</td>
</tr>
<tr>
<td>0,315–0,2</td>
<td>6,3</td>
<td>6,5</td>
<td>7,9</td>
<td>6,0</td>
</tr>
<tr>
<td>0,2–0,1</td>
<td>7,0</td>
<td>5,0</td>
<td>7,0</td>
<td>5,5</td>
</tr>
<tr>
<td>0,1–0,0</td>
<td>3,3</td>
<td>2,7</td>
<td>3,3</td>
<td>2,3</td>
</tr>
<tr>
<td>Razem</td>
<td>100,0</td>
<td>100,0</td>
<td>100,0</td>
<td>100,0</td>
</tr>
</tbody>
</table>

**Fig. B.22.** Report of measurement
Wyniki opróbowania hydrocyklonów Krebs. Plucza poz. 29 m.

Próby pobrano 30.11.2006 r.

<table>
<thead>
<tr>
<th>Nr HC</th>
<th>przelew</th>
<th>wylew 1 (od PF)</th>
<th>wylew 3</th>
<th>wylew 4 (od PK)</th>
</tr>
</thead>
<tbody>
<tr>
<td>DYSZA (mm)</td>
<td>przelewowa</td>
<td>350</td>
<td>400</td>
<td>350</td>
</tr>
<tr>
<td></td>
<td>wylewowa</td>
<td>175</td>
<td>230</td>
<td>175</td>
</tr>
<tr>
<td>zagęszczenie (g/l)</td>
<td>2888</td>
<td>781</td>
<td>902</td>
<td>788</td>
</tr>
<tr>
<td>Ad (%)</td>
<td>4,7</td>
<td>18,8</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

Analiza sitowa

<table>
<thead>
<tr>
<th>wymiar ziarna (mm)</th>
<th>% Ad</th>
<th>% Ad</th>
<th>% Ad</th>
<th>% Ad</th>
</tr>
</thead>
<tbody>
<tr>
<td>+3,15</td>
<td>0,0</td>
<td>2,0</td>
<td>6,7</td>
<td>19,1</td>
</tr>
<tr>
<td>3,15–1,8</td>
<td>0,3</td>
<td>12,9</td>
<td>73,4</td>
<td>3,5</td>
</tr>
<tr>
<td>1,8–1,0</td>
<td>2,6</td>
<td>59,8</td>
<td>21,6</td>
<td>20,3</td>
</tr>
<tr>
<td>1,0–0,75</td>
<td>1,7</td>
<td>6,2</td>
<td>1,7</td>
<td>11,4</td>
</tr>
<tr>
<td>0,75–0,5</td>
<td>4,2</td>
<td>6,3</td>
<td>10,3</td>
<td>22,4</td>
</tr>
<tr>
<td>0,5–0,315</td>
<td>5,5</td>
<td>9,5</td>
<td>19,8</td>
<td>17,5</td>
</tr>
<tr>
<td>0,315–0,0</td>
<td>85,7</td>
<td>4,3</td>
<td>8,1</td>
<td>24,7</td>
</tr>
<tr>
<td>Razem</td>
<td>100,0</td>
<td>100,0</td>
<td>100,0</td>
<td>100,0</td>
</tr>
</tbody>
</table>

Fig. B.23. Report of measurement
Wyniki opróbowania hydrocyklonów Krebs. Płuczka poz. 29 m.
Próby pobrano 29.11.2006 r.  

<table>
<thead>
<tr>
<th>Nr HC</th>
<th>przelew</th>
<th>wylewy</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>3+4</td>
<td>1 (od PF)</td>
</tr>
<tr>
<td>DYSZA (mm)</td>
<td>przelewowa</td>
<td>350</td>
</tr>
<tr>
<td></td>
<td>wylewowa</td>
<td>175</td>
</tr>
<tr>
<td>zagęszczenie (g/l)</td>
<td>78</td>
<td>560</td>
</tr>
<tr>
<td>Ad (%)</td>
<td>15,3</td>
<td>17,4</td>
</tr>
</tbody>
</table>

Analiza ситова

<table>
<thead>
<tr>
<th>wymiar ziarna (mm)</th>
<th>% Ad</th>
<th>% Ad</th>
<th>% Ad</th>
<th>% Ad</th>
<th>% Ad</th>
<th>% Ad</th>
</tr>
</thead>
<tbody>
<tr>
<td>+3,15</td>
<td>0,0</td>
<td>0,7</td>
<td>6,7</td>
<td>1,1</td>
<td>9,0</td>
<td>2,0</td>
</tr>
<tr>
<td>3,15–1,8</td>
<td>0,0</td>
<td>4,8</td>
<td>7,7</td>
<td>5,7</td>
<td>7,7</td>
<td>12,9</td>
</tr>
<tr>
<td>1,8–1,0</td>
<td>0,0</td>
<td>3,8</td>
<td>7,4</td>
<td>26,7</td>
<td>7,8</td>
<td>50,2</td>
</tr>
<tr>
<td>1,0–0,75</td>
<td>21,4</td>
<td>4,9</td>
<td>11,5</td>
<td>3,9</td>
<td>8,4</td>
<td>10,0</td>
</tr>
<tr>
<td>0,75–0,5</td>
<td>1,0</td>
<td>3,6</td>
<td>10,3</td>
<td>10,8</td>
<td>15,5</td>
<td>6,8</td>
</tr>
<tr>
<td>0,5–0,315</td>
<td>5,5</td>
<td>4,4</td>
<td>12,7</td>
<td>7,4</td>
<td>11,4</td>
<td>11,2</td>
</tr>
<tr>
<td>0,315–0,0</td>
<td>93,5</td>
<td>18,3</td>
<td>22,9</td>
<td>26,6</td>
<td>27,5</td>
<td>7,8</td>
</tr>
<tr>
<td>Razem</td>
<td>100,0</td>
<td>100,0</td>
<td>100,0</td>
<td>100,0</td>
<td>100,0</td>
<td>100,0</td>
</tr>
</tbody>
</table>

Fig. B.24. Report of measurement
### Opróbowanie hydrocyclonów D.33.
Próby pobrano 04.02.2008r.

<table>
<thead>
<tr>
<th>Hydrocyclon przelew D.33 Ø 175</th>
<th>Hydrocyclon przelew D.33 Ø 165</th>
<th>Hydrocyclon wylew D.33 Ø 175</th>
<th>Hydrocyclon wylew D.33 Ø 165</th>
<th>Nadawa na flotację</th>
</tr>
</thead>
<tbody>
<tr>
<td>zag./Ad 50 g/l 8,7%</td>
<td>zag./Ad 612 g/l 26,6%</td>
<td>zag./Ad 730 g/l 28,5%</td>
<td>zag./Ad 58 g/l 4,8%</td>
<td>zag./Ad 73 g/l 16,4%</td>
</tr>
<tr>
<td>wym. oczek % Ad</td>
<td>wym. oczek % Ad</td>
<td>wym. oczek % Ad</td>
<td>wym. oczek % Ad</td>
<td>wym. oczek % Ad</td>
</tr>
<tr>
<td>3,15</td>
<td>3,15</td>
<td>3,15</td>
<td>3,15</td>
<td>1,6 - 1,25</td>
</tr>
<tr>
<td>-</td>
<td>1,0</td>
<td>0,6</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>3,15 - 1,8</td>
<td>28,7</td>
<td>9,1</td>
<td>2,6</td>
<td>3,15 - 1,8</td>
</tr>
<tr>
<td>-</td>
<td>1,8 - 1,0</td>
<td>22,3</td>
<td>8,1</td>
<td>1,25 - 1,0</td>
</tr>
<tr>
<td>1,0 - 0,75</td>
<td>2,7</td>
<td>1,0 - 0,75</td>
<td>0,75 - 0,5</td>
<td>0,8 - 0,63</td>
</tr>
<tr>
<td>0,75 - 0,5</td>
<td>2,6</td>
<td>2,6</td>
<td>0,75 - 0,5</td>
<td>0,4</td>
</tr>
<tr>
<td>0,5 - 0,315</td>
<td>4,9</td>
<td>1,8</td>
<td>8,4</td>
<td>19,9</td>
</tr>
<tr>
<td>0,315 - 0,0</td>
<td>58,4</td>
<td>0,75 - 0,5</td>
<td>15,0</td>
<td>63 - 0,5</td>
</tr>
<tr>
<td>Razem</td>
<td>100,0</td>
<td>100,0</td>
<td>100,0</td>
<td>13,2</td>
</tr>
</tbody>
</table>

### Odpady osadz. nr. 4

<table>
<thead>
<tr>
<th>% Ad</th>
<th>% Ad</th>
</tr>
</thead>
<tbody>
<tr>
<td>W</td>
<td>0,7</td>
</tr>
<tr>
<td>P</td>
<td>2,7</td>
</tr>
<tr>
<td>K</td>
<td>96,6</td>
</tr>
</tbody>
</table>
Fig. B.26. Report of measurement
Wyniki analiz sitowych i zagęszczeń.

Próby pobrane dn. 19.08.2008r. zm. A

<table>
<thead>
<tr>
<th>Wymiary [mm]</th>
<th>Hydrocyklon poz.29m ob.nr 3 wylew 3,78</th>
<th>Nadawa na flotację Próba pobr. Ob.3 poz.6m Rurociąg ssący pompę 3,78</th>
<th>Hydrocyklon poz.24m ob.nr 5 wylew</th>
<th>Hydrocyklon poz.24m ob.nr 5 przelew</th>
</tr>
</thead>
<tbody>
<tr>
<td>+ 3,15</td>
<td>Udzielenie %</td>
<td>Ad %</td>
<td>Udzielenie %</td>
<td>Ad %</td>
</tr>
<tr>
<td>3,15 - 2,0</td>
<td>1,7</td>
<td>7.7</td>
<td>1,5</td>
<td>9.2</td>
</tr>
<tr>
<td>2,0 - 1,0</td>
<td>7,5</td>
<td>51,4</td>
<td>14,4</td>
<td>51,8</td>
</tr>
<tr>
<td>1,0 - 0,75</td>
<td>31,0</td>
<td>41,5</td>
<td>33,7</td>
<td>41,2</td>
</tr>
<tr>
<td>0,75 - 0,50</td>
<td>16,4</td>
<td>51,3</td>
<td>15,3</td>
<td>4,7</td>
</tr>
<tr>
<td>0,5 - 0,315</td>
<td>24,4</td>
<td>61,6</td>
<td>11,6</td>
<td>4,7</td>
</tr>
<tr>
<td>0,315 - 0,0</td>
<td>14,0</td>
<td>14,7</td>
<td>12,9</td>
<td>5,2</td>
</tr>
<tr>
<td>Razem</td>
<td>100,0</td>
<td>100,0</td>
<td>100,0</td>
<td>100,0</td>
</tr>
<tr>
<td>Zag. g/l</td>
<td>655,5</td>
<td>71,7</td>
<td>166,2</td>
<td>8,2</td>
</tr>
</tbody>
</table>

Fig. B.27. Report of measurement
Appendix C. Graphic presentation of CFD results

Fig. C.1. Geometry of cyclone prepared in GAMBIT program

Fig. C.2. Sample of generated mesh
Fig. C.3. Second sample of generated mesh

Fig. C.4. Close-up of cyclone’s geometry with mesh

Fig. C.5. Presentation of cyclone’s geometry in different views
Fig. C.6. Cyclone’s geometry

Fig. C.7. Pressure of mixture (vertical cross-section)

Fig. C.8. Pressure of mixture – only positive values
Fig. C.9. Density of mixture

Fig. C.10. Density of mixture on vertical plane

Fig. C.11. Density of mixture on 3 horizontal planes
Fig. C.12. Sample particle trace coloured by residence time

Fig. C.13. Sample particle trace coloured by residence time

Fig. C.14. Sample particle trace coloured by residence time
Fig. C.15. Sample particle trace coloured by residence time

Fig. C.16. Sample particle trace coloured by residence time

Fig. C.17. Volume fraction of phase 1 (dense medium)
Fig. C.18. Volume fraction of phase 2 (coal)

Fig. C.19. Volume fraction of phase 3 (coal)

Fig. C.20. Volume fraction of phase 4 (coal)
Fig. C.21. Volume fraction of phase 5 (coal)

Fig. C.22. Volume fraction of phase 6 (coal)

Fig. C.23. Volume fraction of phase 7 (coal)
Fig. C.24. Volume fraction of phase 8 (coal)

Fig. C.25. Volume fraction of phase 9 (coal)

Fig. C.26. Volume fraction of phase 10 (coal)
Fig. C.27. Volume fraction of phase 11 (ash)

Fig. C.28. Volume fraction of phase 12 (ash)

Fig. C.29. Volume fraction of phase 13 (ash)
Fig. C.30. Volume fraction of phase 14 (ash)

Fig. C.31. Volume fraction of phase 15 (ash)

Fig. C.32. Volume fraction of phase 16 (ash)
Fig. C.33. Volume fraction of phase 17 (ash)

Fig. C.34. Volume fraction of phase 18 (ash)

Fig. C.35. Volume fraction of phase 19 (ash)
Fig. C.36. Value of velocity of mixture on horizontal plane

Fig. C.37. Value of velocity of mixture on horizontal plane

Fig. C.38. Value of velocity (vertical, only positive values)
Fig. C.39. Value of velocity (vertical, only negative values)

Fig. C.40. Values of absolute pressure of mixture

Fig. C.41. Values of dynamic pressure of mixture
Fig. C.42. Values of static pressure of mixture

Fig. C.43. Values of total pressure of mixture

Fig. C.44. Values of turbulent kinetic energy (horizontal plane)
Fig. C.45. Values of turbulent kinetic energy (vertical plane)

Fig. C.46. Velocity magnitude of mixture

Fig. C.47. Velocity vectors coloured by vertical velocity value
Fig. C.48. Velocity vectors coloured by velocity value

Fig. C.49. Vertical velocity value (only negative values)

Fig. C.50. Vertical velocity value (only positive values)
Fig. C.51. Volume fraction of phase 1 (dense medium)

Fig. C.52. Volume fraction of phase 2 (coal)

Fig. C.53. Volume fraction of phase 3 (coal)
Fig. C.54. Volume fraction of phase 4 (coal)

Fig. C.55. Volume fraction of phase 5 (coal)

Fig. C.56. Volume fraction of phase 6 (coal)
Fig. C.57. Volume fraction of phase 7 (coal)

Fig. C.58. Volume fraction of phase 8 (coal)

Fig. C.59. Volume fraction of phase 9 (coal)
Fig. C.60. Volume fraction of phase 10 (coal)

Fig. C.61. Volume fraction of phase 11 (ash)

Fig. C.62. Volume fraction of phase 12 (ash)
Fig. C.63. Volume fraction of phase 13 (ash)

Fig. C.64. Volume fraction of phase 14 (ash)

Fig. C.65. Volume fraction of phase 15 (ash)
Fig. C.66. Volume fraction of phase 16 (ash)

Fig. C.67. Volume fraction of phase 17 (ash)

Fig. C.68. Volume fraction of phase 18 (ash)
Fig. C.69. Volume fraction of phase 19 (ash)

Fig. C.70. Vectors of horizontal velocity coloured by value

Fig. C.71. Vectors of vertical velocity coloured by value