MATERIAL HANDLING LABORATORY

LAB MANUAL

List of Experiments

1. To determine the average particle size of a mixture of particles by sieve analysis
2. To Study and operation of Jaw crusher and thereby verification of Rittinger's constant and to determine the reduction ratio, maximum feed size and theoretical capacity of crushing rolls
3. To find out the enrichment of the coal sample by forth floatation cell
4. To find out the effectiveness of a vibrating screen.
5. To study the operation of a Hammer mill and a pulveriser and finding their reduction ratio
6. To study the operation of a cyclone separator and finding their efficiency
7. To determine the angle of repose of sand particles using sieve analysis
8. To study the operation of magnetic separator and finding their efficiency.
9. To Study and operation of Roll crusher and thereby verification of Rittinger's constant and to determine the reduction ratio, maximum feed size and theoretical capacity of crushing rolls

EXPERIMENT: 1 SIEVE ANALYSIS

AIM OF THE EXPERIMENT:-

To determine the average particle size of a mixture of particles by Sieve analysis.

APPARATUS REQUIRED:-

4-200 mesh BSS Sieves, Sieve shaker, brush, sieve opener, balance with weight box

THEORY:-

T he average particle size is defined in several ways. The most common method being the volume-surface mean diameter.

(1)Volume –surface mean diameter:-

$D_{VS}= \frac{6}{\Phi_S \rho_P A_W} = 1/\sum_{i=1}^{n} xi/Dpi$

Where Φ_s=sphericity

A_W=Total Surface Area

ρ_p=density of particle

X_i=i th mass fraction

$\bar{D}_{pi}$=ith average particle diameter

(2) Arithmetic mean diameter:-

$D_N=\sum X_{pi}/D_{pi}^2/\sum X_{pi}/D_{pi}^3$

(3) Mass mean diameter:-

$D_w=\sum_{i=1}^{n} XiDpi$

(4) Volume mean diameter:-

$D_P=1/\int_0^1 d\emptyset/Dp$ where, Φ=cumulative mass fraction

The volume mean diameter is calculated by taking the inverse of area under the curve of a graph plotted between Φ vs. $\frac{1}{\text{Dpi}}$.

PROCEDURE: ---

Step -1

Take 500 gm of sample by coning and quartering method.

Step-2

Arrange a standard set of sieves serially in a stack the largest mess at the bottom.

Step-3

Take the sample on the top screen. Shake the whole sieve stack for 10 min in a sieve shaker.

Step -4

Take the particles left in each screen carefully and separately weight them to find the mess in each screen.

Step -5

Calculate the mass fraction in each case.

Step-6

Substitute the formula and find out all the average particle diameters.

TABULATION: ---

Sl.No	Mess no	Mess opening	Avg. particle size(Dpi^2)	Mass collected (gm)	Mass fraction (X_i)	$1/D_{pi}$	$\frac{Xi}{Dpi}$	X_i/D_{pi}^2	X_i/D_{pi}^3	X_i*D_{pi}

CONCLUSION:-

EXPERIMENT: 2 PLATE & FRAME FILTER PRESS

OBJECTIVE:-

To study the operation of plate and frame filter press.

AIM OF THE EXPERIMENT:--

To determine Specific cake resistance (α) and medium resistance (R).

CHEMICAL REQUIRED:- $CaCO_3$

INTRODUCTION:--

The separation of solids from a suspension in a liquid by means of porous medium or screen which retains the solids and allow the liquid to pass is termed filtration. In general the pores of the medium will be larger than the particles which are to be removed, and the filter will work efficiently only after an initial deposit has been trapped in the medium. Filtration is essentially a mechanical operation and is less demanding in energy than evaporation or drying. The most suitable filter for any given operation is the one which will fulfil the requirements at minimum overall cost. The most important factors in filter selection are the specific resistance of filter cake, the quantity to be filtered and the solid concentration.

THEORY:--

Filtration involves the separation of solids from liquids by passing a suspension through a permeable medium, which retains the particles. The set up consists of 7 plates and 6 frames. Frames are covered with filter cloth. Feed is fed by gear pump at the top and filtrate collected at the bottom from each plate by operating the cock. After removing cake, washing and cleaning can be done by water provided by overhead tank. Inlet and outlet pressures are measured by pressure gauges. Rate of filtrate removals is measured by calibrated tank provided.

EXPERIMENTAL PROCEDURE: ---Prepare slurry of $CaCO_3$ in water (5%).Filter the prepared solution and fed it in the feed tank.Switch on the agitator of the feed tank.Fix the plate and frames on the press.Connect the outlet of the filter press to the filtrate tank.Start the pump and allow feed to enter the press.Record the Inlet slurry pressure, P_i and Outlet slurry pressure, P_o.Collect the filtrate in the receiver. Record the amount of filtrate collected v in time t.Run the filtration till there is appreciable fall in rate of filtrate collection.

FOR INCOMPRESSIBLE CAKE: ---

The basic filtration equation is:

$$\frac{DT}{DV} = \frac{\alpha * \mu * C}{2A^2 * P} * V + \frac{\mu * R}{A * P}$$

Where,

$A_1 = \alpha * \mu * C$ and $B_1 = \mu * R$

$$\frac{DT}{DV} = A_1 \frac{V}{2A^2 P} + B_1 \frac{1}{A P}$$

Under constant pressure filtration condition, integration of the above equation yields:

$= 2 * A^2$

EXPERIMENT: 3 FROTH FLOTATION CELL

AIM:

To conduct the froth flotation experiment on a mixture of coal and sand and to find the separation efficiency.

THEORY

Froth flotation involves separation of a mixture on the basis of difference in the surface properties when the mixture is suspended in the aerated liquid, the gas bubbles (foam) adhere to one of the constituents- the one which is more difficult to wet by the liquid, where as the other constituents being wet stay with the liquid. Froth flotation is employed widely in metallurgical industries (sulphide ores) and coal. Mixed liberated particles can be separated from each other by flotation if there are sufficient differences in their wet ability. The flotation process operates by preparing a water suspension of a mixture of relatively fine-sized particles (smaller than 150 micrometers) and by contacting the suspension with a swarm of air bubbles in a suitably designed process vessel. Particles that are readily wetted by water (hydrophilic) tend to remain in suspension, and those particles not wetted by water (hydrophobic) tend to be attached to air bubbles, levitate (float) to the top of the process vessel, and collect in a froth layer.

Thus, Differences in the surface chemical properties of the solids are the basis for separation by flotation. Surfaces that do not have strong surface chemical bonds that were broken tend to be nonpolar and are not readily wetted. Substances such as graphite and talc are examples that can be broken along weakly bonded layer planes without rupturing strong chemical bonds. These solids are naturally floatable. Also, polymeric particles possess nonpolar surfaces and are naturally hydrophobic. By contrast, most naturally occurring materials are polar and exhibit high free energy at the polar surface. The polar surfaces react strongly with water and render those particles naturally hydrophilic. The relative wet ability of the solids in a mixture can be enhanced by the addition of various surface chemical agents that are adsorbed selectively on the particle surface.

MINERAL APPLICATIONS

The flotation process is most widely used in the mineral process industry to concentrate mineral values in the ores. Most of the world's copper, lead, zinc, molybdenum, and nickel are produced from ores that are concentrated first by flotation. In addition, flotation is commonly used for the recovery of fine coal and for the concentration of a wide range of mineral commodities including fluorspar, barite, glass sand, iron oxide, pyrite, manganese ore, clay, feldspar, mica, sponumene, bastnaesite, calcite, garnet, khanate, and talc.

FLOTATION REAGENTS

Three types of chemical reagents are used during the Froth Flotation Process: Collectors, Frothers, and Modifiers.

COLLECTORS

These are surface-active agents that are added to the flotation pulp, where they adsorb selectively on the surface of the particles and render them hydrophobic. A convenient classification of the commonly used collectors is shown in Figure. The nonionizing collectors (fuel oils and kerosene) are practically insoluble in water and cause the particles to become hydrophobic by covering them with a thin film. The ionizing collectors dissociate into ions in water and are made up of complex heteropolar molecules in that the molecule contains both a nonpolar hydrocarbon group with pronounced hydrophobic properties and a polar group with hydrophilic properties. The ionizing collectors adsorb either physically or chemically on the particle surface and can further be classified into anionic or cationic collectors depending on the nature of the nonpolar hydrocarbon group. Common examples of the ionizing collectors include fatty acids, long-chain sulphates, sulfonates and amines, xanthates, and dithiophosphates. Dosage requirements for collectors depend on the mechanisms by which they interact with the particle surface, but just enough is needed to form a monomolecular layer. As a rule, high dosages are required for nonionizing collectors and physisorbing ionizing collectors (in the order of 0.1 to 1 g of reagent per kg of solids) and low dosages for chemisorbing ionizing collectors (0.01 to 0.1 g of reagent per kg of solid). Addition of excess quantities of a collector is not desirable because it results in reducing the selectivity and increasing the cost.

FROTHERS

These are also surface-active agents added to the flotation pulp primarily to stabilize the air bubbles for effective particle bubble attachment, carryover of particle-laden bubbles to the froth, and removal of the froth. The frother action is similar to the ionizing collectors except that they concentrate primarily at the air-liquid interface. Commonly used frothers are pine oil, cresylic acid, polypropylene glycol, short-chain alcohols, and 5- to 8-carbon aliphatic alcohols.

MODIFIERS

Flotation modifiers include several classes of chemicals.

1. ***Activators***. These are used to make a mineral surface amenable to collector coating. Copper ion is used, for example, to activate sphalerite (ZnS), rendering the sphalerite surface capable of absorbing a xanthenes or dithiophosphate collector. Sodium sulfide issued to coat oxidized copper and lead minerals so that they can be floated by a sulfide mineral collector.
2. ***PH regulators***. Regulators such as lime, caustic soda, soda ash, and sulfuric acid are used to control or adjust pH, a very critical factor in many flotation separations.
3. ***Depressants***. Depressants assist in selectivity (sharpness of separation) or stop unwanted minerals from floating. Typical are sodium or calcium cyanide to depress pyrite (Fe_2S_2) while floating galena (PbS), sphalerite (ZnS), or copper sulfides; zinc sulfate to depress ZnS while floating PbS; sodium ferrocyanide to depress copper sulfides while floating molybdenite (MoS_2); lime to depress pyrite; sodium silicate to depress quartz; quebracho to depress calcite ($CaCO_3$) during fluorite (CaF_2) flotation; and lignin sulfonates and dextrin's to depress graphite and talc during sulfide flotation.

4. ***Dispersants and flocculants***. These are important for the control of slimes that sometimes interfere with the selectivity and increase reagent consumption. For example, soda ash, lime sodium silicate, and lignin sulfonates are used as dispersants, and starch and polyacrylamide are used as flocculants

EQUIPMENT

Various types of flotation machine designs can be classified into different categories based on the methods used for the generation and introduction of air bubbles into the equipment .Each of the techniques of air bubble generation and particle-bubble contact along with the special features associated with different kinds of equipment has its own advantages and limitations. These must be considered carefully in selecting the equipment for a specific application.

Individual manufacturers can provide basic help in selecting the equipment.

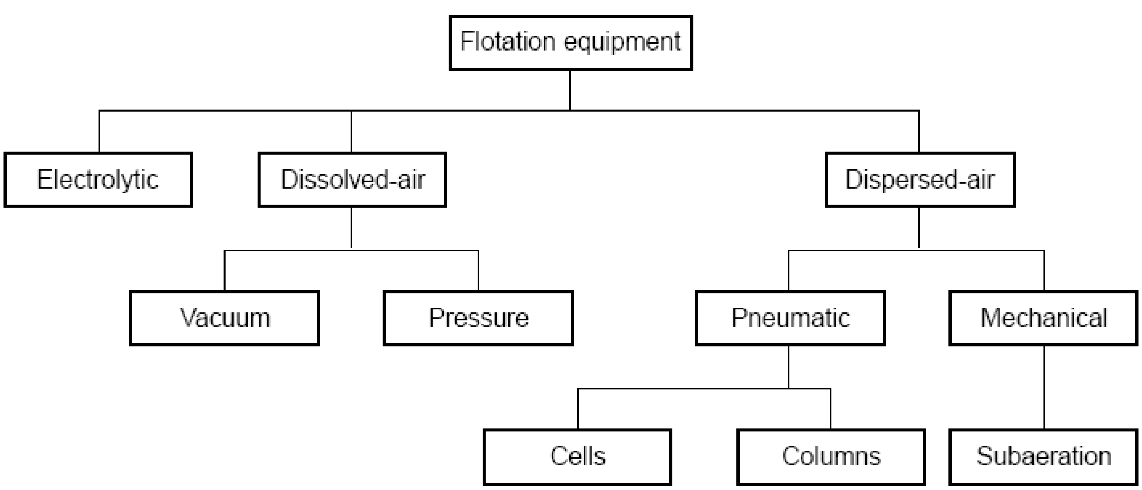


Figure 1: Classification of Flotation Equipment’s

MECHANICAL CELLS

It is characterized by a cubic or cylindrical shape, equipped with an impeller surrounded by baffles with provisions for introduction of the feed slurry and removal of froth overflow and tailings underflow. The machines receive the supply of air through a concentric pipe surrounding the impeller shaft, either by self-aeration due to the pressure drop created by the rotating impeller or by air injection by means of an external blower. In a typical installation, a number of flotation cells are connected in series such that each cell outputs froth into a launder and the underflow from one cell goes to the next one. The cell design may be such that the flow of slurry from one cell to another can either be "restricted" by weirs or unrestricted.

The mechanical cells that are most widely used today in sulfide, coal, and non-metallic flotation operations in the western hemisphere are made by Fagergren (by WEMCO Division of Envirotech Corporation), D-R Denver (by Denver Equipment Corporation of Salas International), Aviator (supplied by Gallagher Ash Company), and Outokumpy (by Outokumpu Oy). These machines provide mechanical agitation and aeration by means of a rotation impeller on an upright shaft. In addition, the Agitating and Denver cells also utilize air from a blower to help aerate the pulp.

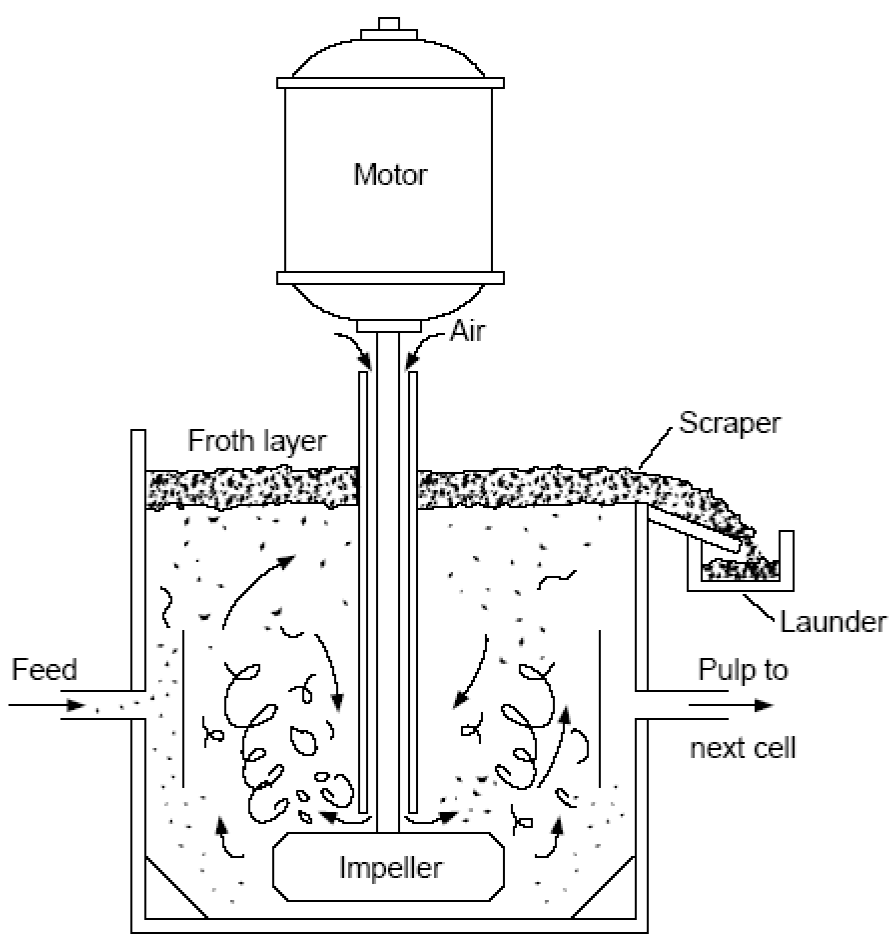


FIGURE: Schematic of Mechanical Flotation Cell.

EXPERIMENTAL SETUP

The unit consists of a froth flotation cell made SS 304 steel with an impeller with a diffusion ring. An air inlet is provided at the top. The same is connected to a ¼ bhp motor through reduction pulley system. A weir attached to the cell helps in collection of the froth.

EXPERIMENTAL PROCEDURE

1. Accurately weigh 45 gm of finely ground coal and 5 Gms of sand. Mix them thoroughly.

2. Add 10 Litters of water to the cell and start the impeller.

3. Add 2/3 drops of pine oil (floating agent and collector) and a pinch of detergent to create a stable froth.

4. Add the coal mixture to the tank. Skim off the froth and col1ect the same through the weir.

5. Dry the froth collected and find the separation efficiency.

OBSERVATION AND CALCULATION

Wt. Of initial coal taken : W_1 = ………..

Wt of sand taken : W_2 = …………

Wt of dried froth : W_3 = …………

Separation efficiency : $(W_3/W_1) \times 100$ = ………. %

EXPERIMENT: 4 SETTLING OF SLURRY

AIM OF THE EXPERIMENT: ---

To find the rate of settling of slurry in water.

APPARATUS AND MATERIALS REQUIRED:--

Balance with weight box,2 cylindrical column of internal diameter 25.4mm and 50.6mm,bucket chalk powder and $kmno_4$ crystals.

THEORY:--

When solid particles are suspended in a fluid, they tend to settle down due to force of gravity, provided the density of a fluid is less than that of solid. The rate of settling depends on the particle properties like density, size and shape of particles and the fluid properties like density and viscosity.

PROCEDURE:-

Step-1: Make slurry of 10% chalk powder in water and add few crystals of $kmno_4$.

Step-2: Take the slurry in two columns. Observe the height of slurry water interface at different intervals. Take first 30 reading in 30 sec interval, next 20 readings at 1 minute interval and last readings at 2 minute interval.

Step--3: Plot graph between height of slurry and time of settling for each column.

Step—4: Calculate the rate of settling dH/dt at five different intervals by drawing tangents at five points on curve.

Step—5: Plot a grape of dH/dt vs. time.

OBSERVATION TABLE:-

SL NO	Time in min	Height of slurry in cm	
		Column-1	Column-2

CONCLUSION: ---

EXPERIMENT: 5 WILFLEY TABLE

AIM OF THE EXPERIMENT: ---

To determine the separation efficiency of the Wilfley Table.

APPARATUS REQUIRED:--Wilfley Table

THEORY:-

Wilfley table is used to separate the materials having difference in density .It consists of flat inclined slightly with differential motion, i.e. it moves slowly and comes back suddenly. The feed slurry is admitted at the feed point where water is allowed to flow at rate, the lighter particles moves across the riffles and can be collected. The heavier particles tend to move along the ridge and collected separately.The separation efficiency of the wilfley table can be calculated as follows:--

%Recovery = (W_F/W_{FO}) *100

Where,

W_F= Weight of the coal particle collected in gm.

W_{FO}=Weight of the coal particles in feed in gm.

PROCEDURE:-

1. Locate the wilfley table (MO/CH/WT/06) in Mechanical operation lab. Then follow with these with detail ID and information as suggested.

2. Weight accurately 25 gm. Of (-35, +60 mesh) coal and add 25 gm of sand to it. Mix it thoroughly and prepare slurry of this feed.

3. Open the water inlet valve at constant flow rate (start with low flow rate).

4. Feed the slurry at low flow rate.

5. Collect ball the fine particles received in the launder.

6. Run the experiment till you get the clear water.7.Stop the water flow and collect the course particles.

8. Dry the course particle and record its weight in the observation table .Then % of recovery can be easily found out from the formula.

9. The experiments may be repeated for different flow rates.

OBSERVATION TABLE: ---

SL NO	WEIGHT OF THE COAL PARTICLE IN FEED IN	WEIGHT OF THE COAL PARTICLE COLLECTED IN

	$GM(W_{fo})$	$GM(W_f)$

Separation efficiency can be found by dividing weight of the coal particle in feed to weight of the coal particle collected multiplied by 100.

PRECAUTIONS:

1) Always keep apparatus free from dust.

2) Never run the pump at voltage less than 180 volt and above 230 volt.

3) Frequently grease or oil in the rotating part once in three months.

4) Always use clean water.

EXPERIMENT: 6 VIBRATING SCREEN

AIM OF THE EXPERIMENT:--

To study and operate of the vibrating screen and to find the effectiveness of the screen.

APPARATUS REQUIRED:--

Vibrating screen,4-60 BSS Mesh sieves, Sieve shaker,Brush,Sieve opener,Balance,Weight box.

THEORY:--

Let a screening operation is being carried out for two materials A & B. F=mass flow rates of feeds, B= mass flow rates of underflow, A=mass flow rates of overflow. The mass fraction of B in feed, overflow and underflow are $(1-X_F)$,$(1-X_D)$,and $(1-X_B)$.

From total mass balance: F=B+D ------------------- (1)

From component material balance: $FXF=BX_B+DX_D$ ---------------- (2)

$D/F= (X_F-X_B)/ (X_D-X_B)$ and $B/F= (X_D-X_F)/ (X_D-X_B)$

Thus the effectiveness of the screen

(i) W.r.t overflows of material (A)

$E_A=DX_D/FX_F=[(X_F-X_B)/(X_D-X_B)]*X_D/X_F$------------ (3)

(ii) W.r.t underflow of material (B)

$E_B=BX_B/FX_F= [(X_D-X_F)/(X_D-X_B)]*X_B/X_F$--------------(4)

The overall effectiveness of the screen

$E=E_A*E_B= [(X_F-X_B)(X_D-X_F)X_D(1-X_B)] / [(X_D-X_B)^2(1-X_F)X_F]$ ---------------(5)

PROCEDURE: ---

STEP-1: Take 1 kg of sample and sieve it in the sieve-shaker. Write down the mass collected in each screen through 4 to 60 meshes and the cumulative mass fraction.

STEP-2: Mix all the particles and feed to the vibrating screen.

STEP-3: Start the motor and collect the overflow and underflow through different bags after 15 minutes of operation.

STEP-4: Sieve the unit oversize and undersize and calculate the cumulative mass fraction for all. Plot Φ vs. D_P (for feed, oversize and undersize and oversize) on the same graph.

STEP-5: Calculate the X_F, XB and X_D at the cut point diameter from the graph. Then calculate the effectiveness of the screen ($E=E_A*E_B$) at the lower screen size -1.5mm= (0.0625 inch).

CALCULATION:-

CONCLUSION: ---

EXPERIMENT: 7 CYCLONE SEPARATOR

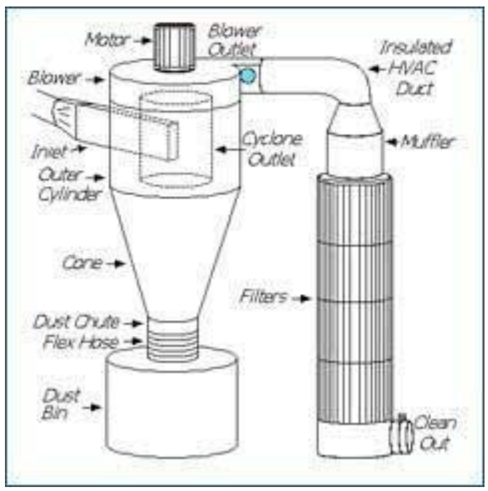
AIM OF THE EXPERIMENT:-

a) To study the performance of a given cyclone.(eff.vs dp).

b) To study the effect of inlet gas velocity on overall efficiency.

c) To study the effect of solid concentration of Δp.

FIGUARE OF CYCLONE SEPARATOR:--



THEORY **:-**

Cyclone is the most widely used centrifugal separation equipment for separating dust or mist from gases.Most centrifugal separators for removing particles from gas stream contain no moving parts. They are multiplied by the cyclone separators. It consists of a vertical cylinder with a conical bottom, a tangential inlet near the top, and an outlet for dust at the bottoms of the cone.The inlet is usually rectangular. The outlet pipe is extended into the cylinder to prevent short circuiting of air from inlet to outlet.The incoming dust laden air travels in a spiral path around and down the cylindrical body of a cyclone. The centrifugal force developed in the vertex tends to settle down into the cone and are collected. The cyclone is basically a settling device in which a strong centrifugal force, acting radially, is used in place of a relatively weak gravitational force acting vertically. The centrifugal force F_c at radius r is equal to mu^2_{tan}/r_c, where m is the mass of the particle and u_{tan} is its tangential velocity. The ratio of centrifugal force to the force of gravity is then

$\frac{Fc}{Fg}$ = mu^{2}tan/r/mg= u$^2_{tan/rg}$ =separation factor

The dust particles entering a cyclone are accelerating radically but the force on the particle is not constant because of the change in r and because the tangential velocity in the vertex varies with r and with distance bellows the inlet. Calculation of particle trajectory is difficult and efficiency of cyclone is generally predicted from empirical co-relations.The lower efficiency of larger cyclone is mainly a result of the decrease of the centrifugal force. For a given air flow rate and inlet velocity, however, moderate increase in cyclone diameter and length improve the collection efficiency, because the increase in surface areas of sets the decreased centrifugal force. Higher and lower efficiency would be expected for larger and smaller units at the same flow rate and inlet velocity.The decrease in efficiency with decreasing particle size is actually more gradual than the predicted by simple theories. For smaller particles the radial velocity and the collection efficiency should be a function of D_p^2, but agglomeration of the fines may occur to raise the efficiency of these particles. Because of the particle size effect, the uncollected leaving the gas has much smaller average size than the entering dust, which may be important in setting emission limits. Also, the overall efficiency is a function of particle size distribution of the feed and cannot be predicted just from the average size.The collection efficiency of cyclone increasers with particle density and decreases as the gas temp is increased because of the increase in the gas velocity. The efficiency is quite dependent on the flow rate because of the u^2_{tan} term in above equation. The cyclone is one of the few separation devices that work better ion full load than at the partial load. Sometimes two identical cyclones are used in series to get more complete solid removal, but the efficiency of the second unit is less than that of the first, because the feed to second unit has a much lower average particle size.

The pressure drop in a cyclone is proportional to the gas density and the square of the inlet velocity; it does not depend upon the density of the solid particles. The pressure drop actually decreases with increasing particle concentration.

PROCEDURE:--

1. Prepare feed stock of cement dust or fly ash with constant average particle size.

Size: 50µm, 40µm, 20µm, 10µm, 8µm, 6µm, 4µm, and 2µ.

Prepare about 200 Gms each.

2. Run the cyclone with fixed air at fixed μ_t(between 50-90 ft/sec.)by adjusting the blower valve.

3. Measure the Δp across the cyclone inlet and outlet in terms of inches of water.i.e.Δp_c=0.

4. Now feed the dust particles of one particle size (say 50µm) 200gms (W_{io}) at a constant rate.

5. Collect the solid at the solid outlet of the cyclone.

6. Measure the weight of collected (W_{ic}) and the pressure drop Δp_i.

7. Calculate the collection efficiency $\eta_i = (W_{ic}/W_{io}) * 100$

8. Repeat steps 4 to 7 for all sizes of solid particles.

Change the air inlet velocity and repeat step 3 to 7 for a particular solid say 6 or 4µm.

RECORDS AND CALCULATION:--

Note all the dimensions of the given Cyclone (D_C=130mm)

At ambient condition note the density and velocity of air.

Note the average particle size and the particle.

Particle size µm	W_{io} gms	W_{ic} gms	$\eta_i=(W_{ic}/Wio)*100$	W_i	Δp in N/m	Conc. Of particles gms/cm^3
50						
40						
20						
10						
08						
06						

$\eta_T = \sum W_i \eta_i$

CO-RRELATIONS:--

Inlet gas velocity $\mu_t = Q/H_c * B_c$, ft/sec

μ_t lies between 50-90 ft/sec.

Number of turns $N_a = (L_c + Z_c/2)/H_c$

Natural length, $l = 2.3 D_c \, (D_c^2/H_c B_c)^{1/3}$

Pressure loss (mm of water) from manometer

$D_p=0.003 \rho_r \mu_r^2 N_H (D_p<10" \text{ water})$

ρ_r=lb/ft^3 (fluid density)

μ_f=ft/sec

$N_H=16(H_cB_c/D_c^2)$ =No. of Intel velocity heads. (usually lies between 5-8)

$\Delta P_{at\ c}=\Delta P_{at\ c=0}$ $[1-0.013\ c^{1/2}]$

C= grains/ft^3

Collection Efficiency [η_i] %= [W_i (collected)/ W_i (feed)]*100

Of ith particle overall all efficiency $\eta_{\%}=\sum w_i\eta_i$

SPECIFICATION:

Motor: 1 HP 2800 RPM, 1 phase, make Crompton greaves

Control panel: Mains on-off switch.

Pipe dia d_p: 50mm=0.05m

Cross section area of pipe: 1.963*10^{-3}m^2

Dia of orifice d_o: 27mm=0.027m

Cross section area of orifice: 5.7255*10^{-4}m^2

Cyclone diameter D_c: 100 mm=0.1 m

Manometer: 2 Nos (one each for orifice and cyclone)

To calculate Air flow:

$Q= 0.6\P/4 d_p^2 d_o^2 \sqrt{2g\Delta H} /\sqrt{(d_p^2-d_o^2)}$ m^3/s

Velocity of air =V/A, where A is cross section area of pipe 1.963*10^{-3}m^2

CONCLUSION:-

EXPERIMENT: 8 MAGNETIC SEPARATOR

AIM OF THE EXPERIMENT:--

To study about the magnetic separator, and the effect of magnetic field on efficiency of the process.

REQURIED: **MATERIALS/APPARATU**s:-

1. Ore

2. Different sieve screens of ASTM size.

3. Magnetic separator

4. Weight balance

THEORY: ---

It is a technique which is used to separate the mineral particles from gangue particles on the basis of its magnetic properties. Actually there are three types of magnetic materials,

1. Ferro magnetic-strong magnetic field

2. Para magnetic-weak magnetic field

3. Dia magnetic-no magnetic field

Type:--

On the basis of magnetic intensity ---

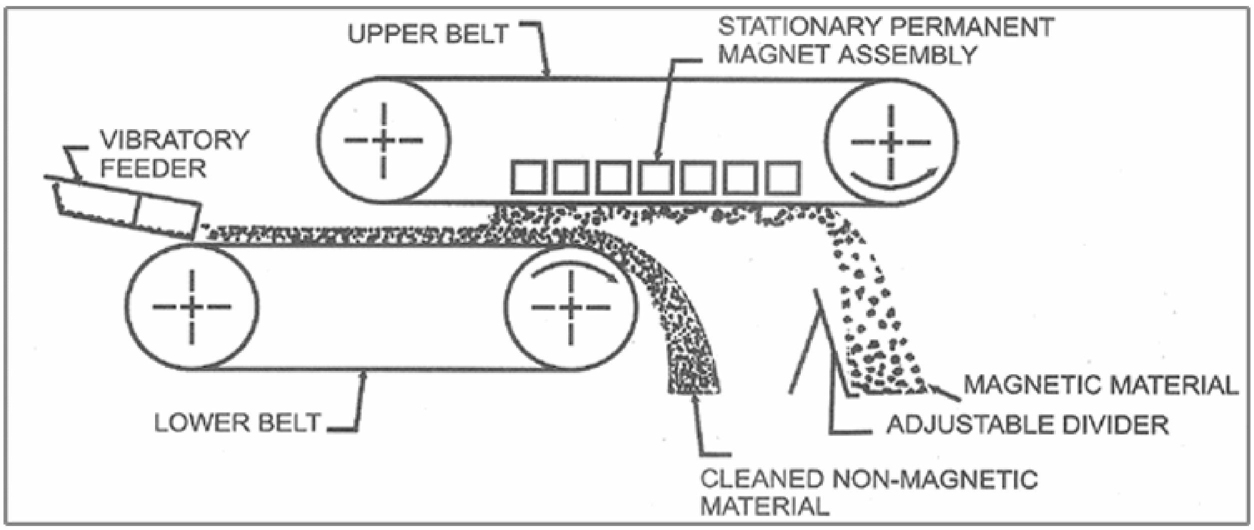
- Low density magnetic separator
- High density magnetic separator

Again, on the basis of medium

- Dry magnetic separator
- Wet magnetic separator

Basic principles of magnetic separation process: ---

It works on the principal of mutual attraction of unlike charges, and mutual repulsion of like charges (Coulomb's law).On the basis of electrostatic charge, a body is said to be positively charged if it is deficient in electrons and is said to be negatively charged if it has excess of elections.



(SCHEMATIC DIAGRAM OF MAGNETIC SEPARATION)

Fundamental concept of magnetic separation:

1. Low intensity magnetic separators are used for the separation of the highly magnetic materials.

2. High intensity magnetic separators are used for the separation of weak magnetic materials.

PROCEDURE:--

- Weight 5-6 kg of feeding ore of particular size by a digital weight balance.
- Switch on the impeller motor, and also the motor of the rake.
- Allow addition of the feed sample through the feed at a steady rate.
- Collect the ore under the flow of magnetic field thrown away from the separator.
- Also collect the free flow placed nearer to the separator, and switch off the machine.
- Calculate the efficiency.

CALCULATION:-

The efficiency of the magnetic separation calculated by using the formula given below:--

Efficiency (in %) = [weight of the collected magnetic materials/weight of the total feed materials]* 100

CONCLUSION: ---

Hence, we studied about the magnetic separator, and also calculated the efficiency of magnetic separation.

EXPERIMENT: 9 ANGLE OF REPOSE

AIM

1) To measure the angle of repose of the sand.

2) To study the factor that can influence the angle of repose of the sand.

3) To study the effect of glidant on the angle of repose.

INTRODUCTION

Basically, the movement/slideness of particle are due to the angle of inclination that is more than frictional force between particles. In Addition, the object in motion will stop sliding when angle of inclination is decrease to its limit, this due to the adhesion and cohesion When bulk granular material is poured on a horizontal surface of conical pile, it will form the internal angle between the surface of the pile and the horizontal surface is known as angle of repose. Angle of repose or the critical angle of repose is the steepest angle of descent or dip of the slope relative to the horizontal plane when material on the slope face is on the verge of sliding. This angle is in the range 0°–90°.In this experiment, we were measuring the angle of repose of the sand with 355 micron, 500 micron, 850 micron and various sizes without and with the addition of glidants (0.5%,1%,2%,3% of magnesium stearate). The experiment is done with a view to assessing the angle of repose of a substance and the factors that can influence the angle of repose.

APPARATUS AND MATERIAL

100g of 355, 500, 850 micron and various size of sand Glidants and ruler

PROCEDURE

1. The 355 micron sand (without addition of the glidant yet) was poured in a level surface allowing it to build from the top.
2. The height of the pile from the peak to the ground eas measured by using the ruler
3. The horizontal distance from the middle of the pile to the edge was measured by using the ruler.
4. The equation tan-1 (height/width) had been used to find the angle of repose.
5. The procedure 1-5 was repeated with the addition of glidant.
6. The procedure 1-6 was repeated by using 500 micron, 850 micron and various size of sand

CALCULATION

$$\text{Angle of Repose}, \theta\ (^\circ) = \tan^{-1} \frac{height}{width}$$

Size of sand (micron)	Angle of repose (without 3% of magnesium stearate)	Angle of repose (with 3% of magnesium stearate)
355	36.3°	50.7°
500	33.1°	44.4°
850	31.5°	41.9°
Various sizes	39.2°	45.6°

Conclusion:-