GOVERNMENT POLYTECHNIC , JAJPUR

LECTURE NOTE ON

" MINERAL DESSING"

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COURSE CONTENTS (Based on specific objectives)

1. Introduction

o Describe the objective & scope of application of mineral dressing in surface & u/g mines.

2. Unit operations

 Explain the principle of Blake & dodge jaw crushers, gyratory & cone crushers, roll crusher.

3. Grinding

 Explain the principle of ball mill operation, open circuit grinding, close circuit grinding, dry & wet grinding.

4. Explain the procedure for size analysis & use of standard screen as also screening techniques employed.

5. Industrial screening

o Explain the principle of industrial screening, type of screening (without calculation)

o Explain the operation of classifier & their application.

6. Gravity concentration

o Explain the general principles of wilfly table & its operation.

o Develop elementary idea regarding the operation jigs.

7. Heavy media separation

 Explain the fundamental principle of heavy media separation – Chance process.

8. Floatation

o Comprehend elementary principle of froth floatation, practical utility of frother, collection, modifiers & depressants.

o Describe & illustrate floatation cell.

9. Magnetic & Electrostatic Separators

o Explain the principle of operation of magnetic & electrostatic separators.

o Describe the application of separators in mineral dressing.

MINERAL DRESSING

Introduction:-

A metal extraction plant's working is conveniently represented by means of a flow sheet. Flow sheet is a combination of processes which are followed in the given plant to extract the metal(s) most economically. While analysing the flow sheet we come across certain *unit processes and operations*. The *unit processes* are usually characterized by certain chemical reactions such as roasting, leaching etc while *unit operations* are usually physical processes carried out discreetly on the ore. These physical processes are usually represented by crushing, grinding and similar such processes. Unfortunately there is no rigid line of distinction between them. However, from metallurgical engineering point of view any physical operation carried out on the ore to enhance its quality and make it more suitable for subsequent operations will be termed as *Ore Dressing or Mineral Beneficiation.*

So mineral dressing or ore dressing is commonly regarded as processing of raw ores to yield marketable products by such physical means those do not destroy the physical and chemical identity of the ore.

Economic Justification of Mineral Dressing:

1. ***To purify and upgrade the ore:***

   It is apparent that many ores & minerals do require some prior preparation to enhance their chemical purity and physical properties before their use in smelters.

2. ***Making smelting practice easier:***

   Hydrometallurgical extraction of metals is very slow, complex and expensive in most of the cases compared to pyrometallurgical process of extraction. In the initial stages the ores can be upgraded by employing inexpensive and simple dressing methods to make them suitable for pyrometallurgical extraction. Such an activity reduces the complexity of the smelting practice resulting in economic justification.

3. ***Savings on Freight:***

   During ore dressing the ores get beneficiated and gangue materials get separated. As the waste products are not to be transported from the mines areas, huge money is saved on freight by transporting upgraded ores.

4. ***Reduced losses of metal at the smelter:***

   As the gangue portion of the ore is separated by means of simple beneficiation methods the slag volume during the smelting process decreases. This ultimately results in a lesser loss of metal into the slag.

1. To prepare ore from physical standpoint.

1.1. Historical Development:

Like other sciences, the art of ore beneficiation has started form historic time and has got modified, refined with the progress of scientific knowledge. It was Agricola who started recording the metallurgical facts relating to ores in the form of a book. He is considered to be the father of mineral beneficiation or ore dressing. The dressing methods started developing in the following manner chronologically:

1. **Hand Sorting:**

Undoubtedly the oldest method of are beneficiation. This is a method of choosing valuable ore lumps from worthless lumps basing on the appearance, fracture cleavage and gross weight. This is still in use where cheap labour is available.

2. **Washing:**

Washing in all probability is the next process that evolved. Water exerts a cleaning action and removes slimes. It is still in use with modification for washing and cleaning of coal and iron ores.

3. **Crushing:**

It was discovered that valuable particles generally occur in association with worthless particles in large lumps quite early. So to separate them it needs breaking of the large lumps. Thus crushing is considered to the next step in the history of mineral dressing. It was carried out by using sledge hammers or brute force of the human operators.

1. **The ideas of washing stretched further with the particular use of specific Tabling and Gravity Concentration:**

gravity of the ore particles for concentrating them in terms of their specific gravity.

2. **Jigging:**

It was developed by the Herz in the Germany. Along with jigs, Vanners and shaking tables were also developed simultaneously.

3. **Grinding:**

Modern grinding machines were developed much late along with stationary screens to produce fine ores required for gravity concentration and froth flotation.

4. **Classification:**

Of late to separate fine size particles classifiers came into picture.

5. **Development in Recent Years:**

In recent year magnetic separators, electrostatic separators, flotation and agglomeration techniques have been developed to upgrade the ores.

1.2. GENERAL OPERATIONS INVOLVED IN ORE DRESSING:

1. **Comminution:**

Comminution or size reduction can be accomplished dry or wet.

2. **Sizing:**

This is the separation of product material into various fractions depending on their size parameter.

3. **Concentrating:**

Concentration of valuable portion of the ore is obtained by the various means which generally involve physical characteristics of the ore particles. Sizing, jigging, tabling, classification, magnetic & electrostatic separation are few such examples. We may exploit an entirely different set of physio-chemical properties for concentrating the ore as it happens during froth flotation.

4. **De -Watering:**

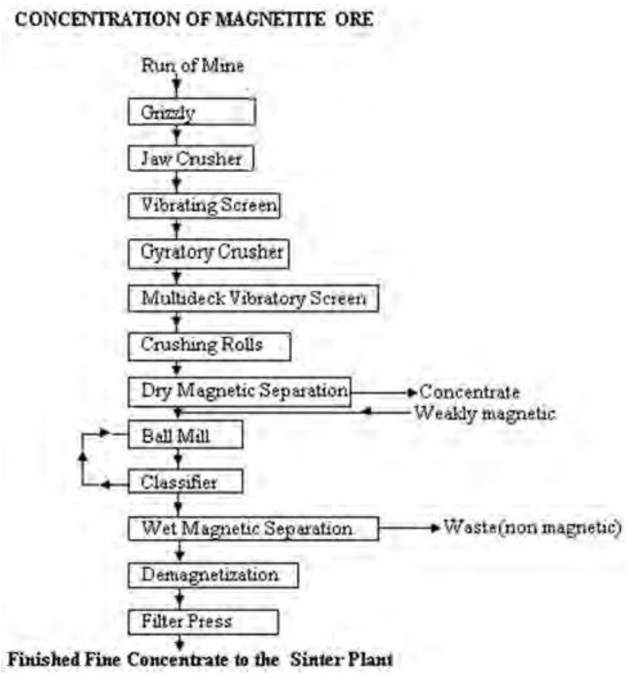
Where aqueous medium is involved, water is to be removed before smelting can take place.

This involves:

a) Removal of most of the water by the use of the thickener.
b) Then use of filter presses to prepare a damp cake of the concentrated ore.
c) Then drying the cake in a furnace.

General Flow Sheet Of a Mineral Beneficiation Plant:

Flow sheet is a typical representation of general processes used in a given plant to obtain the end product most conveniently. For the same end product using similar ores the operating conditions may vary from place to place however the general flow diagram remains the same. A generalized flow diagram for concentrating magnetite ore is illustrated in the figure1.1. shown below.



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CHAPTER 2

Unit operations

- **Explain the principle of Blake & dodge jaw crushers, gyratory & cone crushers, roll crusher.**

- **Introduction:**

The crude ore from the mines contain a number of solid phases in the form of an aggregate. The valuable portion of the ore is known as *mineral* while the worthless portion is known as *gangue*. During ore dressing, the crude ore is reduced in size to a point where each mineral grain becomes essentially free so as to make separation between them. Such a phenomenon of making the mineral grains free from gangue in an ore is termed as liberation. This is practically carried out by size reduction performed by crushers and grinding mills.

The ore lumps from the mines have the lump size of 10 - 100 cm while the individual minerals have grain sizes below 0.1 mm. Hence, the first step in any ore dressing plant is to aim at liberation by size reduction or comminution.

Comminution of any ore is carried out in several stages using different crushing equipments. So the objective crushing is to reduce the large lumps in to smaller sizes. Depending upon the feed and product particle size, the crushing operation can be classified as follows:

1. **Primary crushing:**

The feed material is usually the run of mine.

2. **Intermediate crushing or secondary crushing:**

The feed material is usually product of a jaw crusher.

3. **Fine crushing or coarse grinding:**

The feed material is usually comes from the secondary crushers.

4. **Fine Grinding:**

The objective of fine grinding is to produce ultrafine material less than one micron.

Size Parameter for Different Comminution Processes:

Suitable parameters of feed and product material for different crushing operations are shown in the table. 2.1.

Table.2.1.

s	ize	t size
se Crushing	150-4cms)	cm
mediate crushing.	5cm	1 cm
e grinding	0.2cm	75 microns
Grinding (Special type	m)	crons

Energy Requirement for Different Comminution Processes:

Different size reduction practices requires different amount of energy as shown in the table.2.2.

Table.2.2.

s	e Energy Consumption (kWh/to
se Crushing	0.2- 0.5
mediate crushing.	0.5 - 2
e grinding	1.0 -10
Grinding (Special type	2 - 25

2.1. Mechanism of Size Reduction:

Crushing is a mechanical operation in which a force of large magnitude is applied to a relatively brittle solid material in such a direction that its failure takes place. The theory of size reduction for solids is quite complex, but can be attributed to the action of following forces acting on the particle:

1. A huge compressive force exceeding the ultimate strength of the material may be responsible for size reduction as actually happens in case of jaw, gyratory and roll crushers.
2. A sufficiently high impact force may be responsible for size reduction. Impact force is largely utilized in hammer & ball mills. .
3. Attrition, rubbing action or frictional forces may be utilized for size reduction. Such action is largely responsible for crushing in attrition mill, tube and pebble mills.
4. Cutting force is utilized in knife edge mills to reduce the size of fibrous materials like mica, asbestos.

At least one or a combination of the above forces is always involved in size reduction in any crushing equipment.

2.2. Basic Requirements of Crushing Equipments:

An ideal crusher or grinder should have the following characteristics:

a. It should have a large capacity.
b. It should require a small (energy) input per unit weight of production.
c. It should yield a product of uniform size or in the required size range.

The performance of different crushing operation is studied individually with respect to the ideal operating conditions. A classification of the size reduction equipments can be made on the basis of feed and product size as follow:

2.3. Classification of the Size Reduction Equipments:

(In The Order Of Finer Size Product)

A. Primary Crushers:

1. Jaw crusher.
2. Gyratory crusher.

B. Intermediate crushers:

1. Crushing rolls.
2. Cone crusher.
3. Disc crusher.

C. Fine crushers or Coarse Grinders:

1. Ball Mill.

D. Fine Grinders:

1. Rod mill.
2. Pebble mill.
3. Tube mill.
4. Hammer mill with internal classifier.

2.4. Primary Crushers:

Crushers are slow speed machines for coarse size reduction of large quantities of solids. The major types of crushers are: Jaw, Gyratory, Roll & Toothed roll crushers. The first three types operate on compressive force and can crush very hard & brittle rocks. The toothed roll crusher tears the feed apart as well as crushes it. It works best on softer materials like coal, bone and soft slate. These are the crushers which operate on the run of the mine (*rom*). Primary crushers are of two types:

1. Jaw crusher.
2. Gyratory crusher

2.5. Classification of Jaw Crushers:

From capacity and working mechanism point of view jaw crushers are three types such as:

1. Blake crusher.
2. Dodge crusher.
3. Universal crusher.

The functional figure of different jaw crushers are as shown schematically in the

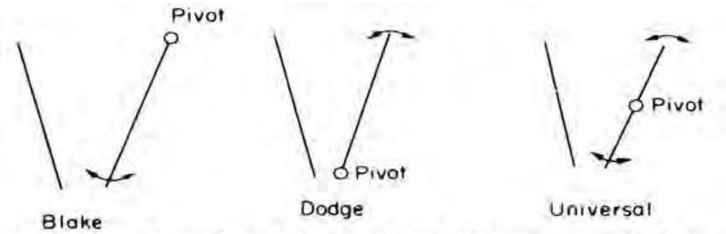


Fig-- Functional figures of different jaw crushers.

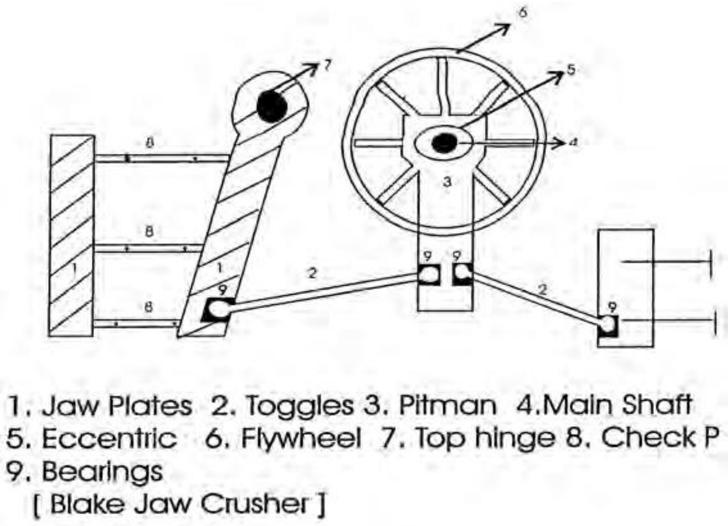
figure.2.1.

Blake Jaw Crusher:

It is a primary crusher used most widely. It has its moving jaw pivoted (hinged) at the top as in the figure2.1a.Though the working principles of Blake and Dodge crushers may be different from constructional point of view they are almost identical excepting two notable differences which will be discussed afterward. The Blake crusher may be classified as *single toggle* or *double toggle* type.

Constructional Features:

As the name suggests a jaw crusher has two jaws set to form a V-shape at the top through which feed is admitted into the jaw space. One of the jaws is fixed to the main frame of the crusher almost vertically while the other one is movable. The swinging jaw, driven by an eccentric, reciprocates in a horizontal plane and makes an angle of 20- 30 degrees with the stationary jaw. It applies a huge compressive force on the ore lumps caught between the jaws. The schematic figure of the Blake crusher is shown in the figure2.2.



On the jaws, replaceable crushing faces are fixed by nut & bolt arrangement. The crushing faces are made of *hadfield manganese steels*. When extensive wear is observed on any of the faces it is replaced with a new one. The crushing faces are rarely flat. They are usually wavy surfaces or may carry shallow grooves on them. The jaw running speed vary from 100-400 rpm.

The jaw widths vary from 2" to 48". The important features of jaw crusher are as follow:

As the moving jaw is pivoted at the top, the amplitude movement is largest at the bottom. The maximum distance the moving jaw travels is called *throw* of the crusher. The throw varies from 1-7cm. Jaw crusher is rated according to their receiving area, i.e., the *length* of the jaw plates and the *gape*. *Gape* is defined as the distance between the jaw plates at the feed opening end. For example an 1830X1220mm crusher has a *length (L)* of 1830 and a *gape* of 1220mm. For jaw crushers the *length or width* is usually greater than *gape*. The Blake crusher has a varying discharge opening. This distance between the jaws in the discharge side is termed as s*et(S)*.

These parameters are shown schematically in the figure 2.3.

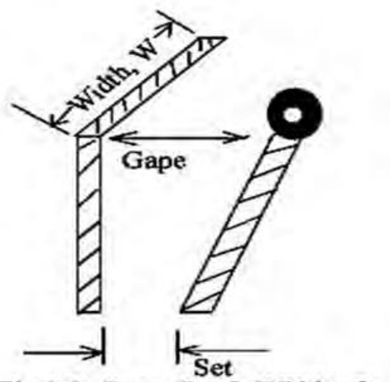


Fig.1.2. Gape ,Set & Width of a Jaw crusher.

Initially the large lump is caught at the top and is broken. The broken fragments drop to the narrower bottom space and is crushed again when the jaws close in next time. This action continues until the feed comes out at the bottom. The crushing force is least at the start of the cycle and highest at the end of the cycle. In this machine an eccentric drives the pitman. The circular motion of the main shaft is converted to up and down motion of the pitman via the eccentric and finally the *up and down* motion is converted to *reciprocating*(*to and fro*) motion with the help of two toggles. One of the toggles is fixed to the main frame and pitman while the other one is fixed to the moving jaw and pitman. From mechanical stand point, toggles are the weakest members of the jaw crusher. This is specifically made so to work as a safety device for the entire jaw crusher installation. There is every probability that an extremely hard material may enter into the jaw space along with the usual feed.

Such an occurrence starts developing a huge stress on the machine members. The stress would continue as long as the hard particle is not crushed. This may lead to situations where the jaw crusher would be severely damaged. Such a situation is avoided as the toggle(s) fails beyond a particular stress level being the weakest link of the jaw crusher members. Hence toggle(s) actuates the moving jaws and simultaneously work as a safety device for the jaw crusher. The failed toggles can be replaced with new ones without much problem. In crushers, the toggle plates are designed to take only a predetermined load.

Another important component of the Blake jaw crusher is the flywheel fitted onto the main shaft. The use of fly wheel is quite important from design point of view. As crushing takes place only during the forward stroke, intermittent and uneven load works on the machine members. To equalize this uneven load one or a number of flywheels are used on the main shaft. During the back-stroke, the material that has already been crushed is allowed to drop freely through the jaws. Forced feed lubrication is the rule in the jaw crushers. The machine is not operated very rapidly to restrict the production of fines.

Characteristics of Blake jaw crusher:

1. **Reduction Ratio:**

Blake crushers are the primary crushers. As the moving jaw is pivoted at the top it makes minimum and maximum swing at the top and bottom respectively. The maximum distance travelled by the moving jaw is defined as *throw* of the crusher. Blake jaw crushers have fixed *gape*. The width or length of the feed receiving opening is somewhat greater than the gape. The *set* determines the product particle size. Depending upon the *gape & set* the size reduction ratio (*R.R.*) generally available varies from 4-7. For a crusher the *R.R.* is defined as the ratio between average feed size to average product size.

Mathematically:

Reduction Ratio *(R.R)* = Avarage Feed Size Avarage Product Size

This is a very important parameter for determining the energy consumption in the crusher. Keeping all other variables fixed, higher the reduction ration (*R.R.*) higher is the energy consumed by the crusher.

2. **Capacity:**

The capacity of the jaw crusher mainly depends on the length and width of receiving opening and the width of discharge. As per Taggart, the empirical formula for capacity of jaw crusher is: $T = 0.6LS$ where,

T is the capacity expressed in tons per hour.

L is the *length or width* of the receiving opening in inches.

S is the *set* or width of discharge opening in inches.

The above empirical relation is quite accurate except for smallest and largest jaw crushers. The capacity of a jaw crusher may be as high as 725tons per hour for 2250x1680mm jaw size.

3. **Energy Consumption and Efficiency:**

Energy consumption in a jaw crusher varies considerably. Largely it depends on following factors:

a. Size of feed
b. Size of Product
c. Capacity of the machine
d. Properties of rock such as hardness, specific gravity, etc.

The energy utilization analysis in a crusher was first carried out by Owens. As per his conclusion the energy consumed in a jaw crusher is utilized in the following manner:

1. In producing elastic deformation of the particle before fracture occurs.
2. In producing plastic deformation which results in fracture of the particle.
3. In causing elastic distortion of the equipment.
4. Frictional losses between the particle & the machine.
5. Noise, heat & vibrational energy losses in the plant.

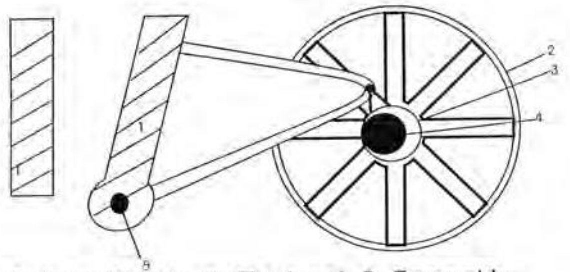
It has been estimated that only 10 - 20% of the total input energy is consumed for size reduction and the rest is lost in the machine in various ways. Out of the total energy consumed, largest amount gets converted to heat energy during crushing. Further this amount increases as the size reduction ration increases.

The jaw crushers are quite inefficient machines. The efficiency can be modified a little by analyzing the modes of energy utilization in a crusher. Proper lubrication and reduction in frictional losses can only increase the efficiency of the crusher. Further the physical properties of the ore which affect the efficiency of crushing are:

1. Specific gravity of the ore.
2. Hardness of the ore.
3. Moisture content in the ore.
4. Structural weakness planes of the ore.

Dodge Crusher:

Both Dodge and Blake crushers look similar to each other. In Dodge crusher the moving jaw is pivoted at the bottom in place of of the top as in case of Blake crusher. Hence the maximum swing of the moving jaw is obtained at the top. The *gape* is a variable while width of discharge opening (*set*) is fixed. Due to the fixed *set*, the product is more uniformly sized as compared to the product from the Blake. The crusher has got fewer mechanical parts as compared to Blake crusher. The moving jaw is activated by a lever It is activated by a lever-eccentric arrangement mounted onto the main shaft as compared



1.Jaws Plates 2. Flywheel 3. Eccentric
4. Drive Shaft 5. Bottom Hinge
[Dodge Crusher]

to the toggle-pitman combination in case of Blake crusher. Dodge crusher is shown schematically in the figure 2.4.

Table 2.3. Comparison between Blake & Dodge Crusher:

Jaw Crusher	**Jaw Crusher**
s got two toggles.	ne toggle in the form of a
s one pitman.	o pitman.
movable jaw is pivoted at the t variable product discharge op e feed receiving opening is fix	vable jaw is pivoted at the bo e discharge opening is fixed. fixed, while the feed receivi ing varies. This results st uniform sized product.
hoking takes place here as it h ble discharge. It operates on e of forced feed.	g is a very common problem quite small compared to ng opening.
crusher is mechanically more nced and has fewer breakdown ner it is built for much larger city.	nically the design of this crus ior. So it is built only to lowe city. This machine has more kdowns as compared to the
uct size distribution is large & es more fines.	size distribution is more n.
e is preferred at large industri s where elaborate screening f ailable along with other nution machines. out.	e is preferred where jaw crus used as the only comm pment.
machine is of higher cost for utput.	achine is cheaper for same
use of forced feed lubrication coarser product.	ke feeding is possible, it can much finer product.

Gyratory Crusher:

Classification of Gyratory Crusher:

Gyratory crushers have been developed recently in order to supply a machine with a larger capacity than jaw crushers.

The best known gyratory crushers are:

1. Suspended spindle gyratory crusher.
2. Parallel Pinch or Telsmith gyratory crushers.

Of late the suspended spindle gyratory has been obsolete and only the parallel pinch gyratory is used widely. Theoretically the parallel pinch is not a gyratory crusher since the crushing head rotates eccentrically instead of gyrating.

It consists of two substantially vertical truncated conical shells. The outer shell has its apex pointing down while the inner cone has its apex pointing up. The outer conical shell is fixed rigidly to the main frame while the inner cone or the crushing cone is mounted on a heavy central shaft also known as spindle.

The upper end-of the shaft is held in a flexible bearing while the lower end is driven by an eccentric so as to describe a circle. Because of this eccentric rotation, the inner cone thus rotates inside the outer cone alternately approaching and receding from all the points on the inner periphery of the outer shell. The solids caught in the V-shaped space between the crushing heads are broken repeatedly until they pass at the bottom. The crushing action takes place all over the cone surface. Fig.2.1. shows the functional elements of a suspended spindle gyratory crusher.

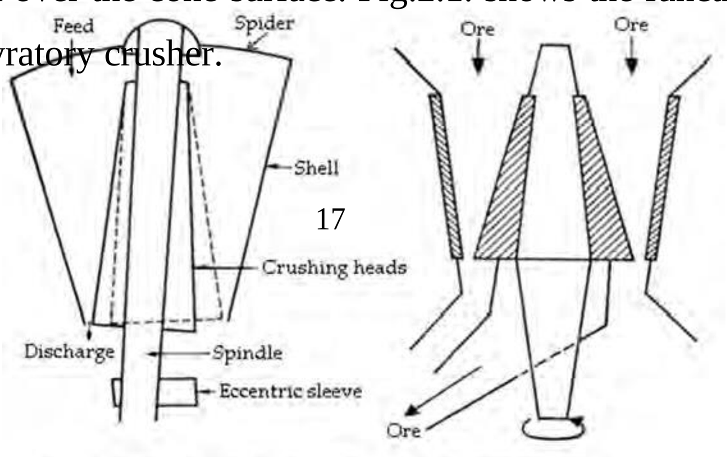


Fig. 2.5. Functional Elements of Suspended Spindle Gyratory Crusher.

Since the jaw movement is largest at the bottom, the operational characteristics of the gyratory crusher are similar to Blake jaw crusher. The machine operates continuously throwing product all around the periphery at different instants. When one point on the periphery is involved in crushing the opposite point is set at maximum opening to accept feed into the V- shaped crushing head. This crusher mainly employs compressive force for size reduction. The materials for crushing head is had field manganese steel in cast form. The gyration speed varies from 125-425 r.p.m. As the gyratory crusher operates continuously, for an equivalent size of the crushing heads, the capacity per unit area of grinding surface of the gyratory crusher is much larger than that of Blake jaw crusher. As the crushing action is continuous, the fluctuating stresses on machine members are minimized and it consumes less power. Thus it has a better efficiency compared to jaw crusher. The product from gyratory crusher is much more uniform compared to the jaw crusher. Because of the high capital cost, the crusher is most suitable for very large output.

Characteristics of Gyratory Crusher:

1. At any cross section there are in effect two sets of jaws opening and closing alternatively like a conventional jaw crusher. Hence gyratory crusher can be regarded as a series combination of infinitely large number of jaw crushers of infinitely small width. Hence the capacity of the gyratory crusher is much greater than that of a jaw crusher having equivalent gape size.
2. It has more regular power draft due to continuous crushing action.
3. With respect to the reduction ratio, at fixed power consumption and equivalent capacity, both jaw and gyratory crusher are at par.
4. The rule of installing a gyratory crushers or jaw crusher is given by Taggart as follows:

If the hourly tonnage to be crushed divided by square of gape expressed in inches yields a quotient less than 0.115 than use a jaw crusher or

else use a gyratory crusher.

Mathematically:

$$\text{If, } \frac{T}{Gape^2} > 0.115, \textit{ select Gyratory crusher.}$$

$$\frac{T}{Gape^2} < 0.115, \textit{ select Jaw crusher ,}$$

Where, *T* is expressed in tons per hour and *gape* is expressed in inches. A comparison between jaw and gyratory crushers is given in the table 2.4.

Table 2.4. Comparison between Jaw & Gyratory Crusher:

usher	**ry Crusher**
ding on machine compon tent and the power draft ular.	n loading on the machine compo power draft.
g action is intermittent.	g action is almost continuous.
articular gape size the cap npared to gyratory crusher	same gape size the capacity is m r.
acceptance size is much l ed to gyratory crusher.	acceptance size is much less co sher for the some capacity.
particle size distribution &it has a reduction ratio l that of the gyratory crushe	niform sized product is obtained r *r.r.*
consumption is higher for ner for a particular *r.r.* & city.	e same r.r. & capacity, the gyrat ner requires less power.
sher is less efficient comp y crusher It has an efficien 0%.	n efficiency of 30 - 50%.
ar on the jaw plates is not rm which causes heavy we w plates at certain areas. lates aced frequently.	ar on the crushing cone is quite u e bottom opening changes, th can be lifted up by the variable duce the gap. heads can serve for a longer time
ch variation can be obtain regards to product particle	ariation in product size can be o the setting of the central shaft. e varied as per requirement.
low cost of installation.	high cost of installation.
ter for lower production ra	ter for higher production rates.

Intermediate Crushers:

Generally products from the jaw crusher or gyratory crusher are not fine enough for the complete liberation of mineral grains and needs further size reduction. The product is charged into either cone crusher or crushing rolls for further size reduction. Cone crushers and crushing rolls are the equipments for intermediate range crushing.

Cone Crusher:

This type crusher is a newer development. They have gained wide popularity because of their economical operation in the intermediate range. The general types are: Simon's Cone Crusher and Telsmith Gyrosphere.

The construction of this cone crusher is much similar to gyratory crusher (Figure2.6.) though the feed size is much smaller and the product is much finer. Here both the rotating inner cone & stationary outer cone apex point upwards. The outer stationary cone is fixed on to the main frame while the inner crushing head is mounted on a heavy central shaft rotating eccentrically. The material used as crushing heads is *hadfield manganese* cast steel containing at least12%Mn. The sectional view of a cone crusher is shown in the figure 2.6.

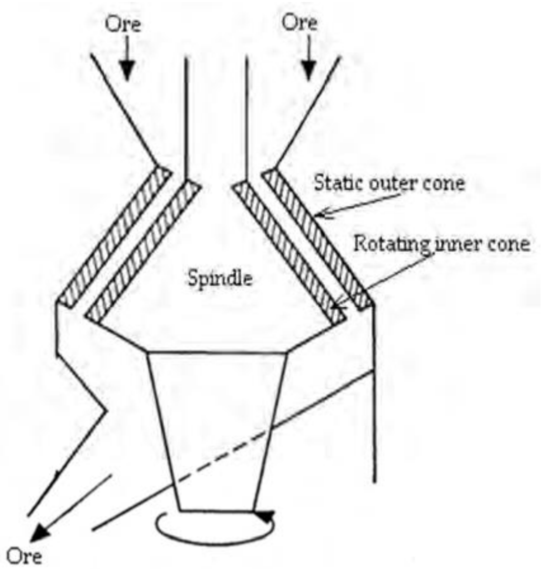


Fig.2.6. Sectional view of a Cone Crusher.

The central shaft is fixed with an adjustable bearing and is mounted on an eccentric drive. Due to the adjustable bearing on the central shaft, the position of the internal cone can be altered so as to provide a variable discharge opening (*set*) as per the requirement. This arrangement also takes care of the wear on the crushing faces which may enlarge the set. The eccentric performs the same work as does in the case of gyratory crusher.

Due to this the inner cone (crushing head) alternately approaches and recedes from a particular point on the periphery of the outer cone resulting in continuous crushing action. This results in regular power draft and much finer product at a better efficiency. The efficiency of the Cone crusher is comparable to that of the gyratory crusher.

The crushing forces here are compressive and frictional in nature. Compared to crushing rolls they have better capacity with comparable product fineness. To operate the cone crushers most efficiently, a dry feed, free from fines are to be used. If wet ore is used the cone crushers may clog. The problem of clogging in cone crushers makes it necessary to use efficient screens in closed circuit with them.

LIMITATIONS:

1. It operates only on closely sized brittle material.
2. It has a low reduction ratio.
3. It needs extensive lubrication of all its moving part regularly.
4. It operates best in closed circuit grinding.

Crushing Rolls:

This is an important class of intermediate comminution machine in the intermediate range of size reduction. Crushing rolls consists of pair of heavy cylindrical rolls revolving towards each other so as to nip a falling ribbon of rock and discharge it crushed below rolls. They were invented around 1850A.D.

Rolls:

1. It has a reduction ratio (*r.r*) is around 3 - 4 only which is very low compared to other size reduction equipments.
2. It yields a uniform sized product.
3. The product of the crushing rolls contains fewer fines as the mastification time is limited and no repeated crushing takes place.

4. Capacity:

Capacity of the roll crusher depends on the following factors:

i. Speed of revolution (*N*).

ii. Width of the faces (*W*).

iii. Diameter of the rolls (*D*).

iv. Set (*S*), the inter roll distance

v. Specific gravity of rock ( ) lb /in^3

The theoretical capacity in tons/hr is given by the expression:

$C = 0.0034\ N\ DWS\rho$, where *W*, *D* & *S* are expressed in inches and in *lb*/in^{-3}. Or, $C = 1.885\ N\ DWS\rho$ kgh^{-1}, where *W*, *D* & *S* are expressed in meter and is expressed in kg m^{-2} . The actual capacity is considerably less and is only around 10-30% of the theoretical capacity. If the set(*S*) is nil the capacity of the rolls is also nil.

6. Rolls can be operated either wet or dry. Dry crushing has a lower output but causes lesser wear of the rolls.

5. It is best operated on choke feeding for maximum output. In open feeding the output is less.

Uses:

The rolls are most suitable in effecting only a smaller size reduction in a single operation. Therefore, it is common to employ a number of pair of rolls in series to achieve higher reduction ration. Crushing rolls are extensively used in crushing oil seeds, gun powder and coal because of lower residence time of the feed as lower residence time reduces the effect of heat on the feed material.

Problem 1:

Coefficient of friction between rock& roll surfaces is μ = 0.4. What is the minimum roll diameter to reduce 1.5" piece of rock to 0.5"?

Solution:

Let us draw a functional figure of a roll crusher as below:

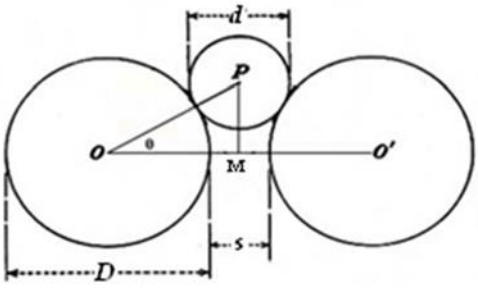


Fig.2.10. Angle of Nip in Roll Crusher.

Limiting value of the angle of nip is described by the relation, tan θ = μ . So, tan θ = 0.4. Hence, θ = 21°48'.

Considering the right angled triangle *POM* we have: Cos θ = *D*+*S*/*D*+*d* = Cos 21°48' = 0.9285 ---(1),

Where, *D* = Roll diameter *d* = Particle diameter = 1.5" *S* = Roll gap = 0.5"

Now substituting the values of the parameters in the equation (1) we have: 0.9285= *D*+0.5/*D*+1.5

Further solving for *D* we have:

D = 12.5 inches.

Problem2:

What should be the diameter of a set of rolls to take feed of size 38.1mm and crush to 12.7mm if the coefficient of friction μ *is 0.35?*

Solution:

Let us draw a functional figure of a roll crusher as below:

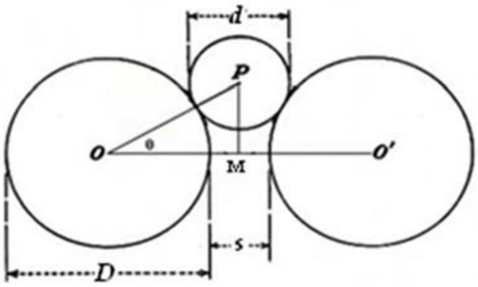


Fig.2.10. Angle of Nip in Roll Crusher.

Limiting value of *angle of nip* is described by the relation tan θ = μ. So, tan θ = 0.35. Hence, θ = 19°17'.

Considering the right angled triangle *POM* we have:

Cos θ = *D*+*S*/*D*+*d* = Cos19°17' = 0.9438 --- (1)

Where, *D* is the roll diameter = ? *d*, Particle diameter = 38.1mm. *S*, roll gap = 12.7mm.

Using the known values of the parameters in the relation (1), We have: 0.9438 = *D*+38.1/*D*+12.7

And solving for *D* we have: *D* = 413.86mm.

Feeding Systems in Comminution Equipments:

There are two distinct methods of feeding material to a crusher. They are:

a. Free Feeding.
b. Choke Feeding.

Chapter- 3

GRINDING

➢ Explain the principle of ball mill operation, open circuit grinding, close circuit grinding, dry & wet grinding.

Open and Closed Circuit Grinding Operations:

The usual meaning of grinding here is comminution and has nothing to do the product particle size. In many mills the feed is broken into particles of satisfactory size by passing it once through the mill. When no attempt is made to return the over sized particles in the product once again to the crusher for further size reduction the product simply passes-off to the next stage of size reduction. Such a method of size reduction at various stages till the desired product is obtained is termed as *open circuit grinding*.

A bright example is dodge crusher operating on choke feeding. This grinding may require excessive amount of power and much of the energy is wasted in regrinding the particles that are already fine enough.

In another method the partially crushed material is screened and the oversized material is returned back to the crusher for further crushing and the undersized product is given as the feed to the next machine for further size reduction. If such a method is followed in all successive crushers till the desired product is obtained it is termed as *closed circuit grinding*. This method of grinding operation is generally adopted as such a process has been found to be economical making full capacity utilization of all equipments efficiently.

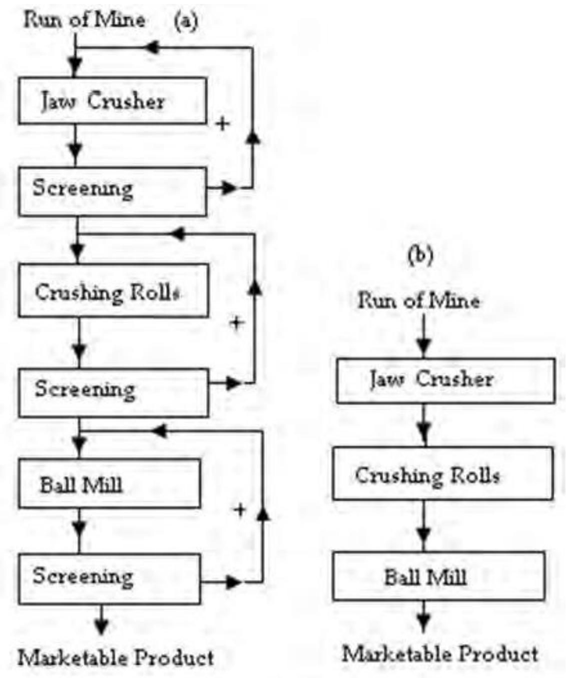


Fig.2.11. (a) Closed and (b) Open circuit Grinding.

This process avoids unnecessary regrinding. Figure2.11. shows the scheme of closed and open circuit grinding.

Fine Crushing or Grinding:

The fine crushing or grinding means product size less than 6*mm* and going down up to 200#(74μ*m*). The usual meaning of grinding is the comminution of an ore particle that has already been reduced to a size less than 6*mm* size by crushing.

Hence any comminution process aiming at a product size less than 6mm size is known as grinding. Grinding is a slower process usually carried out in a ball or tumbling mill or any other equipment like tube, rod & pebble mill. These mills perform size reduction in closed chambers containing hard balls, rods or quartz pebbles as grinding media.

Classification of Ball Mills:

Ball mills can be classified according to the

a. Shape of the mill.

b. Methods of discharge of the ground ore.

c. Weather the grinding is conducted dry or wet.

1. Shape of the Mills:

According to the shape the mills are classified as:

1. Cylindro-conical mills: Harding mill (where feed & discharge ends are fixed).
2. Cylindrical mills. This represents the usual ball mills.

Figure 2.12 shows the shape of different ball mills schematically.

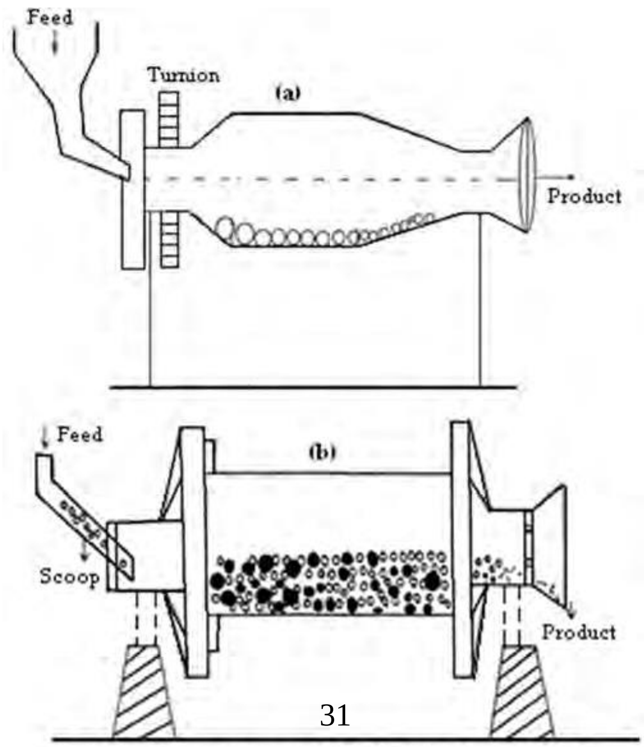


Fig.2.12. Schematic view of Ball Mills; (a) Hardinge Mill (b) Cylindrical Mill.

2. Method of Discharge:

Cylindrical mills are also classified according to the mode of product discharge taking place from the mill. According to the discharge method mills are classified as:

a. Peripheral discharge mill: Discharge of the ground product takes place through meshed cylindrical shell.

b. Grate mill: Discharge of the ground product takes place through a screen extending as a diaphragm across the full section of the mill at the discharge end.

c. Overflow mill: Discharge of the ground product takes place by free overflow from the axis of the mill.

GRINDING MILLS 273

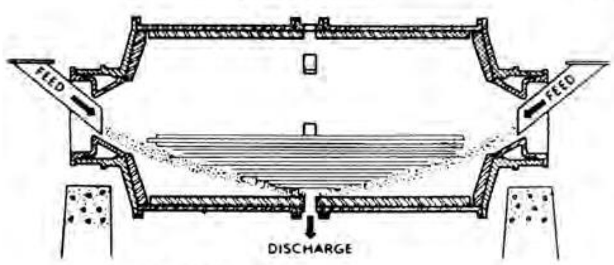


Fig. 7.16. Central peripheral discharge mill.

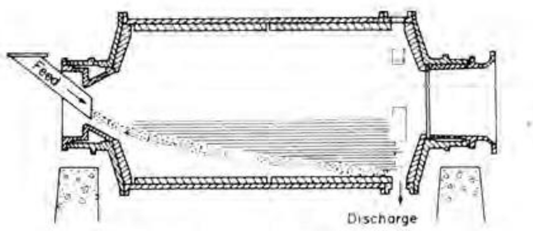


Fig. 7.17. End peripheral discharge mill.

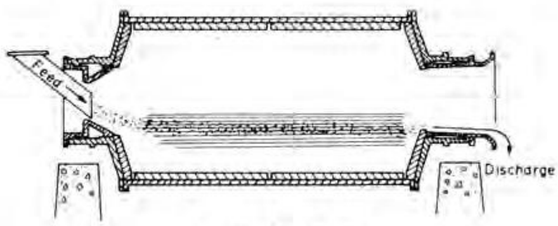


Fig. 7.18. Overflow mill.

Mechanical construction of a Cylindrical Ball mill:

Ball mill has few important components as follows:

1. Cylindrical shell.
2. Inner surface or liners.
3. Balls or grinding media.
4. Drive.

1. **Cylindrical Shell:**

It is the rotating hollow cylinder partially filled with the balls. The ore to .be crushed is fed through the hollow turnnion at one end & the product is discharged through a similar turnnion at the other end. The material of construction for this hollow shell is usually high strength steel. The shell axis is either horizontal or at a small angle to the base. Large ball mills have a length of 4 - 4.25 mts, diameter of 3mts. They use hardened steel balls of size varying between 25-125 *mm*.

2. **Inner Surface or Liners:**

As the grinding process involves impact and attrition the interior of the ball mills is lined with replaceable wear resisting liners. The liners are usually high manganese alloy steels, stones or rubber. Least wear takes place on rubber lined interior. As the coefficient of friction between balls and steel liner is specifically large, the balls are carried up taken to a higher height along the inner wall of the shell and dropped down onto the ore with a larger impact force resulting in a better grinding.

3. **Balls (Grinding Media):**

The balls are usually *cast steel* unless otherwise stated. In some cases *flint* balls may be used. The diameter of the grinding media varies from 1-5inches. The optimum size of the ball is proportional to the square root of the feed size. The ball and liner wear are usually in the range of 450 – 1250 and 0.50 - 250 grams per ton of ore ground.

4. **Drive:**

The mill is rotated by electric motors connected through reduction gear box - ring gear arrangement.

Theory of Ball Mill Operation:

Ball mills may be continuous or batch type in which grinding media and the ore to be ground are rotated around the axis of the mill. Due to the friction between the liners–balls & liners–ore lumps, both the ore and balls are carried up along the inner wall of the shell nearly to the top from where the grinding media fall down on the ore particles below creating a heavy impact on them. This usually happens at the toe of the ball mill.

The energy expanded in the lifting up the grinding media is thus utilized in reducing the size of the particles as the rotation of the mill is continued. In fact the grinding process is attributed to three different stages of ball mill working. They are:

a. Cascading (attrition between the balls and particles).

b. Cataracting (impact of the ball on the particles).

c. Centrifuging.

All these stages of working are shown schematically in the figure 2.13below.

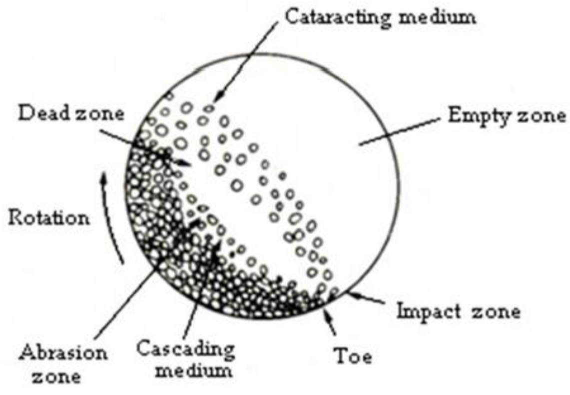


Fig.2.13. Different Stages and Zones of a Ball Mill.

Effective grinding depends on the rotational speed of the mill. If the mill operates at a low speed balls will be carried up along the inner wall to a certain height, but not large enough to give an impact force. Rather, they roll over each other or slip over. This type of operational condition is known as cascading of the mill. Even then some grinding is performed due to attrition. If the speed is raised, the balls start moving up further along the inner wall and suddenly fall form a greater height imparting an impact force at the toe of the mill. This impact is largely responsible for most of the grinding (Fig 2). This condition is known as cataracting. If the speed of rotation becomes too high, the balls are carried over and over again all along the inner lining as if they are sticking to the inner wall and there is hardly any grinding. This condition is known as centrifuging of the mill. If the speed of the ball mill is too low cataracting does not occur. Rolling down of balls and particles lead to particle rubbing and limited grinding only is possible. At the other extreme, that is at very high speed the mill, centrifuging occurs leading to little or no grinding. So mill is to be operated between these two extreme speeds.

Critical Speed of the Ball Mill:

The minimum rotational speed at which centrifuging occurs in a ball mill is defined as its critical speed. It has already been noticed that no grinding takes place in the ball mill when it centrifuges. So the operating speed of the mill should always be less than its critical speed enabling the media to deliver impacts at the toe or knee of the mill to result in grinding. The ***critical speed*** of a ball mill is of immense practical importance with regards to its efficient working.

Determination of Critical Speed of a Ball Mill:

Assumptions:

1. Let the radius of the cylindrical ball mill be, *R*.
2. Only single sized media of radius *r* is used in the mill.

During the rotation of the mill the grinding media is carried up along the inner wall of the mill shell. At any particular instant the forces working on the media is shown schematically in the figure2.14.

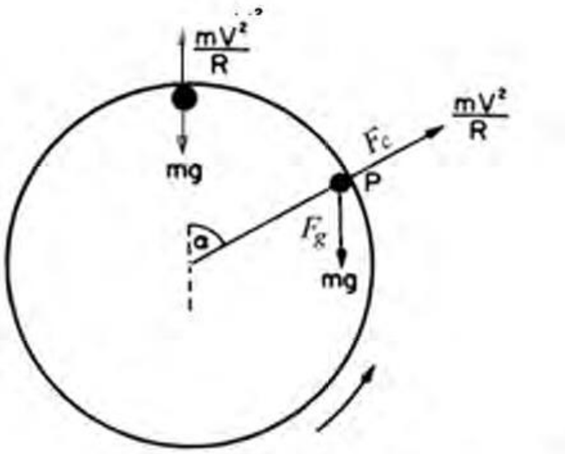


Fig.2.14. Forces Working on the Grinding Media.

Different forces working are:

1. A *centrifugal force* F_c working radially away from the centre of mill.
2. The *gravitational force* F_g acting vertically downward from the centre of the particle as shown in the figure2.14.

The speed at which the outer most balls may lose contact with the inner wall of the mill depends on the balance between *gravitational* & *centrifugal forces.*

Mathematically: Centrifugal force,

$$F_C = \frac{mv^2}{(R-r)} \quad ------ (1)$$

Where, $(R - r)$ is the radius of rotation, gravitational force, $F_g = mg\cos\theta$, g is the acceleration due to gravity and θ is the angle the particle at the centre of the mill.

Let v , be the linear speed of the cylindrical shell at the periphery. Converting the linear speed to rotational speed of the ball mill we have: $v = [2\pi (R - r) N]$, where, N is the rotational speed of the mill. The media will ride up to a point along the inner wall of the mill as long as the centrifugal force is greater than the gravitational force working on the ball. At any point if equilibrium is established, we have:

$$mg\cos\theta = \frac{mv^2}{(R-r)}$$

$$\Rightarrow mg\cos\theta = \frac{m[2\pi(R-r)N]^2}{(R-r)} = \frac{m[4\pi^2(R-r)^2N^2]}{(R-r)}$$

$$\Rightarrow g\cos\theta = 4\pi^2(R-r)N^2$$

For centrifuging condition, the media has to reach the topmost position as shown in the figure2.14 and then roll down to the other side without losing contact with the inner wall of the mill. Hence under the critical condition of centrifuging the media should at least reach the top most position. The speed at which this just happens is known as the critical speed N_c of the ball mill. To achieve such a condition, θ has to be 0^o .

Now, $$N_{Critical} = \frac{g}{4\pi^2(R-r)}$$

$$\Rightarrow N_C = \frac{1}{2\pi}\sqrt{\frac{g}{(R-r)}}$$ is known as the *critical speed* of the mill.

In different units the critical speed of the ball mill can have values as follows:

$N_C = \frac{42.3}{\sqrt{D-d}}$; D & d expressed in meter. -------------- (1)

$N_C = \frac{76.65}{\sqrt{D-d}}$; D & d expressed in feet. ------------- (2)

Usually the ball mill is rotated at 65-80% of the theoretical critical speed. The lower value is for wet grinding while the higher value is opted for dry grinding.

Characteristics of Ball Mill Working:

1. **Speed and Energy Input Interrelation in Ball Mill:**

Speed of the ball mill should be as high as possible without centrifuging. Initially the work input increases steadily as the speed of the mill increases. It reaches a peak at a particular speed and there after the work input decreases rapidly with the increase in speed. This is shown schematically in the figure 2.15.

2. **Ball Load:**

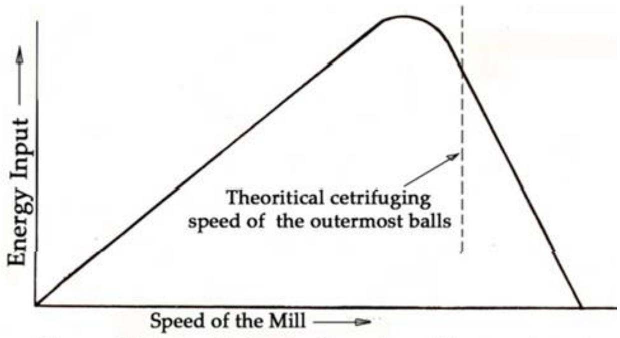


Fig. --- Ralation between Speed and Energy Input

It is defined as the volume that is occupied by the grinding media out of the total volume of the ball mill without ore or water in it. The ball load should be such that it is slightly more than 30% of the total volume of the ball mill. During general operation media occupy between 30-50% of the volume of the mill. When a mill is operated for the first the time, balls of various sizes rather than single size are charged into the shell. The Justification for the use of the various sizes is obvious. If balls of definite size are charged, the *interstitial pores* created by the uniform sized spheres will work as void spaces and ore particles of that particular void size if caught in the void will not be crushed further. So as to avoid such problems balls of various sizes are used in the mill when it is installed and operated for the first time. During grinding the balls themselves get worn-off which reduces the ball load. The reduced ball load is replenished at regular intervals with new ball(s) of largest size only. In fact the larger balls crush the feed material more effectively while the smaller ones are responsible for producing fines.

The energy that the mill is made to consume is a function of speed of the ball mill, ball load, specific gravity of the ore and dilution of the pulp. With the increase in ball load the energy input into the mill is increased gradually but not in direct proportion to the ball load till a maximum is reached. Thereafter the energy input decreases gradually to zero as it had increased earlier.

This is due to the fact that, as the ball load is increased, the centre of gravity of the load comes nearer and nearer to the axis of rotation of the mill which decreases the energy input to the mill. A pulp density of 60 - 75% solids results in maximum energy input. The

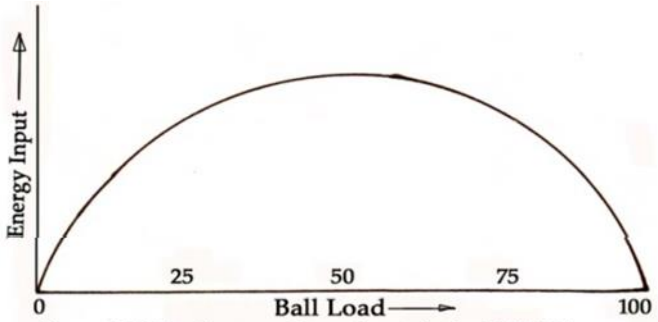


Fig. --- Ball load versus Enery input in the Ball Mill.

energy input versus the ball load is shown schematically in the figure 2.16.

3. **Reduction Ratio:**

The reduction ratio that can be obtained in the ball mill is large compared to reduction ratios obtained in primary or secondary crushers. It may range from 50 -100 for a ball mill-classifier circuit If the *r.r.* is high along with large capacity, it will be more economical to use ball mills in series. The first in the series may be with *r.r* of 20 while the last one may be a fine grinder having

r.r of 5 resulting in an effective *r.r* of 20X5=100.

4. **Capacity:**

The capacity the ball mill depends upon its size, hardness of the ore and the reduction ratio attempted. Ball mills yield 1-50 ton / hr of ore fines with 90% passing through 200 # screen.

5. **Energy consumption:**

Average energy input into the ball mill is around 16 kWh / ton of ore ground.

Factors affecting the size of the Product in a Ball Mill:

1. Rate of feed:

Higher the rate of feed lesser is the size reduction since the residence time of the ore particles in the mill is reduced.

2. Properties of the feed ore:

Under given operating conditions larger the feed larger will be the product. A lower reduction ratio (*r.r*) is obtained with a hard material.

3. Weight of the ball:

Heavier balls produce finer product. Since the optimum condition is 50% ball load by volume, the weight of the balls is normally altered by the use of materials of different specific gravities.

4. Diameter of the ball:

Smaller balls facilitate the production of finer material but they are not effective in grinding larger sized particles in the feed. The limiting size reduction obtained with a given size of balls is known as *free grinding*. As far as possible smaller size balls are to be used.

5. Slope of the mill:

Increase in the slope of the mill increases its capacity of the mill. But a coarser product is obtained as the retention time of the feed in the mill is reduced due to higher slope.

6. Discharge freedom:

Increasing the freedom of discharge of the product has the same effect as that of increasing the slope.

7. Speed of rotation:

The mill should be operated at speed less than N_C . Usually it is operated at a speed, $N_{operational}$ = 0.65 - 0.75 N_C

8. Level of Material in the Mill:

Power consumption is reduced by maintaining a low level of material in the mill. If the level is increased the cushioning action is increased and energy is wasted in producing excessive fines. Total level of material in the mill should be 50% maximum out of which at least 30% should be the ball load.

Advantages of the Ball Mill:

1. The mill can be used both for wet and dry grinding.
2. The cost of installation of a ball mill is low.
3. The ball mill can use an inert atmosphere to grind explosive materials.
4. Media used for grinding is relatively cheap.
5. The mill is suitable for grinding materials with any degree of hardness.
6. It can be operated in batches or continuously.
7. It is used for both open and closed circuit grinding effectively.

Dry & Wet Grinding

It is to be noted that ball mills can he operated dry or wet. Mills are usually employed to grind ore in wet condition. But for some specific purpose essentially in chemical industries dry grinding is employed.

During dry grinding the mills are connected with pneumatic classifiers in closed circuit to produce extremely fine powder. Pulverized coal is obtained in this manner.

Advantages of Wet Grinding Over Dry Grinding:

Though wet grinding is generally applicable in low speed mills there are number of advantages of wet grinding over dry grinding:

1. Wet grinding facilitates better removal of the product, eliminates dust problem, lessen the noise and heat produced though the wear may actually increase by 20 %.
2. Power consumption is lowered by10-30% over dry grinding per ton of product.
3. The capacity increases per unit volume of the mill.
4. This grinding makes wet screening possible for producing materials in narrow size range.
5. Dust problem is eliminated.
6. Wet grinding makes handling & transportation of product easier.
7. Sticky solids are more easily handled.

Hardinge Mill Or Cylindro-Conical Mill:

The Hardinge mill consists of two conical sections connected by a central cylindrical section. The mill is supported by the end bearings on which the hollow turnnions are mounted. The mill is made to rotate by gear - pinion

arrangement. Feed enters through the left side 60° cone to the primary grinding zone where the diameter of mill is highest.

Product pours out as a continuous stream of thick pulp through the

right side 30° cone. It is said that conical sections compel the coarse particles and the larger balls to seek the cylindric

section of larger diameter while fine particles & smaller balls are found in the smaller diameter conical section to the left. As the mill is rotated the larger balls move towards the point of maximum diameter or feed end while the smaller balls migrate towards the smaller diameter or discharge end. So from

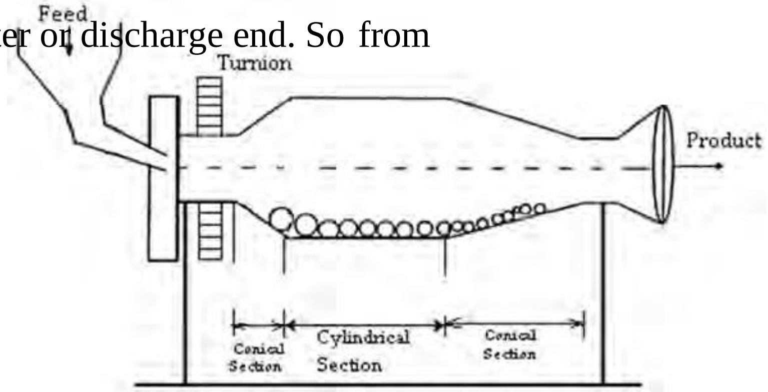


Fig.2.17. Schematic Diagram of a Hardinge Mill.

constructional point of view preferential grinding of coarse particles is performed by the large balls & fine grinding is performed by the smaller balls. The Hardinge mill is shown schematically in the figure 2.17.

Hardinge mills are widely used in metallurgical plants and are usually adopted for wet grinding. Dry grinding of coal, pulverization of lime stone, clay & cement clinker is possible in this mill. This mill operates continuously. This type of mill is shown schematically in the figure 2.18. This mill can be further classified according to the freedom of discharge employed as discussed earlier.

Continuous Cylindrical Mill:

The grinding shell is totally cylindrical with different sized balls in it. The feeding is continuous through the hollow turnnion connect at the central axis and the product comes out through the other turnnion connected at the opposite end. This can also be used for batch production.

Laws of Crushing:

The first step in ore beneficiation is to reduce the size of the ore by crushing & grinding, commonly referred as comminution. The main objective of comminution is to liberate the mineral particle from the unwanted gangue. This is achieved by detaching the mineral particle from the gangue. For different degree of liberation different types of crushing equipments are used. Though crushing and grinding equipments have been developed to a high degree of perfection and automation, not much change has been made in the theory of crushing or grinding. The design of equipments for size reduction largely depends on experience and empirical relationships. The most important consideration in any size reduction is the energy it consumes in performing the activity, as energy is costly. *The empirical relations between the energy consumption and size reduction are termed as laws of crushing*.

Rittinger was the first one to propose such a law termed as Rittinger law which was subsequently modified further by Kick and Bond. Presently we have three laws of crushing.

1. **Rittinger Law:**

Rittinger stated that," Energy expanded during comminution is proportional to the new surface area created as a result of particle fragmentation''. Mathematically, the statement can be represented as:

$E = K_R (S_2 - S_1)$,

Where, K_R is called Rittinger's constant or work index and S_2 & S_1 are the final & initia1 specific surface areas respectively. In terms of particle diameter it becomes: $E = K_R \left(\frac{1}{d_2} - \frac{1}{d_1}\right)$,

Where, d_2 & d_1 are final & initial diameters of the particle respectively.

Rittinger's law applies fairly well in the fine grinding range of 10-1000 *m* in size. Rittinger's law is quite accurate in calculating the energy consumed during fine crushing.

2. **Kick Law:**

According to Kick's law of crushing," The energy consumed during size reduction is directly proportional to the logarithm of size reduction ratio (*r.r*)". If d_2 & d_1 are the final & initial diameters of the particle during size reduction the reduction ratio(*r.r*) is d_1 / d_2. Then energy consumed in size reduction is proportional to $\log(d_1 / d_2)$.

Mathematically this can be expressed as:

$E = K_k \log(d_1 / d_2)$, where K_k is the Kick's law constant.

Rittinger's law is most successful in the range of fine grinding while Kick's law is successful in predicting the energy consumption during coarse crushing that is in the range of 1*cm* and above.Neither of the above laws predicts the energy consumption in the intermediate range of size reduction taking place in cone crusher or crushing rolls, a third law has been proposed by Bond.

3. **Bond Law:**

It is stated as "*The total amount of work input represented by a given weight of crushed or ground product is inversely proportional to the square root of the product particle diameter*".

As per the law:

$$W_b \propto 1/\sqrt{D_p}$$

Where, D_p is the average size of the particle and W_b is the Bond's work input during crushing.

Mathematically this law can be written as:

$$W_b = 10\, W_i \left[\frac{1}{\sqrt{D_p}} - \frac{1}{\sqrt{D_f}} \right]$$

Where, D_p and D_f are the average size of the product &feed respectively and W_i is the Bond' work index an intrinsic property of the material being crushed. Work index is the comminution parameter that expresses the resistance of the material to crushing and grinding. Numerically it is equal to the work input in kWh/ton that is required to reduce a material from an infinitely large sized feed to a product 80% of which passes through the screen of 100 μm aperture size.

Ore Grindability:

Ore grindability refers to ease with which the material can be crushed or ground. The most widely used parameter for measuring grindability is the Bond's work index, W_i . If the breakage characteristics of the material remain constant over all the size ranges, then the calculated work index also remains constant and expresses the overall resistance of the material to breakage as per Bond's law:

$$W = 10\, W_i \left[\frac{1}{\sqrt{D_p}} - \frac{1}{\sqrt{D_f}} \right]$$

Criteria of Selecting Comminution Equipment:

The choice of machine for a given crushing operation will be affected by the following factors:

1. Size of the product required (coarse, intermediate or very fine) along with the size of the feed.
2. Quantity of the material to be handled (input/output capacity).
3. Physical properties of feed material to be crushed. Irrespective of the feed and product sizes, the physical properties of the material under consideration are of primary importance in selecting suitable crushing equipment. Hence it is imperative here to discuss regarding the physical properties of the material in more detail.

Significant Physical Properties of Feed Ore:

1. **Hardness:**

Hardness of the mineral affects the power consumption and wear of the machine. For hard and abrasive minerals low speed machines developing high compressive stresses are preferred.

2. **Structure:**

Normal granular minerals like coal, ores and rocks can be effectively crushed employing normal compressive and impact forces. With fibrous minerals it is necessary to effect tearing action for size reduction. Hence knife edge mills are widely used for asbestos and mica live minerals.

3. **Moisture Content:**

It is found that minerals with higher moisture content (5-50%) do not flow efficiently. Under such conditions they tend to clog the crusher. Wet grinding can be carried out satisfactorily on these minerals.

4. **Crushing strength:**

The power required for crushing is almost directly proportional to the crushing strength of the minerals.

5. **Friability:**

The friability of the mineral is its tendency to fracture during normal handling. Crystalline minerals will break along well defined planes and power required for crushing of such minerals will increase as the product particle size is reduced.

6. **Stickiness:**

A sticky mineral will tend to clog the equipment so should be ground in a mill that can be readily cleaned.

7. **Friction factor or Soapiness:**

If the coefficient friction is low, usual crushing will be difficult. In such cases size reduction can be carried out by employing impact or shear forces.

8. **Explosive mineral:**

These mineral must be ground wet or in the presence of an inert atmosphere otherwise they may catch fire or explode.

9. **Mineral producing heavy dust:**

Dusts are harmful for health so should not be allowed to escape to the atmosphere. Special crushing methods are to be employed while crushing minerals producing heavy dust.

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CHAPTER 4

➢ **Explain the procedure for size analysis & use of standard screen as also screening techniques employed.**

Introduction:

Size analysis of various products of a crushing mill constitutes a fundamental part of the laboratory testing procedure. It is of great practical importance to make a correlation between the particle size and degree of liberation. Further it is to be understood that, particle size has a great role to play during reactions between solid - liquid or solid-gas. Most of our extractive processes, such as calcination, roasting, reduction, oxidation and leaching involve a solid and a gas or liquid phase. As the processes are diffusion control, the particle size plays a very important role during different *unit processes*. Rapid and efficient working of roasters, smelters, froth flotation cells & leaching tanks largely depends on the size of the beneficiated ore. So the product from the crushing equipment is to be analysed for its size for all practical purposes. Further the size analysis of the product is required to evaluate the energy consumption and the size reduction process it may require for further size reduction.

Particle Size & Shape:

The primary function of precise particle analysis is to obtain quantitative data about size and size distribution of the particles in the product material. The shape of the particle plays an important role in the size determination. The size of a spherical particle can be defined uniquely by its diameter. However, there is no unique dimension by which the size of an irregular particle can be described. The term most often used to describe an irregular particle is the equivalent diameter ($\bar{d}$). There can be various shapes to
describe a particle as discussed below:

1. Accicular: Needle like particles.
2. Angular: Sharp edged polyhedrons.
3. Crystalline: Particles of regular geometric shapes.
4. Fibrous: Regular or irregular thread like particles.
5. Dendritic: Particles having branched crystalline structure.
6. Flaky: Plate like particles.
7. Granular: Equidimensional irregular shaped particles.
8. Irregular: Lack of any symmetry in the particles.
9. Nodular: Particles having rounded irregular shape.
10. Spherical: Globular particles.

Particle Size:

The crushed ore particles are generally irregular in shape and it is quite difficult to define the size of the particle uniquely. In case of spherical particles, the diameter is the size. For cubes the edges, the long diameter or diameter of a sphere of equal volume may be considered as the size. But for totally irregular particles there is no such standard method. So it is impossible to define what is meant by size of the particle.

Common Methods of Size Analysis:

Particle size is usually defined as the narrowest regular aperture through which mineral particle passes through. Through this definition is applicable to polyhedrons it is not valid for rod shaped narrow particles. Particle size can be determined by various methods as described below in table3.1.

Table 3.1.Methods of Particle size Determination:

iods	**roximate size range rons) (1** $\mu m = 10^{-}$
e analysis	100000 -10
iation	40 – 5.0
cal microscopy	50 – 0.25
nentation(gravity	40 – 1.0
nentation(centrif	5 – 0.05
ron microscopy	1 – 0.005

L. Microscopic Measurement:

For measuring the particle size under microscope, it is customary to sprinkle them on a slide and to measure their diameter in random directions or in any two perpendicular axes within the plane of vision. In both the cases the smallest dimension is neglected. For number of particles the dimension x_i is measured and tabulated as follows:

Number of observations	x_i
1	x_1
2	x_2
n-1	x_{n-1}
N	x_n

Now average size, $\bar{x} = \frac{x1+x2+x3+xn-1+xn}{n}$

2. Elutriation:

Elutriation is based on the fact that a particle will just be sustained in an upward rising current of water or any other fluid if the velocity of the water current is equal to that which the particle would attain when falling in still water. This works on the principle of Stoke's law of settling.

3. Sieve Analysis:

This is the most important method of sizing the mineral particles. This is widely used to determine the efficiency of size reduction operations and also used as a yardstick for assessing the fineness of a ground product. As sieve analysis has been the most important method of size analysis it has become pertinent to discuss about the standard screens or sieves used worldwide for the purpose.

British Standard Sieves:

In the British system a screen is designated with a number called *mesh number* and the aperture of screen opening is termed as *mesh size*.

Mesh number is defined as the number of square openings available per linear inch length on the screen surface. If the screen has 4 openings per linear inch length of the screen surface then the mesh number of the screen is 4. Likewise we have screens of 20, 40 … 200, 270and 400 mesh number. When mesh number increases aperture of the screen opening decreases and vice versa. In British standard, aperture size of the successive screens varies with factor $\sqrt[4]{2}$. However, it is better to indicate the screens with their aperture size rather than their mesh number because screens with same mesh number may have different aperture depending on the thickness of the wire used to manufacture such screens. This is an inherent problem associated with the British standardization regarding classification of screens. The above drawback of the British system has been taken care of in the ASTM standardization where a screen is represented by its aperture rather than the mesh number. In the ASTM standardization, screen openings are regulated by the Tyler mathematical series where the opening of each successive screen usually varies with a factor $\sqrt{2}$.

Tyler Series and ASTM Standard Screens:

Tyler mathematical series is the most widely used for manufacturing ASTM standard screens for sieve analysis. The screens are made-up of bronze brass or stainless steel wires woven into a screen cloth having square openings.

In case of ASTM standard screens, the screen opening area of each successive screen is either double or half the area of the next screen in the series. This implies that the aperture size varies with a factor √2.

The 200# screen has an opening of 74μm and the lowest screen opening available in this series is 37μm. This is because below this opening fabrication of screens becomes very difficult. But there is no upper limit to the screen opening size. The 200 # sieve (74μm) is chosen as the reference screen in the ASTM standard sieve series and relates both ASTM and British standard screens. In the ASTM standardization, mesh number ranges from 3- 400.

ASTM Standard Sieve Series:

ı be	tur ım)	ı ber	ture ım)
	)		7
	9		5
	2		3
	2		7
	1		4
	3		4
	3		2
	9		7

Sieve or Screen Analysis:

Screen analysis is the experimental method to determine the average size of the crushed product. The product from jaw gyratory or any other crusher is hardly uniform in size. In fact the product consists of particles of various sizes and it is impossible & impractical to know the size of each product particle. Hence, an average size of the product is determined by sieve analysis method as it proves to be the quickest and most reliable method.

Average Size Determination for Large Sized Particles:

1. For large sized particles having a diameter of few centimeters, it is better to know the size of each particle and then average them out for calculating an average size. A sample is taken from the bulk by coning and quartering technique and the sample may consist of 10, 20 or 100 particles.

2. Measure-out the dimensions of each particle in three perpendicular directions to reflect dimensions in the three co-ordinates (*x, y and z*).

3. The data recorded for each particle is made into a table as shown below:

cle be	ension			

4. Then find out the maximum and minimum *d* value from the table. If, $d_{max} / d_{min} < 1.5$, use arithmetic, geometric or harmonic mean methods to find out the average size of the product as illustrated below.

a. Arithmetic mean diameter, $\bar{d} = \dfrac{\bar{d}_1 + \bar{d}_2 + \bar{d}_3 + - - - - \bar{d}_{10}}{10}$

b. Geometric mean diameter, $\bar{d} = \sqrt[n]{d_1 \times d_2 \times d_3 - - - \times d_n}$.

c. Harmonic mean diameter, $\dfrac{1}{\bar{d}} = \dfrac{1}{d_1} + \dfrac{1}{d_2} + \dfrac{1}{d_3} + - - - + \dfrac{1}{d_n}$.

Table. 3.3. Typical Screen Analysis data Of Ball Mill Product:

ing rons)		*n*	*ned*	*ng*

Simple and Cummulative Weight Percent Retained or Passing:

Weight percent retained is the percentage of weight retained on a particular screen basing on the original weight of the sample taken. By cummulative weight percent it is meant that the total weight which would be retained on a testing sieve or pass through the sieve if only one sieve were used for testing the whole sample.

Example:

Let us imagine 10, 14, 20, 28# screens are used in sieve analysis. Let the weights retained on the consecutive screens be w_1, w_2, w_3 & w_4

respectively. So the cumulative weight retained on 14# screen is: $w_1 + w_2$.

Similarly the cumulative weight retained on 28# screen is: $w_1 + w_2 + w_3 + w_4$. Now the cumulative weight percent can be calculated out by taking the total sample weight used in the screen analysis process. A plot of cummulative weight percent passing or retained against the aperture size is drawn as shown in the figure 3.1.

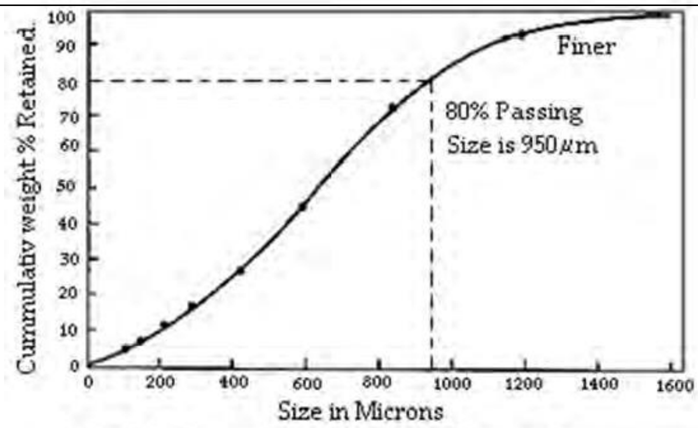

Fig. 3.1. Cummulative Charting of Size analysis.

A quick & easy method of determining the average size of the product from the screen analysis is the 80% passing size which is indicated in the plot 3.1.The eight percent (80%) passing size is accepted as the standard size of the crushed product universally unless otherwise stated. The standard condition may be changed as per the requirement and is to be specified by the buyer which may be 70 or 90% passing. This % passing means that at least that percent of the material would pass through on the specified sieve when screened. This kind of plots is most commonly used in mineral industries.

Screen Analysis Equipment:

For sieve analysis, screening is usually carried out in a mechanised sieve shaker called Ro-tap sieve shaker.

Ro-Tap Sieve Shaker:

Figure3.2.shows the Ro-tap machine schematically. It consists of a movable cage with a base *a* and a top plate *b* between which 13 half height or 7 full height sieves with pan and cover

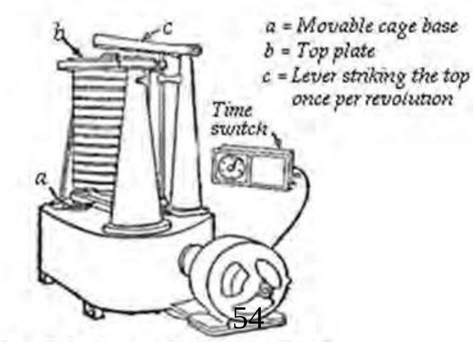

Fig 3.2. Rotap Sieve Shaker.

lid can be mounted.

The mounted sieves are subjected to rotary shifting motion while at the same time the lever *c* strikes the top plate once per revolution. This striking vibrates the screen cloth for better screening. A timer switch with the motor is used to control the time duration of screening. The machine is so designed that it performs the most ideal screening operation within the specified time period.

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CHAPTER -5

INDUSTRIAL SCREENING

4.1. Introduction:

By this time it has been clear that the screening of the crushed product is quite important in a large scale. Screening segregates the bulk of the crushed product into few fractions. This segregation is beneficial in many ways as follows:

a. Properly sized or the required sized material is charged into the next comminution equipments for further size reduction. Proper feed size reduces the overloading on the subsequent size reduction machines and increases the overall efficiency of the comminution.

b. Properly sized material can be charged into the process reactors such as smelters, roasters or calcinators making the process more efficient.

Till now screening has only been discussed on a laboratory scale but for industrial need, the screening has to be carried out in a much larger scale. Thus large scale screening is termed as *industrial screening* which differs from the laboratory screening practices in many ways. It is important to know the methods those are available and also the factors which affect the process of industrial screening.

4.2. Purposes of Screening:

1. To prevent the entry of undersized material to the crushing machines so as to increase the capacity and efficiency of comminution.
2. To prevent oversized material from passing to the next stage in closed circuit crushing or grinding.
3. To prepare closely sized feed for next stage of unit operation such as gravity concentration.
4. To prepare closely sized end product as per specification and requirement.

4.3. Mechanism of Screening:

When a crushed product is kept on a screen something would pass through &something would be retained on it. The material passing through screen openings is known as under flow or under sized while the material retained is known as over flow or over sized. So the basic fact attached to screening is the passage of under sized material through the screen. There are several factors affecting this passage. The factors are:

1. The absolute size of the screen openings.
2. The relative size of the particle to that of the screen aperture.
3. The percentage of open area available on the screening surface.
4. The angle at which particle strikes the screening surface.
5. The speed with which the particle strikes the screening surface.
6. The moisture content of the material to be screened.
7. The opportunity offered to each particle to hit the screening surface that is the probability that a particle will hit the screening surface before it is taken away by overflow.

4. 3. 1. Effect of Screen Opening Size:

The passage of undersized particles though each opening is inversely proportional to the screen aperture. This leads to the fundamental conclusion that the other conditions remaining unchanged the capacity of a screen given in *tons per hour per sq. foot per millimeter screen aperture* increases with increase in screen opening size.

4. 3. 2. Effect of Relative Particle Size:

The relative size of the particle and the aperture size control the passage of the particles through the screen. Larger sized particles with larger aperture get screened easily as compared to smaller sized particles on finer screens.

4. 3. 3. Percentage of Open Area on the Total Screening Surface:

If the total surface area is one square meter and there are only few openings on it then the quantity of screened material will also be quite less.

If large numbers of openings are available on the same screen area, automatically quantity screened would go up. However there is always a limit to the extent of open area which can be available per unit surface area of the screen. This is due to the fact that the screens are made up of materials such as rods, wires & etc having definite dimensions. These dimensions depend strongly on the load that the screen is going to bear during screening operation. The dimensions of the wire or rod increase with an increase in aperture size so as to have better strength.

4. 3. 4. Angle at Which the Particle Strikes Screening Surface:

The crushed particles are always irregular in size and shape. Hence, the angle at which the particles hit the screen surface is extremely important. A rod like particle gets through an aperture which is little above its diameter if the particle hits the screen surface with its long axis perpendicular to the screen surface. However, the same particle will not be able to pass through a screen of larger aperture when the particle hits the screen surface with its long axis parallel to the screen surface. Most efficient results are obtained, when the

particles hit the screen surface at angle in the range of $45 – 60^{o}$.

4. 3. 5. Speed at which the Particles Strike the Screen Surface:

Speed of movements of the particle over the screen surface is also an important factor in controlling the extent of screening. It is important to note that effective screening is zero when the speed of the particle is zero on the screen surface. With an increase in particle speeds the effectiveness of the screen increases. However, if the speed is excessively high the particle passes off to the overflow before it gets a chance to pass through any particular aperture of the screen. This implies that the particle gets very little scope to pass through the sieve. Further the particle movement during screening is also quite important as it reduces the effect of oversized particles trying to blind the screen. If the screen does not vibrate properly it may be clogged completely by the oversized particles in the product and thereafter no screening would take place. For effective screening, both vibratory and circular motions are usually employed simultaneously.

4. 3. 6. Effect of Moisture in the Feed:

When little moisture is present in the feed material to be screened, the screening efficiency gets reduced enormously.

In fact, it becomes impossible to screen them effectively. The difficulty is due to the fact that, moisture tries to bind few smaller particles into larger aggregates and such aggregates are large enough to pass through the smaller screen opening. It is found that either totally dry or wet pulps can be screened with relative easiness.

4. 3. 7. Probability Effect:

It is of utmost importance that each particle is given an opportunity to strike the screen surface so as to get screened or to pass-off to the overflow. If the particle is given 2, 5 or 8 chances of striking the screen surface, it can always be qualitatively pointed out that probability of screening is increased when more and number chances are given to the particles to interact with the screen surface.

4.4. Screening Surfaces:

Screening surfaces are the surfaces through which screening takes place. Screening surfaces are categorised according to the mode of their manufacturing classified as follows:

4. 4. 1. Parallel Rods:

Such a surface is usually made-up-of steel bars, rails, channels and etc.
It can also be made from wood and bamboo.

4. 4. 2. Punched Plates:

The surfaces are punched steel sheets or plates of various patterns.
The openings are normally circular, rectangular, hexagonal and slot like.

4. 4. 3. Woven Wires:

The screening surfaces are woven carefully by gauged wires. These wires are generally made up of steel, bronze, copper & monels. The screen surfaces are shown schematically in the figure 4.1.

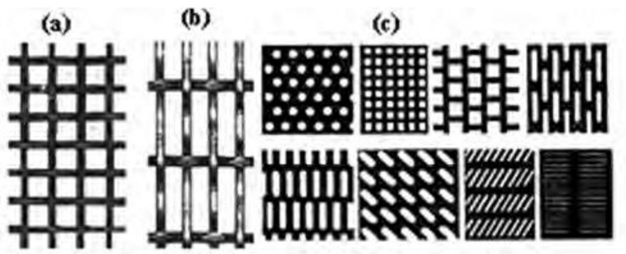

Fig.4.1. Screening surfaces; (a) Woven wires,(b) Parallel Rods, (c) Punched plates

4. 5. Types of Screens:

The screens are classified as:

1. Stationary.
2. Moving.

4. 5.1. Stationary screens:

These screens are of limited use but are not totally obsolete. These screens are grizzlies. They consist of parallel rods, bars or woven wire mesh set at an angle to the ground. They have heavy screening surfaces. The bars are usually held together at right angles to their length and are spaced at the desired distance sleeves on the bolts. They are usually employed in case of coarse crushing. A slope is generally provided so that the material fed onto the screen surface would roll down facilitating better screening. A typical stationary grizzly is shown in the figure 4.2.The major disadvantage of this type

Fig.4.2. Stationary grizzly.

of screen is *clogging*. Rails are used under severe service conditions with openings greater than five (5) inches.

4. 5.2. Moving Screens

1. Moving grizzlies.
2. Trommels or Revolving screens.
3. Shaking screens.
4. Vibrating screens.

4. 5.2.1. Moving Grizzlies:

The grizzly is made up of rods and bars but have movements as compared to stationary grizzly.

In moving grizzlies alternate bars or rods alternatively rise and subside, so that the feed material move forward gently with sufficient turning over. There are different grizzlies such as:

a. Moving-bar grizzly.
b. Chain grizzly.
c. Travelling grizzly.
d. Disc or Roller type grizzly.
e. Vibrating grizzly.
f. Shaking grizzly.

4. 5.2.1.1. Advantages of Grizzlies

a. Low floor space is required for installation.
b. They act as feeders to intermediate crushers.
c. Result in better screening than stationary screens.

4. 5.2. 2. Trommels or Revolving Screens:

Revolving screens or Trommels have been used more widely than any other type of movable screens but recently they have been replaced by vibrating screens. Trommel consists of rotating cylindrical, prismatic, conical or pyramidal sells of punched plates or thick woven wires. A trommel has one or more shells which are arranged in a concentric manner. When the trommel has only one shell, it is known as simple trommel. With more than one shell it is known as compound trommel. In case of compound trommels screen opening aperture) gradually decrease from the innermost screen to outermost screen. The trammel is commonly 3 - 4ft in diameter and 5-10ft. in length. The Shells lire driven by a central shaft attached to them by 4 or 6 armed spiders. The material to be screened is charged into the inner most shell and is made to flow out peripherally. When the trommel is rotated by the central shaft the material inside starts revolving and gets screened. The under sized material comes out of the trommel all along the periphery & oversized material comes out at the other end.

Central shaft of the trommel is made to inclined on the horizontal to facilitate automatic flow of the material from one end (feed end) to the other end (discharge end) due to force of gravity.Cylindrical trommels outnumber all other types of trommels. Figure 4.3 shows a typical trommel schematically.

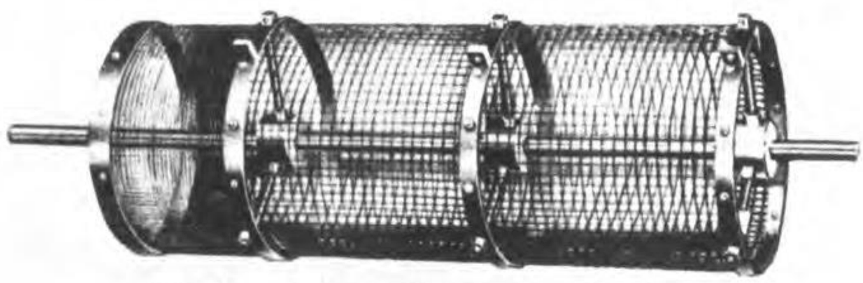

Fig.4.3. Schematic figure of a trommel.

Compound Trommels:

Compound trommels have two or more concentric screening surfaces on the same shaft. The coarsest is the inner most while the screen apertures reduce successively from inside to outside. They are used when several short- range products are desired from a single long-range feed and the floor space is limited. There may be conical and prismatic trommels but cylindrical is the most common one.

Advantages of Trommels:

1. It requires smaller floor space
2. It has a larger capacity per unit screening area.
3. It is cheap to operate.
4. Several fractions are obtained in one go.
5. Screening operation is quite efficient, can utilize both wet and dry screening.

4.5.2.3. Shaking Screens:

It essentially consists of a shallow rectangular box where the length is at last 2- 4 times the width. It is open at one end and is fitted a screen bottom.

It is shaken by means of a suitable mechanism. Speed, slope and length of the stroke should be adjusted to produce rapid stratification of the feed with a forward motion so that minimum blinding of the screen surface is resulted. It is widely used in case of screening of coal. It looks very similar to the vibrating screen.

4.5.2.4. Vibrating Screens:

Vibrating screens are recent development and have made most of the other screening practices obsolete. It is essentially a flat plane screening surface made from punched plates or wire woven which is secured rigidly on a steel frame. This frame is attached to certain mechanical device which imparts a reciprocating up and down motion to the screen in the direction either normal to the screen surface or at a high angle to the screen surface. These screens can be driven electrically or mechanically. The particles passing through the screen is the under flow and particles retained on it are discharged as overflow continuously at the other end.

4.6. Multi Deck Vibrating Screens:

When only one screen is used in the vibrating setup it is called single deck vibrating screen. But similar to compound trommel, multiple numbers of screens can be used in the set up. Then it will be called a multideck vibrating screen. In case of multideck vibrating screen a number of screens are used one over the other, fixed rigidly to the main frame. The coarsest screen is at the upper most position and the finest screen is at the bottom most position. So by using this technique we get number of over sized material fractions on each screen. Sometimes the vibrating screens are placed in an inclined fashion so as to facilitate automatic discharge utilizing the natural force of gravity.

4.7. Advantages of Multideck Screens:

I. It requires minimum floor space.

2. It operates continuously.

3. The problem of screen blinding in this screen is less.

4. The screen surface can be repaired easily compared to trommels.

4.8. Disadvantage of Multideck Screens:

1. There is heavy wear of screen cloth or material in vibratory screens.

4.9. Comparison between Shaking & Vibrating Screens:

1. Shaking screens have number of advantages over most of the vibrating screens in terms of cost of operation& installation.
2. Shaking screens can be set almost flat during operation.
3. But they are more prone to heavy wear and require more frequent and expensive repairs compared to vibrating screens.

4.10. Operating Characteristics of Screens:

The operating characteristics of any industrial screen are:

a. Capacity.

b. Efficiency or performance.

c. Operating cost.

4.10.1. Capacity:

Capacity of the screen depends upon:

1. The area of the screening surface.
2. The size of the opening.
3. Characteristics of the ore such as specific gravity, moisture contents, temperature, proportion of fines particularly slime or clay in the product.
4. Type of screening mechanism used.

Capacity and efficiency are interrelated upto a particular extent. If the capacity is to be large, the efficiency has to be low. If the efficiency is to be improved capacity has to be sacrificed. Because of the direct dependence of screening capacity upon the area of screening surface and upon the screen aperture, it customary to express the capacity in the term of tons per square foot per millimeter screen aperture per 24 hours.

A comparison is made regarding capacities of various industrial screens in the table 4.1.

Table 4.1. Capacity Comparison of Various Industrial Screens:

Type of Screen	**Capacity Range** (Ton/sq.foot area/millimeter aperture/24 hr.)
Grizzly	1-5
Trommel	0.3-2
Shaking	2-8
Vibrating	5-20

4.10.2. Performance or Efficiency of Screens:

It is difficult to quantify the screen efficiency. According to mechanical engineering efficiency is defined as the ratio of energy output to the energy input during execution of a particular work. But in case of screens the efficiency that is measured is not the mechanical efficiency in exact sense. Screen efficiency defined here is a *measure of effectiveness of the screening operation as compared to a perfect screening operation.*

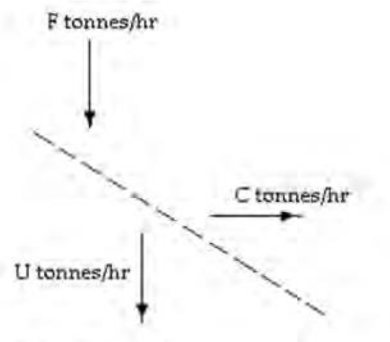

Fig.4.4. Mass balance on a Screen

Let us imagine a screen receives a feed of F and produces C &U as overflow and underflow respectively. Referring to the figure 4.4 the percentage efficiency of a screen is expressed mathematically as follow:

$$F = C + U$$

$Ff = Cc + Uu$, where, *f, c & u* are the fractions of oversized material in the *feed, overflow and underflow* respectively. They are determined by sieving a representative sample from each fraction in the laboratory. Applying the principle of mass balance we have:

$$F(1 - f) = C(1 - c) + U(1 - u)$$

Or, $\frac{C}{F} = \frac{f - u}{c - u}$ & $\frac{U}{F} = \frac{c - f}{c - u}$

Hence, the recoveries of oversized material into screen overflow:

$$\frac{Cc}{Ff} = \frac{c(f - u)}{f(c - u)} \text{ – – – – – (1)}$$

Similarly, the recovery of undersize material into screen underflow:

$$\frac{U(1 - u)}{F(1 - f)} = \frac{(1 - u)(c - f)}{(1 - f)(c - u)} \text{ – – – –(2)}$$

Hence the overall efficiency of the screen in classification:

$$\frac{c(f - u)}{f(c - u)} \times \frac{(1 - u)(c - f)}{(1 - f)(c - u)} = \frac{c(f - u)(1 - u)(c - f)}{f(c - u)^2(1 - f)} \text{ – – – – – (3)}$$

Assuming that there is no oversized material in the underflow, $u = 0$ we have efficiency,

$$E = \frac{(c - f)}{c(1 - f)} \text{ – – – – – (4)} .$$

This equation is widely used to calculate the efficiency of the screen and implies that recovery of coarse material in the overflow is 100%. Another equation which is also used to calculate the efficiency is: **E =**$10{,}000U / uF$, where, U is the tonnage passing through the screen for each F tonnes of feed u is the percentage of undersize material in the feed as obtained from laboratory screen analysis.

4.11. Operating Cost of Screens:

The operating cost of screens is small. For stationary screens, power cost is nil but there are other costs like attendant, replacement and repair. In addition to these costs moving screens have a cost for the power consumed during their operation.

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CHAPTER- 7

HEAVY MEDIA SEPARATION

Introduction:

If a fluid is available whose specific gravity is intermediate between two solids which are to be to separated, then one of the simplest process will be to suspend the mixed mass in that fluid.

As per law of buoyancy, one of the solids will float at the top of fluid level while the other one will sink to the bottom of the vessel.

Then a mechanical arrangement will be required to drawout different products from the top and bottom of the vessel.

A typical example can be the separation of wood chips from gravel or sand using water medium.

Laboratory Grade Heavy Fluids:

Most of the heavy organic liquids are used as heavy fluids and can be used only on a laboratory scale to assess the optimum separation obtainable by gravity concentration.

One of the most useful heavy fluids is acetylene tetra bromide whose specific gravity is **2.96.** This fluid can be diluted with carbon tetra chloride with sp.8 of 1.59 to yield a series of fluids with a sp.g varying from **2.96 to 1.59**.

Another group of useful fluids of low specific gravity is the aqueous solution of zinc chloride (**$ZnCl_2$**) and calcium chloride (**$CaCl_2$**).

High cost of laboratory heavy fluids precludes their employment in industrial applications.

Principle of Heavy Media Separation:

The basic principle involved in the gravity concentration process is the ‘Float and Sink’.

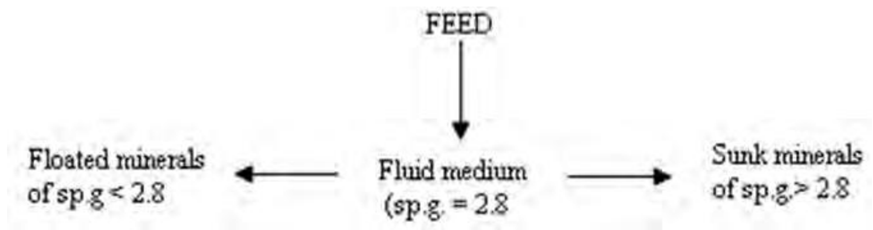
This is carried out by using a fluid whose specific gravity is in between the specific gravities of the two mixed up minerals particles in the crushed ore.

Since most of the minerals are heavier than water, water is not a suitable fluid medium for practicing ‘float and sink’ method of separation.

For this process to be effective fluids heavier than water are required.

The figure 6.1 explains the basic principle involved in *HMS.*

Fig.6.1. Scheme of heavy Media Separation.



6.1. Industrial Grade Heavy Fluids:

For industrial application pseudo liquids can be prepared by suspending solids in water.

These fluids can be used almost like true liquids provided the particles to be separated are coarser compared to the size of particles used to prepare the medium.

This medium is continuously agitated to prevent settling of particles used to form the pseudo fluid but the agitation allows the settling of heavy particles in the crushed ore to be separated.

Finely divided quartz, magnetite, galena or ferrosilicon is used for making up the suspension. The range of specific gravity for fluids of commercial interest is 1.3-2. Such fluids are mainly used to separate coal from clay. Pseudo fluids are much cheaper than organic liquids of high specific gravity, so the cost of fluid loss is not significant. But on the other hand, the use of pseudo fluids is not as simple as that of true fluids.

6.2. Heavy Media Separation Circuit:

A simple heavy media separation circuit would essentially consist of the followings:

i. A separating vessel in which heavy suspension is kept with a provision for introducing the feed and withdrawing the product continuously.

ii. Means to clean the product separated, recover the media and recirculate it to the vessel for further utilization.

6.7. Specific Industrial Processes Using Heavy Liquids:

Three different processes have been developed until now using true heavy liquids. The processes are:

1. Lessing Process.
2. Bertrand Process.
3. Du Pont Process.

6.7.1. Lessing Process:

Lessing process is used to clean coal in a solution of calcium chloride having an approximate specific gravity of 1.4. It is most useful in separating coal from clay & slate.

6.7.1.1. Lessing's Settling Tank:

Settling takes place in a cylindrical tank of **30 ft** height & **6-10 ft**. diameter with a conical bottom as shown schematically in the figure6.2. Graded raw coal freed from dust and fines is introduced into the tank through a central pipe to mix up with the separating solution thoroughly. As per "float & sink" principle cleaned coal floats up and is removed from the tank by a chain scrapper or any such mechanical arrangement. The slate, shale and sand drop to the 'conical bottom and are removed by the help of a bucket conveyor. Both cleaned coal and slate are delivered to the draining towers.

After draining, they are washed clean of the $CaCl_2$ solution. The wash liquor is returned to the concentration tank for recalculation $CaCl_2$ solution to the settling tank.

320 liters of $CaCI_2$ 1iquor is withdrawn from the separating tank after each ton of raw coal cleaned. During cleaning of coal the specific gravity of the parting solution drops to 1.2 from 1.4 due to addition of wash water and inherent moisture in the coal. **320** liters of parting liquid withdrawn from the tank is made-up to **640** titers and concentrated to a volume to yield $CaCl_2$

solution of specific gravity 1.4. Subsequently the solution is recirculated to the separating tank for further cleaning of coal.

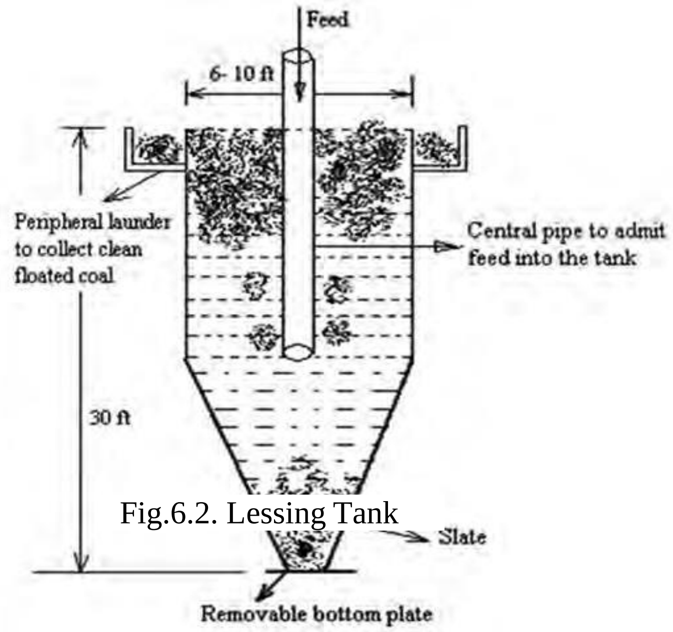


Fig.6.2. Lessing Tank

6.7.1.2.

Process Characteristics:

1. The loss of calcium chloride solution during washing of coal is in the order of 2-3 liters per ton of raw coal cleaned.
2. The process produces extremely clean coal.
3. Because the process constitutes a costly thermal concentration process, widespread adoption of this process has been restricted.

6.7.2. Bertrand Process:

Bertrand process also uses calcium chloride solution as separating medium and is applicable only to deslimed coal. The process is mainly utilized for washing of coal of 1-5mm size. This process is different to Lessing process with respect to feeding method. Here the feed material is charged into the system in a counter current fashion starting from water to separating solution.

Purified coal & waste are being withdrawn in a similar counter current fashion. There are five (5) circulating liquors such as hot water, weak solution, medium solution, strong solution & separating solution as shown schematically in the figure 6.2.

6.7.2.1. Characteristics of the process:

1. This process avoids costly thermal concentration of dilute solution.

2. This process introduces relatively complex hydro-metallurgical flow sheet compared to Lessing process.

3. The results obtained by the above two process are excellent and coal of extremely high grade coal is obtained.

Coal of such purity is utilized in manufacturing special carbon electrodes & hydrogenation.

6.7.3. Du-Pont Process:

Du Pont process is the practical adoption of laboratory heavy - liquid separation. This doesn't differ from laboratory procedure in basic principles, but requires some special treatments to be commercially viable.

6.7.3. 1. Special Requirements of Du Pont Process:

1. Parting liquid or the separating solution should have low solubility of the in water and water in parting liquid.
2. Parting fluid should have low viscosity or high fluidity at the operating temperatures.
3. Parting fluid should have high stability, low vapour pressure.
4. Parting fluid should be nonflammable.
5. Prior preparation of the ore is required for removing fines before parting.
6. Prior preparation of the ore with suitable chemicals is required to make the surfaces of the particles immune to wetting by the parting liquid.
7. Complete sealing of the separating system to prevent loss of parting fluid by evaporation and further to eliminate health hazards due to the noxious vapours emanating from the parting liquid.
8. Procedure should be available for complete separation of parting liquid from the minerals so as to regenerate the parting liquid.
9. The process should use of a scheme to purify the parting liquid constantly.

The requirement listed at No.6 is the most important among all conditions. During cleaning of coal, active agents like starch acetate or stannic acid of the order of 0.01l %wt. of the total weight is used. The process is shown schematically in the figure6.2.

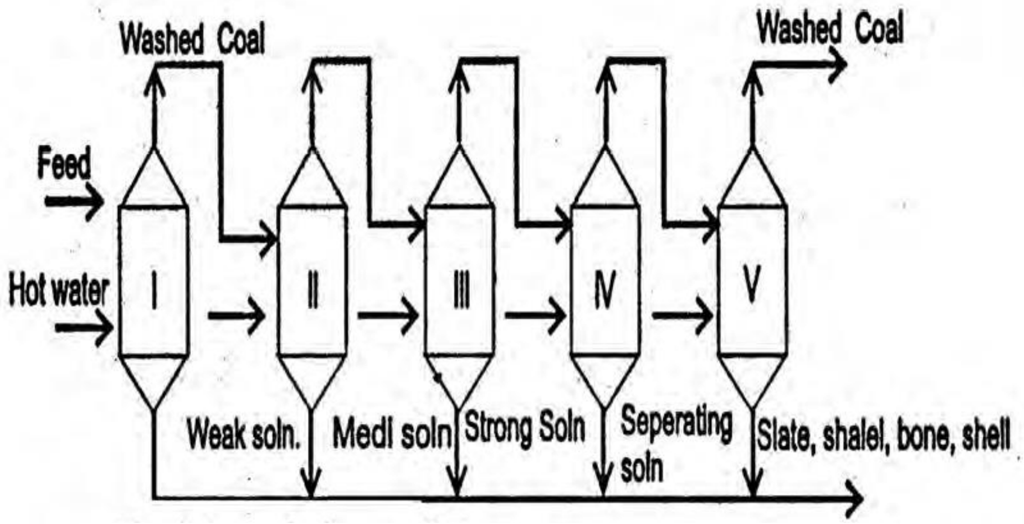


Fig.6.2. Schematic Flow Diagram of Du-Pont Process.

6.7.3. 2. Characteristics of Du-Pont Process:

1. Parting liquid in case of Du-Pont process is a mixture of several halogenated hydrocarbons.
2. The main expense in Du Pont process is the cost of parting liquid.
3. The consumption of parting liquid is in the order of 450grams per ton of coal cleaned.
4. Separation process is fairly simple in principle but requires a number of adjunct operations for the sake of economy in reagents consumption.
5. The process is not applicable to fine particles and is limited to coarse state of sub-division.

6.8. Industrial Processes Using Heavy Suspensions Or Pseudo Fluids:

Pseudo heavy fluids are manufactured by suspending quartz, ferrosilicon or galena in different proportions to have the requisite specific gravity. The processes are:

1. Chance process
2. Vooys process
3. Wuensch process

6.8.1. Chance Process:

Chance Process is in use for last 100 years for cleaning coal. The parting fluid is a suspension of quartz or sand particles in water. The sand used here is in the size range of -40 to +80 #. The *Chance Cleaner* consists of a separating tank or a *Cone Separator* in which sand suspension moves up gently. An agitator is used for stirring the suspension to prevent packing. The overflow of clean coal and sand passes over to the cleaning screens which desand and dewater the coal. Spray water is used for desanding. The specific gravity of the fluid is adjusted by varying the proportions of sand and water. For cleaning anthracite coal a heavier fluid is used than compared to the fluid used for cleaning bituminous coal. Figure 6.3 shows the Chance process schematically.

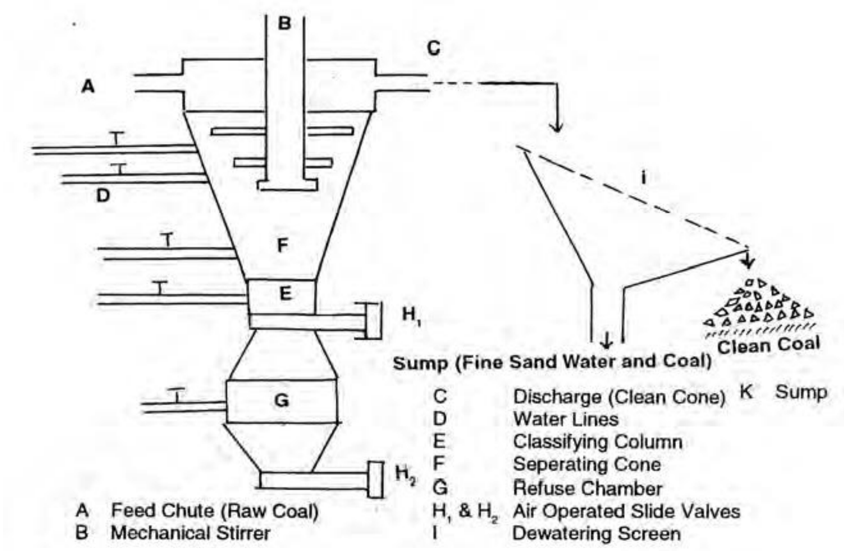


Fig.6.3. Schematic Chance Cleaner.

6.8.2. **Vooys process:**

This process uses a suspension of finely grounded barite (-150 to +200#) in water. Specific gravity is adjusted to 1.47 to clean coal. Coal particles finer than 100- mesh are excluded. Since the solid particles used to manufacture the parting fluid are much smaller than what is used in the chance process, the coal that can be treated by Vooys process can also be much finer.

6.8.3. **Wuensch process:**

This is a process for concentrating ores those contain lighter materials as waste (sp.g> 2.7). A mineral having specific gravity more than 5.25 must be used for making the suspension as suspensions containing more than 40% solid by weight are too plastic for partitioning work. It is preferable to use a suspension containing less than 30% solids by weight. It has been found out that galena is most suitable material to be used as it yields a heavy fluid with a specific gravity in range of 3 - 4.5 very easily at low concentration of solid. Since galena is relatively valuable the loss of medium must be reduced to a minimum level. The medium is purified periodically by flotation of galena.sometimes ferrosilicon in water is also used as heavy fluid.

6.9. Washing of Coal:

The major industrial application of heavy media separation is the washing of coal where raw coal is washed to prepare cleaner coal for further industrial application. Coal is always associated with mineral matter either intrinsic or extrinsic in nature. These mineral materials are known as impurities and generate out as *ash* on burning of coal. As mineral materials serve no purpose, it is therefore desirable to remove them from coal before distribution. The extrinsic impurities can be removed completely while intrinsic impurities are difficult to remove. Depending on the local condition the upper limit of ash content in the coal is fixed for its acceptance. Indian coals are high in ash. Therefore in India even metallurgical coke producers accept as high as 25% ash in the coking coal. In this connection it must be remembered that an increase in ash content of the coke adversely affects the blast furnace operation and reduces the pig iron productivity. Similarly the productivity of sponge iron is also hampered in the rotary kiln. An increase of 1% ash in the coke reduces the pig iron productivity by 5%. Therefore upgrading of inferior grade coals has become a necessity as better grade coals are being exhausted now.

6.9.1. Advantages of coal washing:

The advantages of cleaning coal are as follows:

1. Transport and handling charges are reduced.
2. The efficiency of coal utilization increases.
3. The calorific value of the coal increases.
4. Sulphur & phosphorus content in the coal is reduced which improves the quality of coal.
5. Greater cleanliness and less ash to be handled during industrial activities.

6.9.2. Principle of Coal Washing:

The specific gravity of pure coal varies from1.2-1.7 and that of free impurities from 1.7- 4.9. If the average specific gravity of pure coal is 1.3 and the same is suspended in a heavy fluid of specific gravity 1.5 (called washing medium), the impurities being heavier than the fluid sink in the fluid and pure coal floats on the fluid.

The washability characteristic of coal is best evaluated by the float and sink test.

6.9.3. Washability Characteristics of Coal:

A suitable range of specific gravity of the separating medium for most purpose is from 1.30 to 1.60 varying by an increment of 0.05. These gravity baths can be made from organic liquids or inorganic salt solutions.

The float and sink test starts at the lowest specific gravity bath with the float being removed and the sink materials are placed in the next higher specific gravity bath. The information desired is the weight of the float coal and the weight of the sink materials at each gravity bath. Further the float and sink materials are fire assayed to find 5carbon and % ash in them. The table 6.1 shows the result of a typical float and sink test.

Table.6.1.

Sp.gr of the bath	Float material(clean coal)		Sink material (dirt)	
	Carbon wt%	% of ash	Carbon wt%	% of ash
125	5	1	5	6
1.30	65	2.5	35	39.5
1.35	75	3.0	25	51.5
1.40	80	4.0	20	58.5
1.50	82	4.5	18	64.0
1.60	85	5.5	15	70.0

The observations listed in the table 6.1 are plotted as washability curves shown in the figure 6.4. Washability curves as applied to the coal washing are graphical representation of the results of specific gravity analysis for a sample of coal. Different

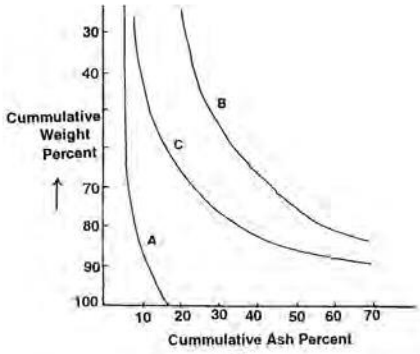


Fig.6.4. Washability curves of a typical coal.

curves shown in the plot are as follow:

1. *Curve A*:

Curve *A* represents the maximum yield of a float coal which may be obtained for any ash content or vice versa.

2. *Curve B:*

Curve *B* represents the ash content of the sink material corresponding to any given yield of the floated coal.

3. *Curve C:*

Curve *C* represents the average ash content of each specific gravity fractions.

From the above graphs it follows that, if a clean coal of 4.5% ash is required, the cut-off specific gravity of the separating medium is 1.5.This would correspond to a yield of 82% of clean coal having 4.5% ash in it.

6.9.4. Industrial Coal Washing:

In general gravity concentration method is used to clean coal after their size reduction in crushing rolls, hammer mills and etc. Chance method is the most widely used process of washing coal while Baum jig method is exclusively used for cleaning bituminous coal. The efficiency of a coal washing unit primarily dependents on the following factors such as:

1. The densimetric composition of the feed coal.
2. The total ash to be tolerated in the cleaned coal which is interlinked with the cut-off density of the gravity bath employed for washing.
3. The sharpness of separation to be attained in the cleaning or washing.

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CHAPTER-6

GRAVITY CONCENTRATION

Wilfley Table:

The constructional feature of a Wilfley table is shown schematically in the figure8.3. The table is made from wood or similar such material. The table surface is cleated specifically as shown in the figure.

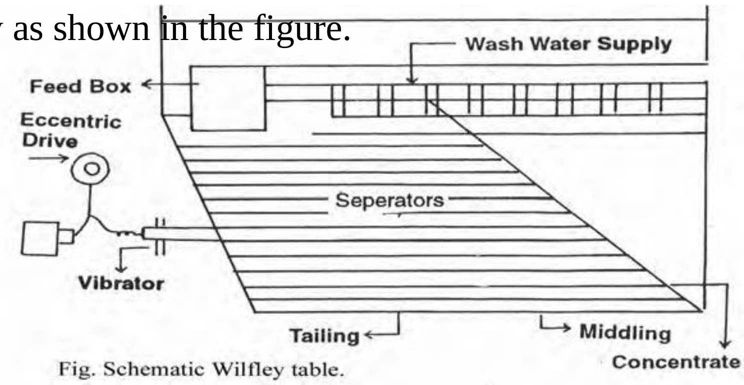
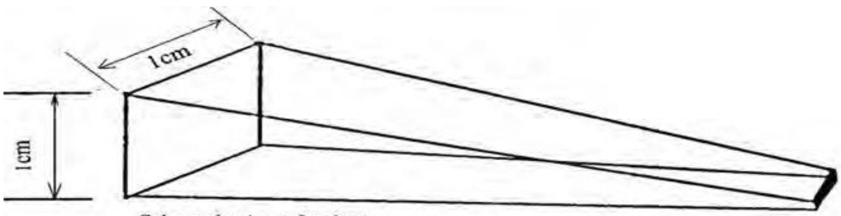


Fig. Schematic Wilfley table.

\Specific discussion on cleats or riffles is required as they require frequent replacement during the working of Wilfley table. The cleats are usually made up wood with a maximum height and width of *one centimeter* each as shown in the figure8.4.



8.3. Schematic view of a Riffle or Cleat.

The cleats are tapered from one end to another. They are so placed that they form channels of around 1cm width and deep at the left hand side end and the same tappers down to zero depth at the opposite end. The cleats end along a diagonal line imagined on the Wilfley table which approximately divides the total surface area of the table in the ratio of 2: I. This means 2/3 of the total surface area of the table is cleated (riffled) and rest 1/3 portion is unriffled. The inclination of the table is from left to right and from the back to front. Such inclination increases the ore handling capacity of the table.

The Jigs:-

A jig is essentially a water filled box in which a bed of mineral grains are supported on a perforated surface or screen. Jigs are usually made up of wood or other materials. In place of one compartment there may be several compartments connected in series.The tailing of one compartment works as feed for to the next consecutive compartment in the series. The amplitude of jigging is maximum in the first cell and minimum in the last cell.When water is pulsed through the screen, the particles are brought into suspension in water and are allowed to settle under hindered settling conditions which are modified greatly by differential acceleration (the theory of jigging has been discussed earlier).If the settling periods are of very short duration, the separation of two materials according to the specific gravities may be possible almost regardless of the size. This explains how the jig can handle wide range of size distribution. It is evident that with a feed of a wide size range, a very short settling time must be used for complete stratification.

Basic Construction of a Jig:
The major components of a Jig are:
A shallow open tank containing a screen-bottom on which the ore is supported.

1. A hydraulic water chamber or *hutch*.
2. A reciprocating mechanism for pulsating water through the sieve.

Classification of Jigs:

Jigs are classified to two types:

a. Hand jig.
b. Mechanical jig.

Hand jig:

This is the simplest of all jigs which consists of a framed sieve held by hands and is actuated by the operator with a reciprocating vertical motion. In general a perforated cylindrical shape container is used. After filling up the vessel with minerals up to the desired level it is closed tightly. With a rope and pulley arrangement it is made to move up and down in a water tank to attain the condition of pulsation and suction of water in the mineral bed. As the process is continued or repeated for several times complete stratification takes place. This jig is mainly used in the laboratory to demonstrate the effect of jigging operation. Figure7.8 shows the basic features of a hand jig.

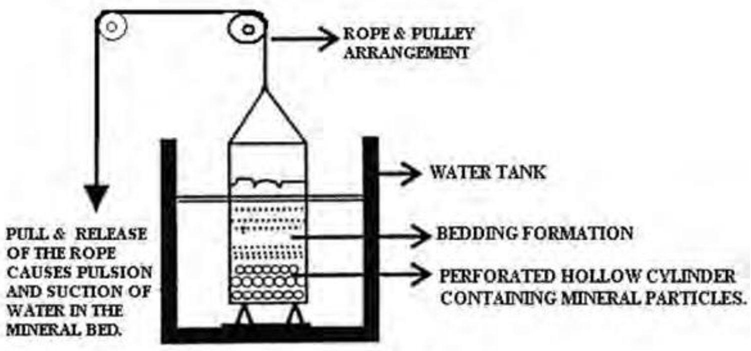


Fig. Laboratory Hand Jig.

Mechanical jigs:

Mechanical Jigs are of various types. But regardless of type they are essentially composed of:

i. A shallow open tank containing a screen-bottom on which ore is supported.
ii. A hydraulic water chamber or *hutch*.
iii. A reciprocating system for pulsation and suction of water through the screen.

Typical Mechanical Jig:

There are different mechanical jigs such as:

1. Fixed sieve plunger jig.
2. Fixed sieve Pulsator jig.
3. Pneumatic or Baum jig.

Working of few important jigs is discussed below.

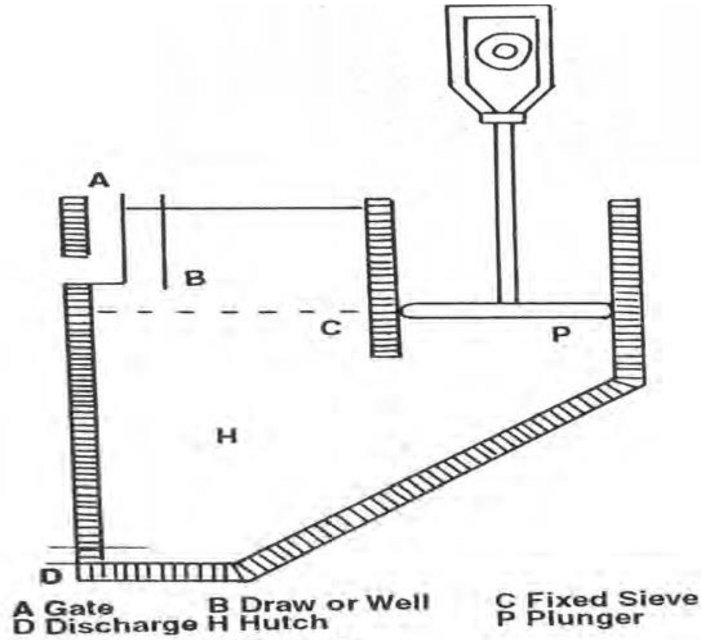
1. Fixed Sieve Plunger Jig (Harz Jig):

The harz jig has a fixed sieve. The jigging motion is obtained by plunger, *P* reciprocating in a compartment adjoining the sieve compartment, *C*. The bottom layer (usually the concentrate) is removed through the gate, *A*. The upper layer (usually tailings) is discharged at the end away from the feed.

Working:

The crushed & graded ore is held on the sieve, *C*. Water is held in the hutch, when the plunger is pushed down water rushes up and when the plunger is moved up, water rushes down through the mineral bed held on the screen.

When water moves up it imparts a pulsation and when water moves down it imparts suction to the mineral bed. So both pulsation and suction takes place alternatively resulting in jigging. Jigging duration ranges from 0.2 to 0.6 sec (100-300 cycles per minute).



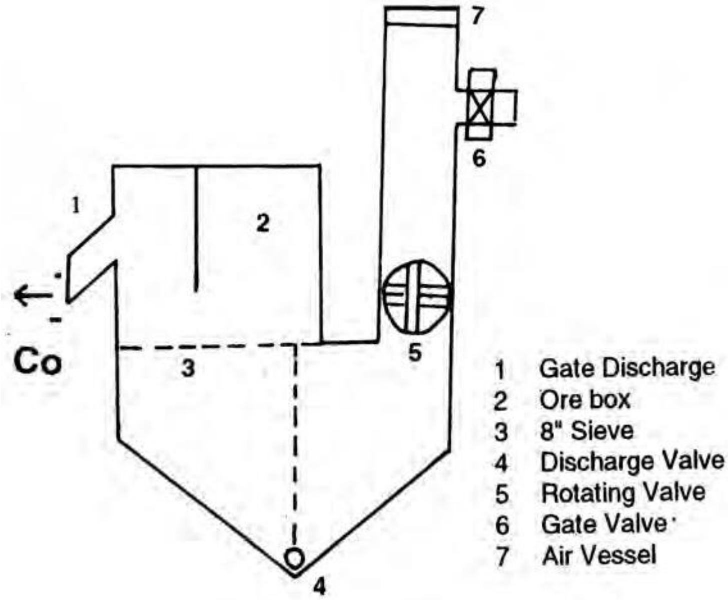
Harz jig

2. Plunger Jig: The plunger jig consists of ore box of size: 24''x8''x6'' fitted to one half of the tank and then plunger is fixed in the other half. The plunger is made to move up and down by mechanical arrangement. The bifurcation board between the jigging and plunger section at the centre extend sufficiently below the jigging sieve to ensure even arrival of water impulses at the sieve. Sieve plays an important role in jigging. Different types of jigs are used for different materials. Smaller materials use woven wire sieves, average sized material use punch plates while larger sized materials need barred grates

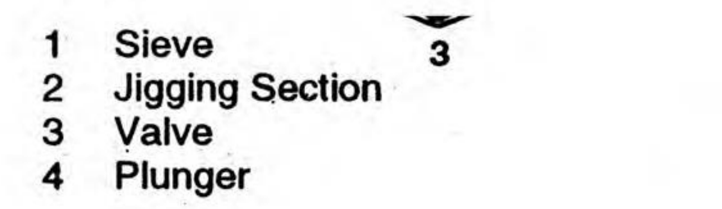
.

3. Pulsator jigs:

In this class of jigs there no suction stroke. The jigging is due to impulses of water flowing under pressure from the water service point. These impulses are obtained by placing a rotating device in the water service line. The number of impulses is around 200/minute. This type of jig can handle around 100tons/sq.foot/day.



Pulsator Jig



Plunger Jig

4. **Diaphragm Jigs:**

Bendelari diaphragm jig is the most popular diaphragm jig used worldwide. This jig is an improved version of Harz type jig. In this case a diaphragm is used in place of plunger to produce pulsion and suction of water in the ore bed held on the sieve. The mineral separation is rapid compared to Harz jig.

Constructional features of this jig are shown schematically in the figure7.12.

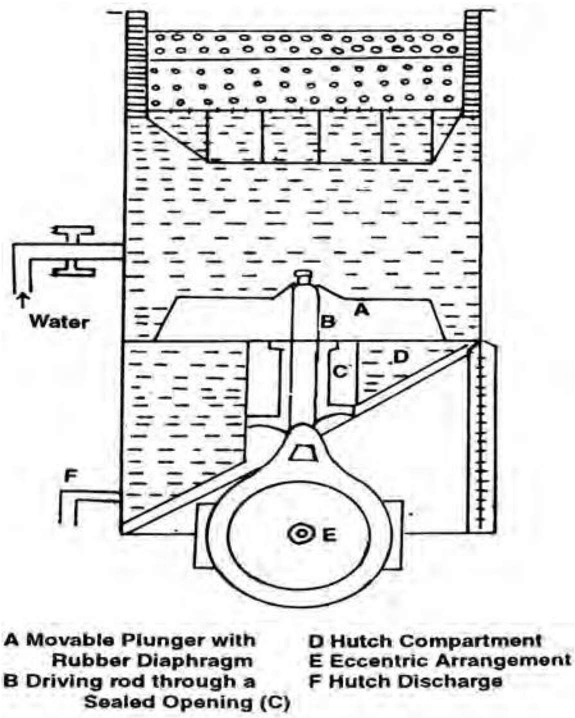


Fig. Bendelari Jig

In this case the plunger is sealed to the frame by a rubber diaphragm; hence there is no water leakage around the plunger which is a frequent problem in harz jig. Further the jigging surface is more accessible in this case as the actuating mechanism is placed at the bottom. This results in an appreciable saving in floor space and weight.

Compared to the Harz jig, the Bendelari jig has a more open bed, larger capacity consumes less water and requires less maintenance. The jigging cycles range from 0.2-0.8 seconds, i.e.100-160 strokes per minute.

5. Pneumatic or Baum Jigs:

Baum jig resembles the plunger jig in construction but differs in the working principle. With little modification it has been in use for the last 150 years. Presently it is extremely popular in coal washing.

In this case air under pressure is forced in& out of a large air chamber on one side of the jig vessel causing pulsion and suction to the jig water. This in turn causes pulsion and suction through the crushed coal bed held on the screen. Thus stratification is caused finally. Baum jig has the advantage of handling wide range of sizes with high capacity.

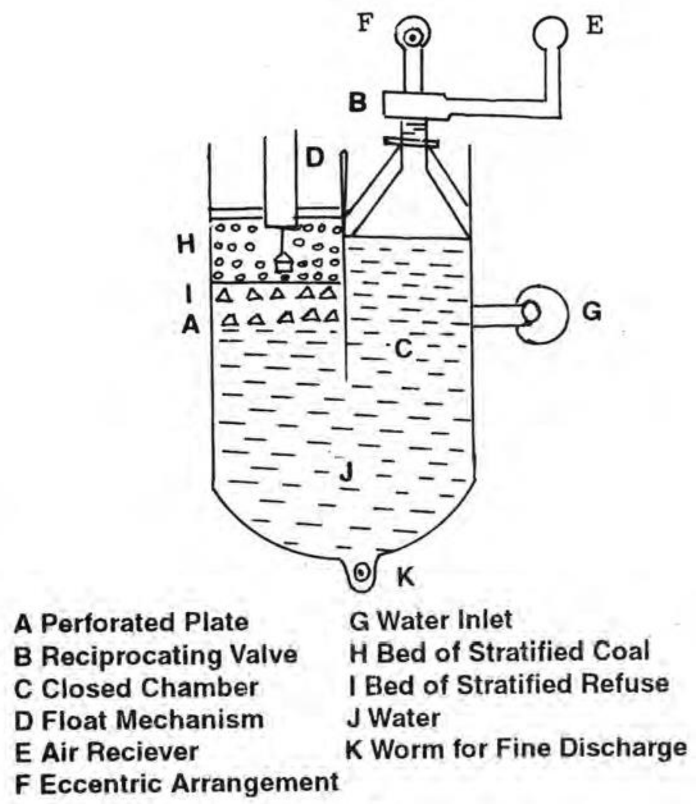


Fig. Baum Jig

7.6. Advantages of Jigs:

1. Jigs are primarily used to concentrate coarse-minerals. In coal washing, up to 4 - 5inches coal pieces can be washed in Jigs. In case of ores, pieces up to I inch size can be treated. Hydraulic jigs can wash coal up to 1/8 inch & minerals as fine as 20#. Pneumatic jigs can treat minerals as fine 65# mesh and as coarse as 1-1.5 inches but not in a wider size range.
2. Excluding washing of coal it is used widely to beneficiate non magnetic iron ores.
3. Jigs are cheap to operate and substantially foolproof and offers an easy access for inspection.
4.

7.7. Limitations of Jigs:

1. Jigs are obsolete for sulphide ores.
2. It requires large amount of water during ore beneficiation.
3. Fines cannot be treated in jigs. Jigging is applicable to the ore that is too coarse for complete liberation.
4. Jigs do not provide a complete solution to any mineral beneficiation problem.

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CHAPTER-8

FLOTATION

INTRODUCTION

Flotation is the most widely used method of wet concentration of ores for separating the valuable constituent of the ore from the worthless gangue. The process is primarily a surface phenomena based on the adhesion of some mineral particles to air and simultaneous adhesion of other particles to water in the pulp. It is the most efficient but is the most complex of all ore beneficiation processes. A In this process adhesion is made between air bubbles and small mineral particles in such a way that they rise in that pulp. The floating mineralized froth is then skimmed off while the other minerals are retained in the pulp. The above fact is known as *flotation proper*. There is another term called *skin flotation*. In such a case the adhesion is affected between a free water surface and the mineral particles. The particles involved in skin flotation are usually larger than the particles involved in froth flotation. To obtain adherence of the desired mineral particles to the air bubbles, a hydrophobic surface film should be formed on the particle surface. Hydrophilic surface film must be created on the particles which are to be retained in the pulp phase. The most striking outcome of this process is that the specific gravity of the mineral particle has no effect on the flotation. This suggests that minerals irrespective of their specific gravities can be floated.

Another important idea in case of flotation process is the existence of a selective tendency on the part of some mineral particles to adhere to air and others to water. Much research has been done on this most recent and complex means of ore beneficiation which are summarized as follow:

1. Most minerals if suitably protected from contamination adhere to water but not to air.
2. Paraffin & other hydrocarbons adhere to air in preference to water.

3.But for minerals to be separated by froth flotation, floatability is to be induced on the surface. This is known as *acquired floatability*. For the minerals to acquire floatability suitable chemical reagents are to be added to the pulp for changing their surface properties. The reagents vary in nature depending on the type of ore to be floated. The quantities to be used are extremely small but just sufficient to develop a continuous film around the mineral particles of at least few molecular level thicknesses.

4.Almost all the minerals can be made to adhere to air or water selectively by using suitable chemical reagents. But this selectivity can not be 100% efficient. This means when we are trying to float a particular mineral selectively, other mineral present in that pulp would also float up.

5.Change in the surface condition of the minerals (due to oxidation) will affect the floatability of such minerals considerably.

In general flotation depends on a number of interrelated physico-chemical factors. After treatment with reagents, the air bubbles attach it to the mineral particles and lift them up to the surface of water. The mineral is usually transferred to the froth leaving behind the gangue in the pulp. This is termed as *direct flotation*. However, during

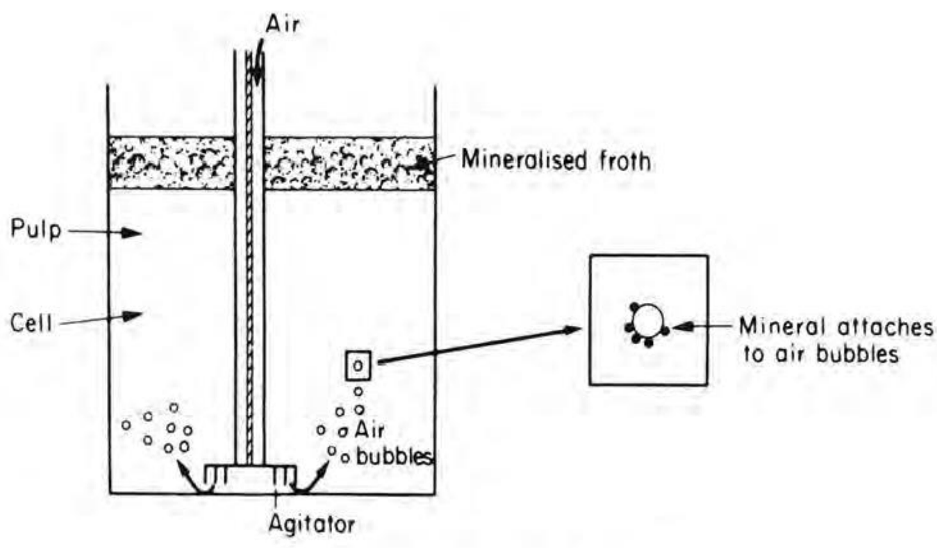


Fig.9.1. Basic Principles of Flotation.

reverse flotation the gangue is separated into the float fraction while the valuable mineral is retained in the pulp.

The process can only be applied to relatively fine particles. The basic idea of flotation is shown schematically in the figure 9.1.

9.1. Classification of Floatability:

Floatability can be classified as:

Natural floatability and

Acquired floatability.

9.1.1. Natural floatability:

It is generally agreed that hydrocarbons, coal, graphite, sulphur shows large degree of natural floatability. It is to be observed that substances showing natural floatability are non polar substances. So minerals that are polar in nature lack in natural floatability.

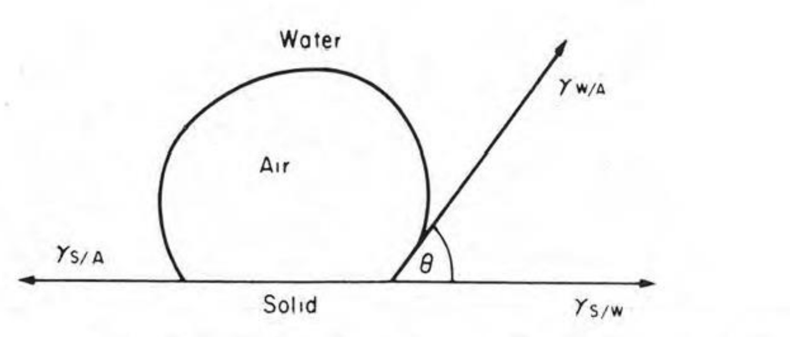
9.1.2. Acquired floatability:

By suitably coating the surfaces of one or another a group of minerals with a film that is non-polar, particles of the selected group can be made to act as if they are non polar throughout and made to acquire floatability. Acquired floatability is the result of the actions of a group of reagents called collecting agents or collectors. When the ground ore is mixture of several of minerals of similar nature, to separate them from each other, some minerals should be made more floatable compared to others. To acquire such selectivity specific reagents are to be added to the pulp and are termed as *activators or depressors*. Another group of reagents added to the pulp are known as *modifiers*. *Modifiers* are chemical reagents which suitably modify the surface properties of the minerals so that the surface becomes more amenable to the action of collectors.

9.2. Physico-Chemical Principles of Flotation:

Physico-chemical principles of flotation can be explained in terms of surface energy & surface tension, contact angle, polarity and adsorption.

9.2.1. **Surface Energy or Surface Tension and Contact Angle:** At any interface there exists certain amount of energy called surface energy. The surface forces at the bubble-mineral interface in an aqueous medium are shown schematically in figure 9.2.



Contact angle between bubble and particle in an aqueous medium.

From the figure it understood that at equilibrium,

$yS{-}a = y_{c-w} + y_{w-a}cos8$

Where, y_{S-a} , y_{c-w} and y_{w-a} are the surface energies between the solid-air, solid-water and water-air respectively and

8 is the contact angle between mineral and the bubble as shown in the figure9.2. Now work of adhesion:

$W_{c-a} = y_{w-a}(1 - cos8)$.

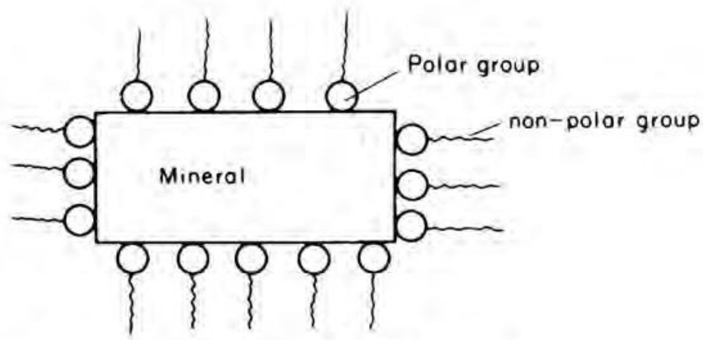
It can be seen that greater the contact angle 8 , greater is the work of adhesion between particle and the bubble. The floatability of a mineral therefore increases with the increase in contact angle. Minerals with higher contact angle are said to be aerophilic (air attracting) and minerals with smaller contact angle are said to be aerophobic (air repelling).

9.2.2. Polarity and Adsorption:

All the minerals are classified into polar and non-polar type according to their surface characteristics. Non-polar surfaces do not attach readily to the water phase and are called hydrophobic minerals. Graphite, coal, talc and sulphur are nonpolar minerals and exhibit natural floatability and readily float on water.

Minerals of polar type are hydrophilic and do not float naturally on water. These minerals have to acquire floatability to get floated up. To induce floatability these mineral particles are to be treated with some specific chemical reagents called *collectors*. *Collectors* are organic compounds which get *adsorbed* on the surface of selected mineral particles and produce a continuous heteropolar film in such a fashion that, the nonpolar part of the film is oriented away from the mineral body(as shown in the figure 9.3. Thereafter the mineral particle as a whole becomes non-polar, non-wettable and water repellant. Further it attaches itself preferentially with an air bubble.

2. The air bubble-mineral combination floats up in the fluid as per Archimedes' principle as long as the specific gravity of the combination is lower than the specific



Collector adsorption on mineral surface.

gravity of the fluid.

9.3. Flotation Reagents:

Froth flotation being a physico-chemical process requires a number of chemical reagents for its successful operation. Broadly the flotation reagents can be classified under following categories:

1. Frothers
2. Collectors &
3. Modifiers.

Operational Principles of Flotation:

The success of the flotation operation depends on the following factors:

a. Particle size,
b. Surface preparation of the minerals or conditioning,
c. Pulp density,
d. Temperature of operation,
e. Time duration of flotation.

1. Effect of particle size on froth flotation:

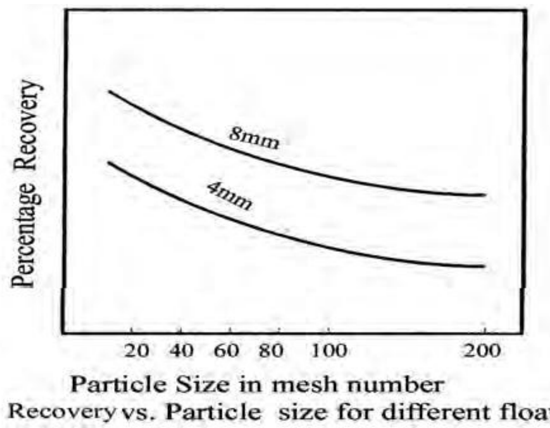
Particles of various sizes do not float equally. From experiments it has been found out that flotation is most efficient for particles in the size range of 20-200#. Recovery falls off distinctly in the very fine and coarse range of the feed. The failure to float coarse particles arise from:

1. Incomplete liberation.
2. Too small a contact angle.
3. Violent agitation required to form suspension.

The failures to float extremely fine particles are due to:

a. Poorer chance for mineral - bubble encounter in the fine size range of the mineral.

b. The finer particles have an older surface than coarse particles. As the surfaces of the particles is affected by ions derived from other minerals, oxygen and water during fine grinding, they become unresponsive to reagents and lose their capacity to float. The



Recovery vs. Particle size for different floating time.

percentage recovery versus particle size is shown schematically in the figure 9.7.

The tentative maximum size ranges of different minerals for efficient flotation are shown below:

Ore / Minerals	Maximum size range (in mesh)
Coal	10-14 #
Sulphides	4-65 #
Gold	100~150 #

Further rate of flotation is also depends on the particle size as shown

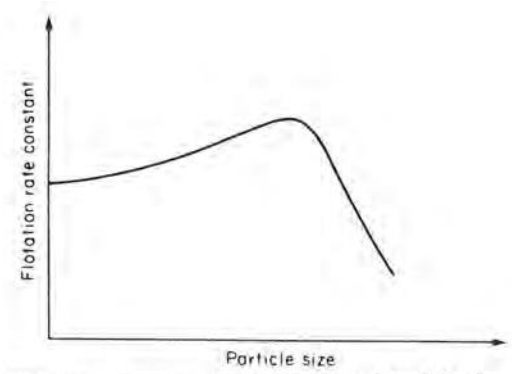


Fig.9.8. Rate of Flotation vs. Particle Size.

schematically in the figure 9.8.

2. Conditioning:

Conditioning is nothing but mixing of ore with water & aeration prior to flotation in a cell. Usually a big tank is used for this purpose. Improper conditioning will have adverse effect on flotation.

3. Pulp density. .

For the mineral and gangue particles to get separated during flotation the pulp should be dilute enough to permit particle rearrangement to take place freely. A pulp density of 35% solids by weight shows the best result. Over dilution should be avoided as it results in larger consumption of water and reagents.

4. Temperature:

For obtaining best result during flotation the pulp temperature is to be maintained between l2-20°C.

5. Time Duration of Flotation:

The time duration of flotation has a strong bearing on the extent of recovery and grade of the concentrate floated. As time duration increases, the extent of recovery increases with a fall in the grade of the concentrate as shown in the figure 9.9.

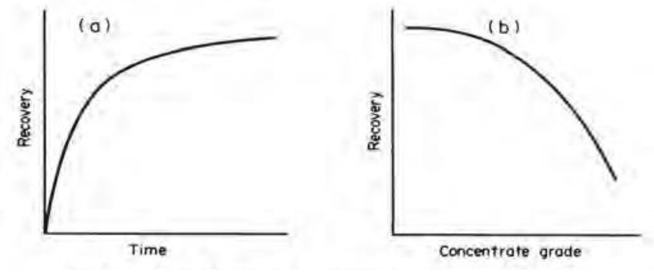


Fig.9.9. Time, Recovery and Concentrate Grade.

Flotation Machines:

Two important flotation machines are:

1. Pneumatic cell.
2. Mechanically agitated or Sub-aeration cell.

In the pneumatic flotation cells compressed air is directly blown into the pulp while in the sub-aeration cell a rotating impeller serves as a pump which draws in air through the hollow shaft of the impeller and distributes the same into the pulp to produce the froth. In the laboratory, usually a rotating, hollow impeller type sub-aeration cell is used which is shown schematically in the figure 9.10.

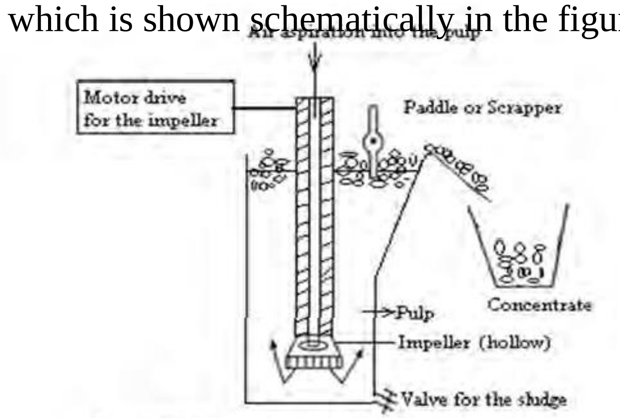


Fig.9.10. Laboratory model Sub-aeration Cell.

Industrial Model:

In industries hardly a single cell is used for practical floatation work. Rather a series of 10-15 cells connected in series are used-simultaneously. They are connected in such a fashion that one cell receives the defrothed pulp from the preceding cell as its feed. The recovery of such process is usually more than 90%. An industrial pneumatic cell is shown schematically in the figure 9.11.

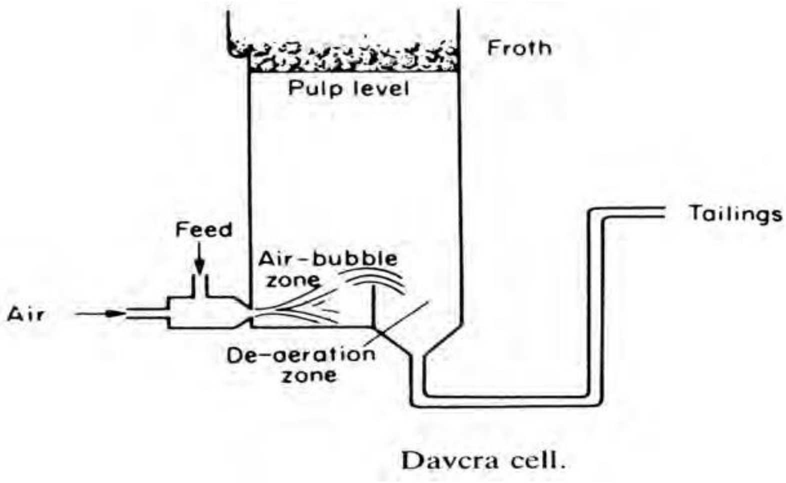


Fig.9.11. Davera Cell: Industrial Flotation Cell.

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CHAPTER -9

MAGNETIC & ELECTROSTATIC SEPARATION

Introduction:

It is a fact that various metallic minerals exhibit magnetic properties. They are attracted by the magnet exhibiting specific attractability. Basing on the degree of attractability minerals can be classified as:

a. Ferromagnetic
b. Paramagnetic
c. Diamagnetic

1. **Ferromagnetic Minerals:**

Few minerals such as magnetite and pyrrhotite are strongly attracted by magnets and behave as temporary magnets under the influence of magnetic fields. They are known as ferromagnetic minerals.

2. **Paramagnetic Minerals:**

These are the minerals which are weakly attracted by the magnets. Minerals in this group are illmenite, hematite, garnets etc.

3. **Diamagnetic Minerals:**

Minerals such as quartz, calcite and many others are practically non magnetic or may even be diamagnetic minerals. These minerals are repelled by a magnetic field along the lines of forces to a point where the magnetic field intensity is much smaller. The magnetic nature of the minerals or ores can be exploited in an industrial sense to separate them into three different groups such as:

l. Highly magnetic.
2. Weakly magnetic.
3. Nonmagnetic or diamagnetic.

This method of separating minerals is broadly termed as magnetic separation. Magnetic separation has found largest application in concentrating ferromagnetic minerals particularly magnetite ores with less than 50% Fe to 70% Fe.

It should be noted that subjecting the minerals to a magnetic field may result in magnetic concentration or separation. *Magnetic concentration* is the separation of valuable mineral from the gangue while *magnetic separation* is the separation of one mineral from another essentially based on the difference in the value of magnetic attractability of the minerals.

Elements in Designing Magnetic Separators:

The following facts are essential and to be considered during the designing of a magnetic separator:

1. Production of a suitably converging magnetic field.
2. Easy regulation of magnetic field intensity.
3. Even feeding of ore particle as a stream or ribbon.
4. Controlling the passage speed of ore particles through the magnetic feed.
5. Avoidance of nonmagnetic materials within magnetic field as occlusion.
6. Suitable means to dispose the products.
7. Provision for production of a middlings.
8. Elimination or reduction of moving parts to a minimum.

Types of Magnetic Separation:

Depending on the magnitude of magnetic flux density, magnetic separation can be classified as follows:

a. Low intensity magnetic separation.

b. High intensity magnetic separation.

A further subdivision within the group is possible depending on the medium in which separation is carried out. Depending on the medium of separation it classified as:

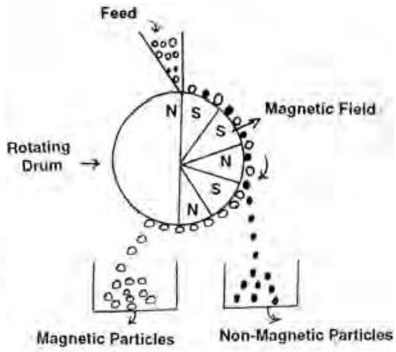
i. Dry magnetic separation.

ii. Wet magnetic separation.

Different Types of Magnetic Separators:

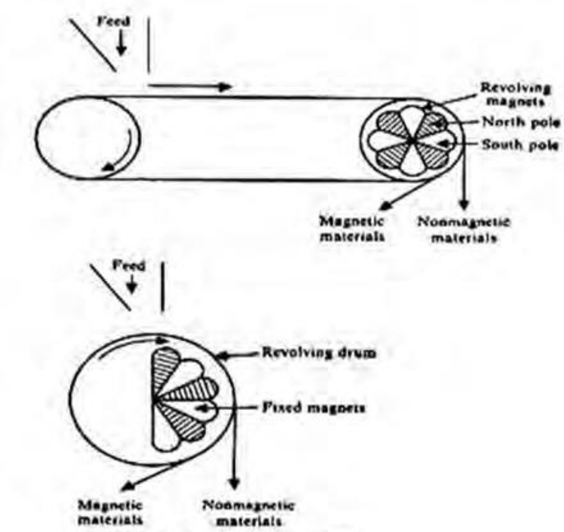
***a.* Low intensity dry magnetic separator:** This is type of separation is commonly applied to separate highly magnetic particles like magnetite, tramp iron from the non-metallics utilizing a low intensity magnetic flux.

When ore is travel on an endless conveyor belt passing over a magnetic pulley, the non magnetic particles follow a normal trajectory and are thrown clear but the magnetic particles are held firmly to the belt until it is carried out of the field and fall down when the belt just leaves the pulley. This phenomenon is shown schematically in the figure 10.1.



Drum Separator

2. Low Intensity Wet Magnetic Separator:



Dry magnetic separators.

This is widely used today for concentrating of low grade magnetite ore.

Wet type has the advantage of treating very fine ores almost in the slurry-form. Fines are more readily-separated and higher grade product is obtained because water causes a better dispersion of particles and, presents the feed to the separator efficiently.

The Ball-Norton drum separators consist of one or two rotating drums of nonmagnetic metals. In the drum(s), a number of fixed magnets are arranged in such a fashion that consecutive poles are of opposite nature. Much of the magnetic field passes directly from one pole to the other inside the drum, and thereby get wasted.

Applications of Magnetic Separation:

1. For removal of tramp iron in coarse and intermediate crushing circuits as a protection to the crushing machineries.
2. To concentrate magnetite ore.
3. To concentrate ores other than magnetite after converting iron ores to magnetite by magnetic roasting.

Electrostatic Separation:-

Electrostatic separation is a method of concentrating or separating minerals from each other on the basis of their differences in electrical conductivities.

The basic principle of electrostatic separation is the coulomb's law which implies like charges repel and unlike charges attract. It was first used to separate zinc ore from lead sulphide ore.

However, it was abandoned after introduction of froth flotation. But recently it has got a new lease of life for separating non- metallics.

Electrical concentration can be broadly classified into:

1. Electrostatic separation.

2.High tension separation.

Low and High Tension Electrostatic Separation:-

Similar to high tension magnetic separation, there is also a high tension electrostatic separation. During this separation the material grains are charged- up electrically due to ion bombardment on them along with the induction from the electrified drum. Ions are produced in the air gap between the electrically charged wire and the grounded electrified roll due to very high potential difference of few thousand volts maintained between them. The air around the wire becomes ionized and is attracted toward the grounded roll to discharge its ions.

Usually a potential difference of 30kV and above is applied to the wire electrode to make a corona discharge. The wire electrode is also known as corona electrode. If the voltage difference is sufficiently high the ionized corona is visible as a luminous discharge. On entering into the electric field the conducting mineral particles are bombarded with gaseous ions and get charged negatively and thus get deflected away from the ground roll. The non conducting particles are not deflected and have a free fall as it happens in case of usual electrostatic separator. The working principle of high tension separator is shown schematically in the figure10.5.

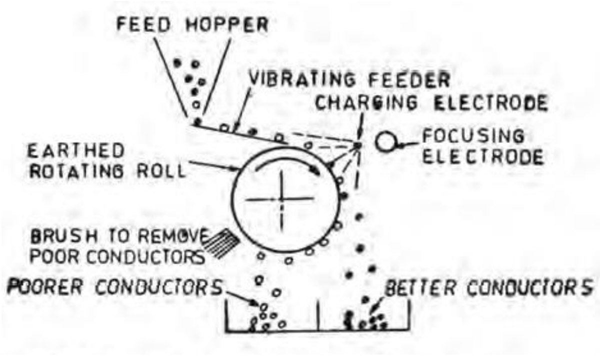


Fig.10.5. High Tension Electrostatic Separator.

The dry mineral grains are fed as a layer of one particle deep onto the electrified roll with the help of a vibrating and get separated as per the principle discussed earlier. High tension electrostatic separator is also known as Huff's separator.

Requirements for the Proper Working of an Electrostatic Separator:

1. For electrostatic separation, feed materials must be dried prior to separation.

2. For effective separation dry minerals grains are to be fed as a layer of one particle deep at the top of the rotating electrified roll. This is achieved by using a vibrating feeder.

3. For effective high tension separation, feed must be closely sized in the range of 1.0 - 0.1 mm free from fines. Quite often the feed material the feed material is to be heated above room temperature for effective separation.

Use

1. It is employed to separate conducting ores and minerals from non-conducting materials in ceramic industries.

2. This is applied for beneficiating rutile beach sands from non-conducting silica sand in rare earth plants.

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