

LECTURE NOTES
ON
IRON MAKING
SUBJECT CODE: PCMT 4307

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INDIRA GANDHI INSTITUTE OF TECHNOLOGY, SARANG
(An Autonomous Institute of Govt. of Odisha)

Iron Making

Subject Code PCMT 4307

Branch: Metallurgical and Materials Engineering

Semester: 5th Semester B.Tech

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PCMT4307	Iron Making	3-0-0	Credits 3
Objectives of the course			
1. This course introduces the principles and practices of iron making.			
Module-I		(12 hours)	
<u>Unit-1</u>			
Raw materials and their properties: Iron ores, Limestones, Agglomerates and Coke.			
Preparation of ores: sintering and palletizing, blast furnace burdening and distribution, testing of raw materials for blast furnace.			
<u>Unit-2</u>			
Numerical problem solving: Material balance, Feasibility of reactions and chemical kinetics.			
Module-II		(12 hours)	
<u>Unit-3</u>			
Design: Blast furnace profile, stove and gas cleaning units; instrumentation, refractory used in blast furnace and stove.			
<u>Unit-4</u>			
Reactions: Fe-C-O, Fe-O-H phase equilibria, Reactions in stack, bosh and hearth; formation of primary slag, bosh slag and hearth slag. Slag composition and its control, Metal-slag reactions, Control of hot metal composition.			
Module-III		(12 hours)	
<u>Unit-5</u>			
Process Control: Factors affecting fuel consumption and productivity, Recent developments in Blast furnace operations like, Bell-less top charging system, High top pressure, Humidified & Oxygen enriched blast and Auxiliary fuel injection through tuyers.			
<u>Unit-6</u>			
Irregularities in blast furnace operation and their remedies.			
Module-IV		(6 hours)	
<u>Unit-7</u>			
Alternative routes of iron making: Introduction, Processes of Sponge Iron production; SL/RN, MIDREX, HyL processes. Smelting Reduction Processes; COREX, ROMELT, Hismelt.			
Suggested Text books			
<ol style="list-style-type: none"> Ahindra Ghosh and Amit Chatterjee: Ironmaking and Steelmaking Theory and Practice, Prentice-Hall of India Private Limited, 2008. Dipak Mazumdar, A First Course in Iron and Steel Making, University Press-IIM-2015 			
Suggested reference books			
<ol style="list-style-type: none"> An introduction to modern steel making, R. H. Tupkary, Khanna Publishers (2000) An introduction to modern iron making, R. H. Tupkary, Khanna Publishers (2004) 			
Course Outcomes			
After completing this course, the student should be able to:			
<ol style="list-style-type: none"> Describe the physical and chemical processes that take place during iron making Analyse the effect of change in process parameters in iron making Describe the methods for control of quality in iron production Solve numerical problems involving reaction kinetics and composition control 			

LESSON PLAN**SUB: IRON MAKING****PCMT4307**

Lectures Topics

Module I

- 1 Introduction to Iron Making
- 2 Blast Furnace Plant
- 3 Raw materials and their properties: Iron ores and flux
- 4 Raw materials and their properties: Coke
- 5 Testing of raw materials for blast furnace.
- 6 Burden preparation: Agglomeration
- 7 Agglomeration of ores: sintering
- 8 Agglomeration of ores: Pelletizing
- 9 Blast furnace burdening and distribution
- 10 Design: Blast furnace profile
- 11 Blast furnace stove and gas cleaning units & instrumentation
- 12 Refractory used in blast furnace and stove
- 13 Blast furnace Operations
- 14 Material Balance

Module II

- 15 Reduction of Iron Ore
- 16 Fe-C-O, Fe-O-H phase equilibria
- 17 Zones in Blast Furnace, Reactions in stack
- 18 Reactions in bosh and hearth
- 19 Blast furnace slag: composition and its control, Metal-slag reactions
- 20 Process Control: Factors affecting fuel consumption and productivity
Recent developments in Blast furnace operations like,
Bell-less top charging system, High top pressure, Humidified &
- 21 Oxygen enriched blast and Auxiliary fuel injection through tuyers.

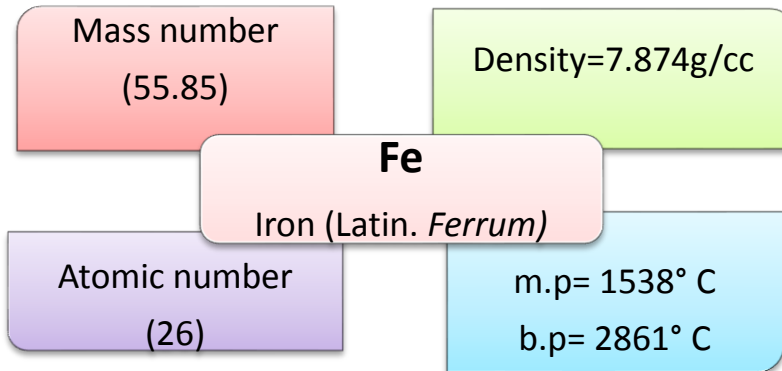
Module III

- 22 Irregularities in blast furnace operation and their remedies.
- 23 Alternative routes of iron making: Sponge Iron
- 24 SL/RN process
- 25 MIDREX process
- 26 HyL process
- 27 Smelting Reduction
- 28 COREX, ROMELT, Hismelt process

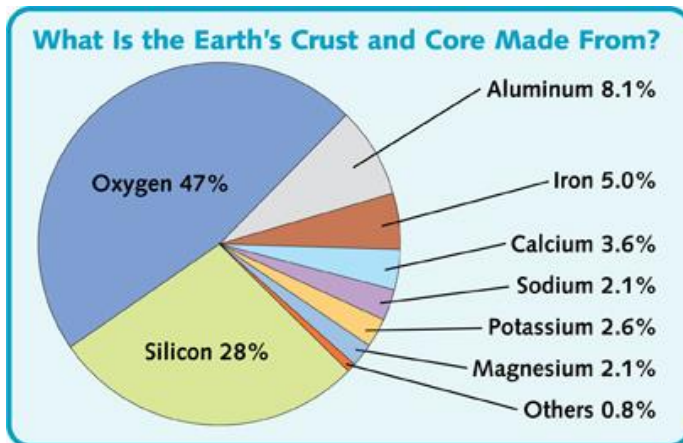
LECTURE-1

Introduction to Iron Making

Introduction



Iron is a chemical element with symbol **Fe** (from Latin: *ferrum*) and atomic number 26. It is a metal in the first transition series. It is by mass the most common element on Earth, forming much of Earth's outer and inner core. It is the fourth most common element in the Earth's crust after (*Oxygen, Silicon and Aluminium*).



The Iron making is necessarily a reduction process, as far as the chemical theory is concerned, where reduction of the iron ores takes place to produce metallic iron.

- Iron ores: *Generally in Oxides: Hematite (Fe_2O_3), Magnetite (Fe_3O_4)*

Theoretical % of Fe in Fe_2O_3

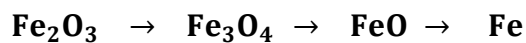
$$\begin{aligned} &= \frac{2 * 56}{2 * 56 + 3 * 16} \\ &= 70\% \text{ (roughly)} \end{aligned}$$

Theoretical % of Fe in Fe_3O_4

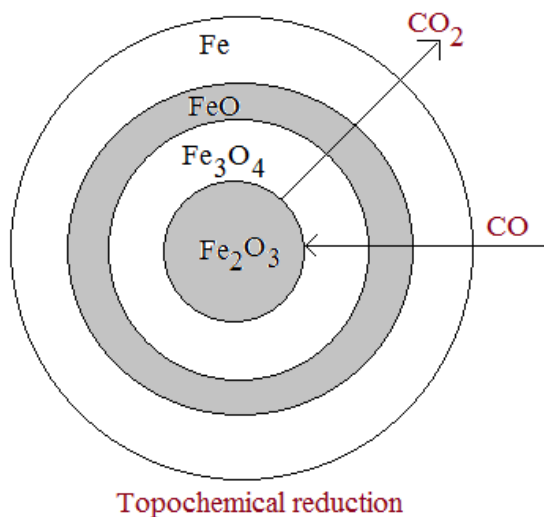
$$\begin{aligned} &= \frac{3 * 56}{3 * 56 + 4 * 16} \\ &= 72.4\% \text{ (roughly)} \end{aligned}$$

By Reduction conversion of Fe_2O_3 to Fe_3O_4 to FeO to Fe takes place. So Iron Making is necessarily a **REDUCTION** process. Out of a many reducing agents, like C, CO, H_2 , we select Carbon monoxide (CO). In case of Iron making, both the Direct and Indirect Reduction takes place. Where direct reduction needs Carbon, which is endothermic in nature, on the other hand, we have the Indirect Reduction, which is the exothermic reaction and utilizes CO in the reaction. For **direct reduction** of 1 kg. Fe, only 0.23 kg. C is consumed but results in absorption of 656 kcal of heat but 0.81 kg. C is required for **indirect reduction** of 1 kg. Fe from Fe_2O_3 and about 1790 kcal of heat is evolved in the process

Topo-Chemical reaction

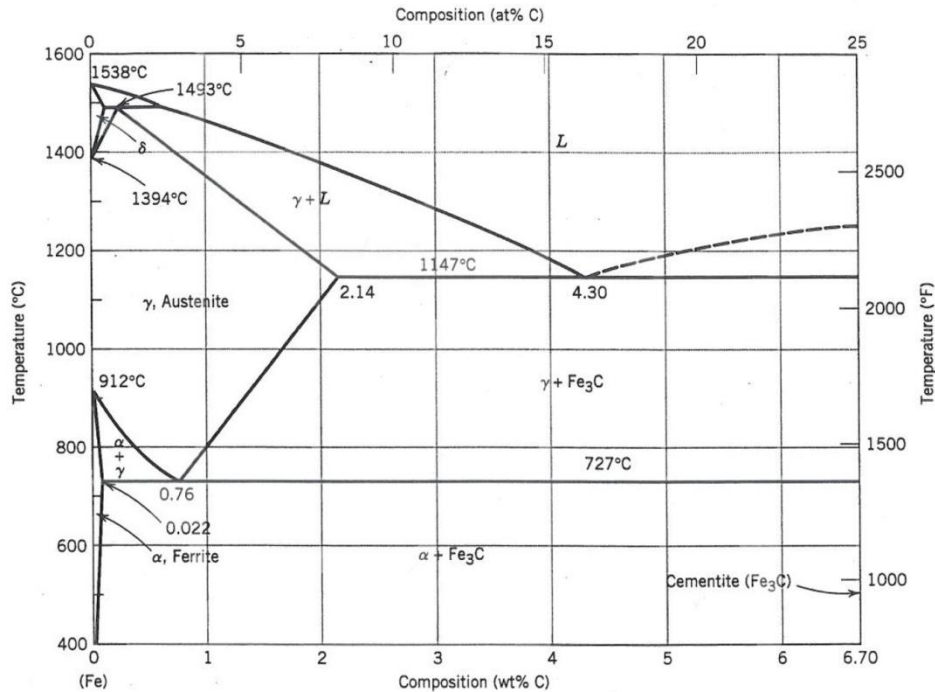


Step by step reduction of iron ore



Steel:

Steel is World's most useful and inexpensive alloy. It is an alloy of Iron & Carbon with/without other alloying elements. Iron making is the 1st step of an integrated steel plant.



Steel Cast Iron

PLAIN CARBON STEEL: here the carbon content is quite below 2.11% (in actual practice, it takes 1.6% value)

CAST IRON: Eutectic ferrous alloys (>2.11% C and eutectic reaction takes place in this alloy during cooling unlike steel)

Consumption Per Capita

- World- 150 Kg
- Developed World- 350 Kg
- INDIA – 50 Kg
- CHINA – 250 Kg
- The estimated India per capita consumption is expected to reach approximately 165 kg in 2019-20.

Different routes of Iron making

1. The Conventional route of Iron making

Raw materials → Blast furnace → Basic Oxygen Furnace

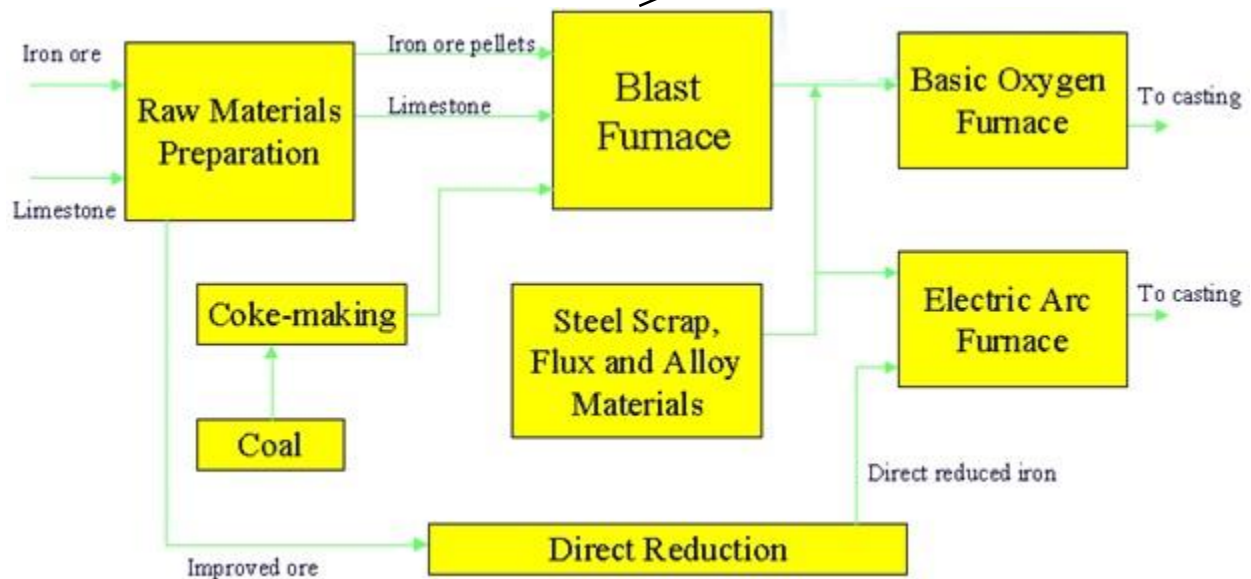
2. The Secondary route of Iron making

Scrap materials → Induction furnace/Electric arc Furnace

3. Alternative routes of Iron making

Raw materials → DR → Electric Arc Furnace

STEEL

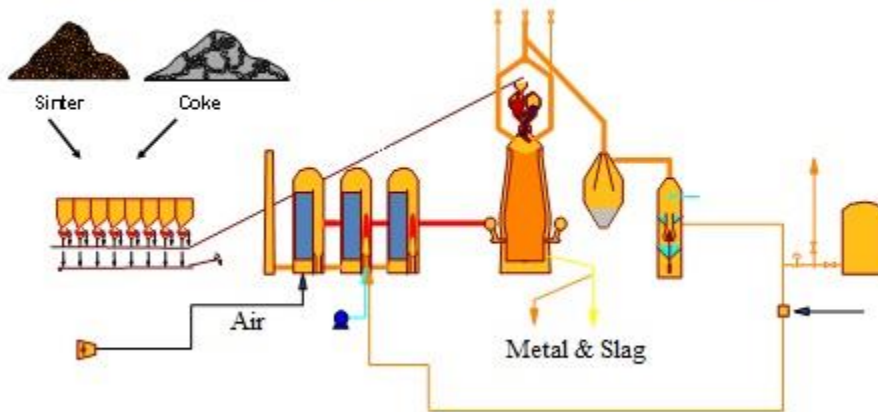


Blast F/c

- Iron can be extracted by the blast furnace because Oxygen of Iron Oxides can be displaced by carbon.
- This is more efficient method than electrolysis because it is more cost effective
- The B.F. works on a counter current principle
- Ascending hot gases meet Descending solid charge
- The charge includes Iron bearing materials (ore, sinter, pellets), coke & flux (Lime stone, Dolomite)
- The ascending gases cause reduction of Iron oxide in the Iron bearing materials while progressively heating it.

- The result is Production of
 - Liquid slag
 - Liquid Metal
 - B.F. Gas of considerable calorific value
- “Stukofen ” furnace was considered to be the progenitor of the modern blast furnace

Iron making In the Blast Furnace Plant



Raw materials for Blast Furnace

Iron Ore

Hematite Fe_2O_3	Limonite $Fe_2O_3 \cdot xH_2O$			
Magnetite Fe_3O_4	Siderite $FeCO_3$			
Fe content : 20 - 70%				
SiO_2	Al_2O_3	CaO	MgO	P, As
S - unwanted, less than 0,15%				

Flux

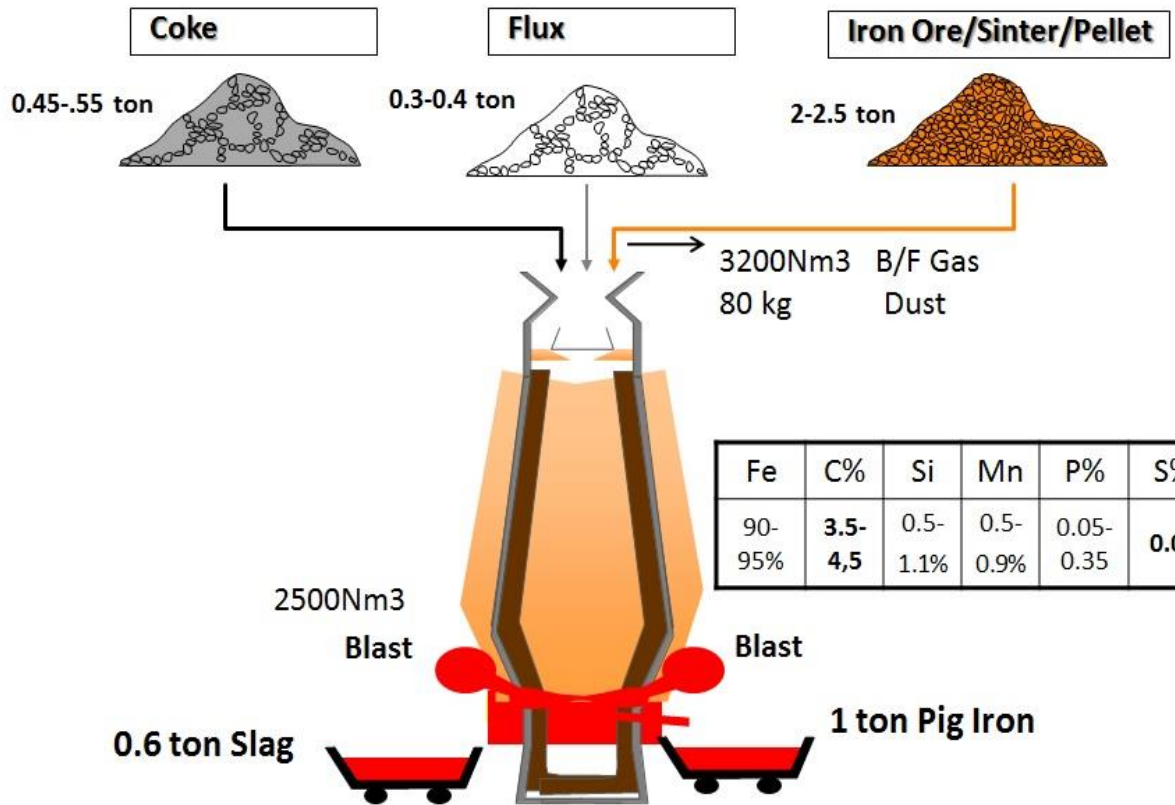
$CaO + SiO_2 = CaSiO_3$
 Flux + Gangue = Slag

Lime, dolomite $CaMgCO_3$, CaO

Coke

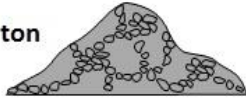
CO production
 Fuel
 Spacer
 Decrease MP

From Metallurgical coal



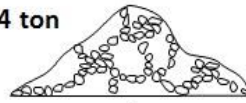
Coke

0.45-0.55 ton



Flux

0.3-0.4 ton



Iron Ore/Sinter/Pellet

2-2.5 ton



3200Nm3 B/F Gas
80 kg Dust

Fe	C%	Si	Mn	P%	S%
90-95%	3.5-4.5	0.5-1.1%	0.5-0.9%	0.05-0.35	0.03

2500Nm3

Blast

Blast

0.6 ton Slag

1 ton Pig Iron

LECTURE-2

Blast Furnace Plant

The blast furnace iron making typically implies a reduction and smelting in a single reactor. It is operated on counter current principle. The charge descent in the downward direction and the blast (hot air) ascends from tuyeres. Therefore the residence time of the reducing gas inside the furnace is less.

❖ **A modern blast furnace plant mainly consists of:-**

- *Blast furnace proper*
- *Hot blast supply equipments*
- *Gas cleaning system and gas storage*
- *Raw material storage and handling*
- *Liquid products disposal*
- *Process control equipments*

Blast Furnace Proper

The blast furnace is a tall shaft-type furnace with a vertical stack superimposed over a crucible-like hearth. Iron-bearing materials (iron ore, sinter, pellets etc.), coke and flux (limestone and dolomite) are charged into the top of the shaft. A blast of heated air and also, in most instances, a gaseous, liquid or powdered fuel are introduced through openings at the bottom of the shaft just above the hearth crucible. The heated air burns the injected fuel and much of the coke charged in from the top to produce the heat required by the process and to provide reducing gas (CO) that removes oxygen from the ore. The reduced iron melts and runs down to the bottom of the hearth. The flux combines with the impurities in the ore to produce a slag which also melts and accumulates on top of the liquid iron in the hearth. The iron and slag are drained and separated out due to their density difference through tap holes.

The structure essentially consists of the *following parts*:.

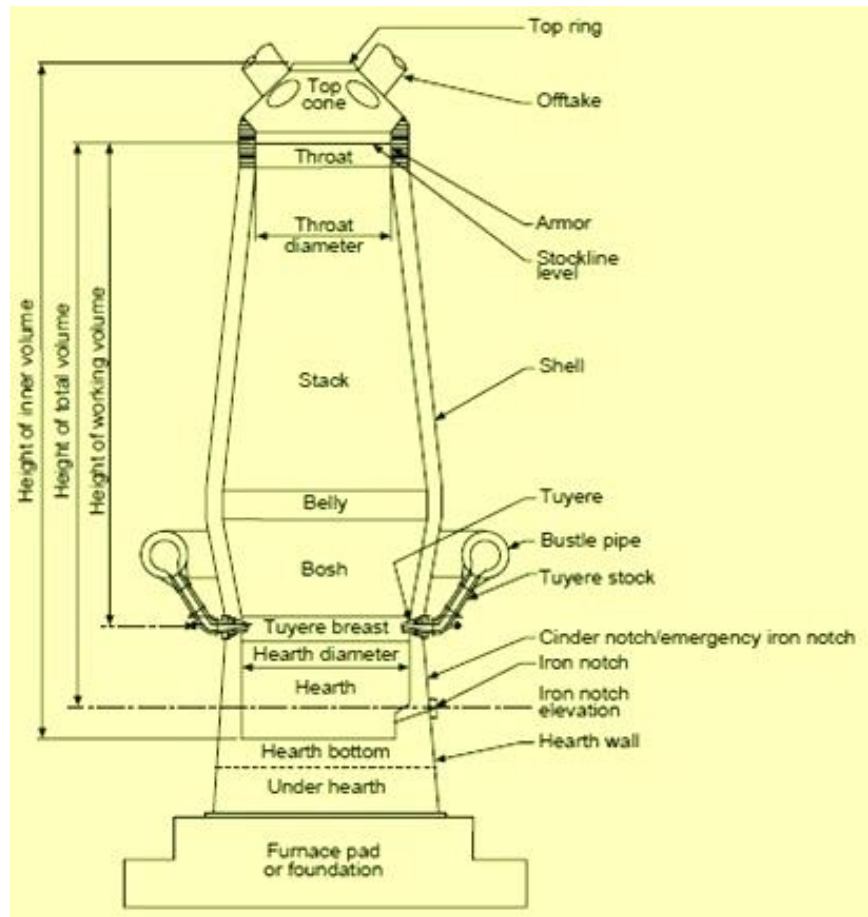
❖ FURNACE DETAILS:-

1. Foundation:

It is a massive steel reinforced concrete mass partially embedded below the ground level. It should be sufficiently strong to withstand the loaded furnace weight. It may be about 15m in diameter and 6-8m thick and lined with *fireclay* bricks.

2. Hearth:

It is a receptacle to collect the molten metal and slag. It is lined with *carbon* blocks. It essentially consists of a tap hole for Iron extraction and slag notch for slag removal.



But modern practices involve a common tap hole for both Iron and slag. These holes are closed with clay by means of a *Mud-gun* when not in use. It is about 3-3.5m in depth.

3. Bosh :

It is the zone of intense heat constructed with a steel reinforcement. The top of the bosh has the maximum diameter of the furnace to accommodate the volume expansion of the charge materials. Tuyeres are present at the bottom level of the bosh, which are inclined at some angle in the upward direction. It is about 3-4m high.

4. Stack :

It is a frustum of a huge cone mounted on the mantle and extends to the furnace top i.e., the bell, the charging system, the gas off-takes, etc. It is lined with *Fireclay* and is about 18- 20m in height. The top 2-3m, known as *Stock-line*, is protected from abrasion caused by the falling charge by providing armour plates on the inner surface of the lining.

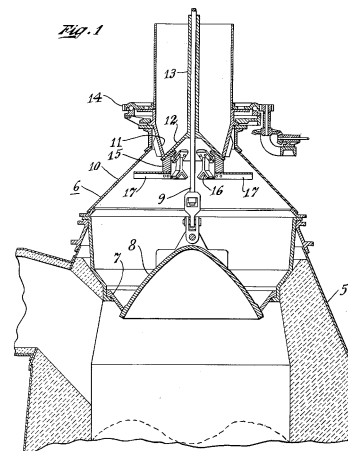
5. Tuyere and bustle pipe :

Immediately above the hearth are located the tuyeres through which hot air blast is blown for fuel combustion. Usually, the tuyeres are even numbered (between 10- 20) and uniformly spaced. Air from the hot blast stove is supplied through a huge circular pipe, known as the **bustle pipe**, which by virtue of its enormous size, equalizes the pressure of the blast at the tuyeres.

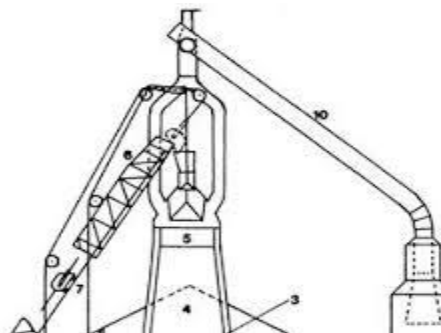


6. Bell and hopper :

It is a cup and cone arrangement used for charging the solid charge. But, modern furnaces prefer bell less charging system. Some furnaces use a **double bell** arrangement to ensure that the charging continues without the leakage of exhaust gas out of the furnace.



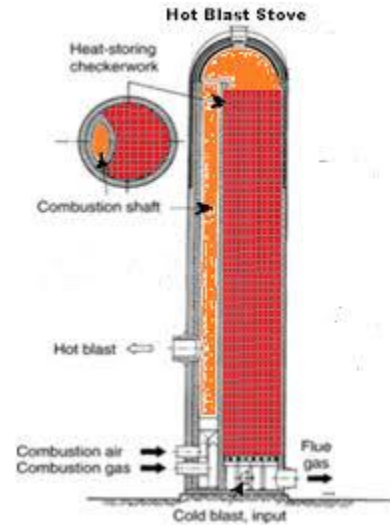
7. Off-takes :



These are four exhaust pipes connected to the furnace top evenly at four points. These rise vertically up above the furnace top and then join to a bigger single pipe known as the **down-comer** which delivers the gas to the gas cleaning system i.e, **dust catcher**.

❖ HOT BLAST STOVE:-

It is a regenerative furnace where the air blast is preheated to a temperature of 700- 1300 °C before entering into the blast furnace. Generally, three to four hot blast stoves are provided for each blast furnace. It is about 6- 9m in diameter and 30- 35m in height. During working one stove is '*on-blast*', where the pressurized air is inserted into the stove, while the remaining two or three are '*on-gas*', that is getting themselves heated by burning the cleaned blast furnace gas. Each stove consists of two chambers- *chequers chamber* and *combustion chamber*. The combustion chamber involves heating of the cleaned B.F. gas which results in absorption of heat by the chequer bricks.



Consequently, in the next cycle, the pressurised air supply is then forced into the chequers chamber which in turn absorbs the heat from the chequer bricks. This heated air blast is then supplied into the B.F. proper.

❖ GAS CLEANING:-

The B.F. gas is cleaned thoroughly before being used as a fuel. The *down-comer* opens up in a *dust-catcher*, where the coarse particles present in the gas settle down due to change in the direction of flow. This gas then passes through *wet and dry scrubbers* and an *electrostatic precipitator*. Nearly 25% of the B.F. gas is consumed in the stove for producing the hot blast. The remaining gas is used in other processes in the plant. A gas holder is used for storing the surplus gas.

LECTURE-3

Raw materials and their properties: Iron ores and flux

The raw materials required for the production of one tonne of iron under Indian conditions are given below:-

<i>Material</i>	<i>Quantity in kgs</i>
Iron ore/sinter/pellets(+60% Fe)	2000- 2500
Coke	450- 550
Pulverised coal(for injection through tuyeres)	50- 200
Limestone	300- 400
Air	4000- 5000

- Around 3.5- 4.0 tonnes of solid charge materials are required to make one tonne of pig iron.
- Iron constitutes about 4.6% of the earth's crust.

The function of the iron-bearing materials is to supply the element iron, which is 93–94% of the pig iron produced. The major iron-bearing materials are ore, sinter and pellets. Most of the ore is the ferric oxide known as hematite (Fe_2O_3) or the hydrated ferric oxides known as limonite or goethite ($\text{Fe}_2\text{O}_3 \cdot \text{XH}_2\text{O}$). However, in some instances the ores contain magnetite (Fe_3O_4) or siderite (FeC_2). Chemically pure ferric oxide contains 70% iron, but most ores contain only 50–65% iron as they contain 2 to 10% gangue (which consists mostly of alumina and silica) and chemically combined water. Most of the iron-bearing materials are screened to remove fines, to permit the achievement of higher wind and production rates and to permit smooth burden movement. The portion of the ore that is too fine to be charged directly is usually agglomerated in a sintering plant or as waste oxide briquettes.

➤ **Classification of Iron Ores:-**

- ***On the basis of their appearance:***

Black ore	Magnetite
Red or reddish black ore	Hematite
Brown ore	Limonite
Grey or whitish ore	Carbonate

- *On the basis of quality of gangue associated with them:*

<i>Nature of ore</i>	<i>Characteristics of associated gangue</i>
Siliceous	Predominantly silica
Aluminous	Largely alumina
Argillaceous	Chiefly clayey matter
Calcareous	Predominantly lime
Bituminous	Large amount of Bituminous or Coaly matter
Titaniferous	Large amount of ilmenite($\text{FeO} \cdot \text{TiO}_2$)

- *Hematite ores:-* These are usually of the *sedimentary- metamorphic* type. The *Sulphur* and the *Phosphorous* contents of Hematite ores are usually *low*. The gangue mainly consists of Silica but Alumina can also be appreciable.

World deposits of iron ores:-

The major iron ore producing countries are given in the table:

COUNTRIES	MINE PRODUCTION		CRUDE ORE	
	2005	2006	<i>Reserves</i>	<i>Reserve Base</i>
United States	54	54	6900	15000
Australia	262	270	15000	40000
Brazil	280	300	23000	61000
Canada	30	33	1700	3900
China	420	520	21000	46000
India	140	150	6600	9800
Iran	19	20	1800	2500
Kazakhstan	16	15	8300	19000
Mauritania	11	11	700	1500
Mexico	12	13	700	1500
Russia	97	105	25000	56000
South Africa	40	40	1000	2300
Sweden	23	24	3500	7800
Ukraine	69	73	30000	68000
Venezuela	20	20	4000	6000
Other countries	42	43	11000	30000
World total(rounded)	1540	1690	160000	370000

- India ranks 5th in the world in Iron ore resources and 3rd largest producer and exporter of iron ore in the world, only after Brazil and Australia.
- *Jharkhand* and *Orissa* have 43% of the total Indian ore reserves. Major mining regions are located in *Singhbhum* district of Jharkhand and *Keonjhar, Bolani, Mayurbhanj* and *Cuttack* districts of Orissa.
- *Bailadila* ore is the richest iron ore in India and is also one of the richest in the world.
- In Madhya Pradesh, iron ore deposits are located in *Bailadila range* of Baster district and *Dalli and Rajhara hills* in Durg district.
- In India, the cut-off grade for estimating the Hematite ore resources have been taken as more than 55% Fe.

VALUATION OF IRON ORES:-

- The value of Iron ore deposit depends on the following factors:-
 - Richness
 - Location
 - Composition of the gangue
 - Treatment and preparation needed before smelting
 - End use
- ✓ ***Richness*** means the percentage metallic iron in the ore. The richer the ore, the lesser will be the amount required to produce a unit weight of pig iron.
- ✓ ***Location*** means both geographical and geological. The geological location is related to whether the deposit is underground, as an outcrop or is in the form of a hill. Most of the iron ore deposits are located as hills, which allow them to be obtained at the ground level railway station under gravity. Geographical location is related to the distance and mode of transport required to bring the mined ore to the smelter and which includes the capital expenditure on installation of railways, harbours, ore carries, etc.
- ✓ ***The composition of gangue*** associated with an ore may reduce the value of a rich ore or may enhance the value of a lean ore, depending on its nature. For example, presence of Phosphorus in the ore affects the properties of cast iron adversely and complicates the steel making process. Alumina raises the m.p. of slag and hence requires additional fuel

to raise the temp. in order to make the slag a free flowing fluid. The presence of lime and Magnesia can be however mixed with siliceous ores to obtain a self-fluxing blend. As a whole, the value of an ore is drastically reduced by the presence of alkali oxides, reduced to some extent by the presence of Alumina but, enhanced by the presence of lime and/or magnesia.

- ✓ ***The treatment and preparation of ore before smelting*** determines the cost of the ore at the furnace e.g. dense ores require crushing, fine ores will have to be agglomerated and carbonate ores will have to be calcined to render them fit for charging in the B.F.

EVALUATION OF FLUX:-

A flux is a substance added during smelting to bring down the softening temp. of the gangue materials and to reduce the viscosity of the slag.

$$\text{(Slag=gangue+ flux)}$$

The value of a flux is expressed in terms of '***available base***', which depends upon the basicity (the ratio of base to acid content) of the slag under operating conditions.

$$\text{Available base} = \%(\text{CaO} + \text{MgO}) - \%(\text{SiO}_2) \cdot B$$

Where 'B' is the basicity.

- ❖ ***What is the available base of a flux containing 97%CaCO₃ and 3%SiO₂ in a blast furnace with slag basicity 1.5?***
- ❖ ***Why Hematite is mostly preferred instead of Magnetite?***

Ans. In Fe₂O₃(Hematite), the percentage of Fe is 70%.

In Fe₃O₄(Magnetite), the percentage of Fe is 72%.

Though iron content in Magnetite is more, it is not usually used. Because Hematite is abundantly available in the nature as compared to Magnetite and Hematite first converts into Magnetite as per the following reduction reaction which involves structure change from hexagonal to cubic with increased volume and makes it porous and more reducible.



LECTURE-4

Raw materials and their properties- Coke

Introduction

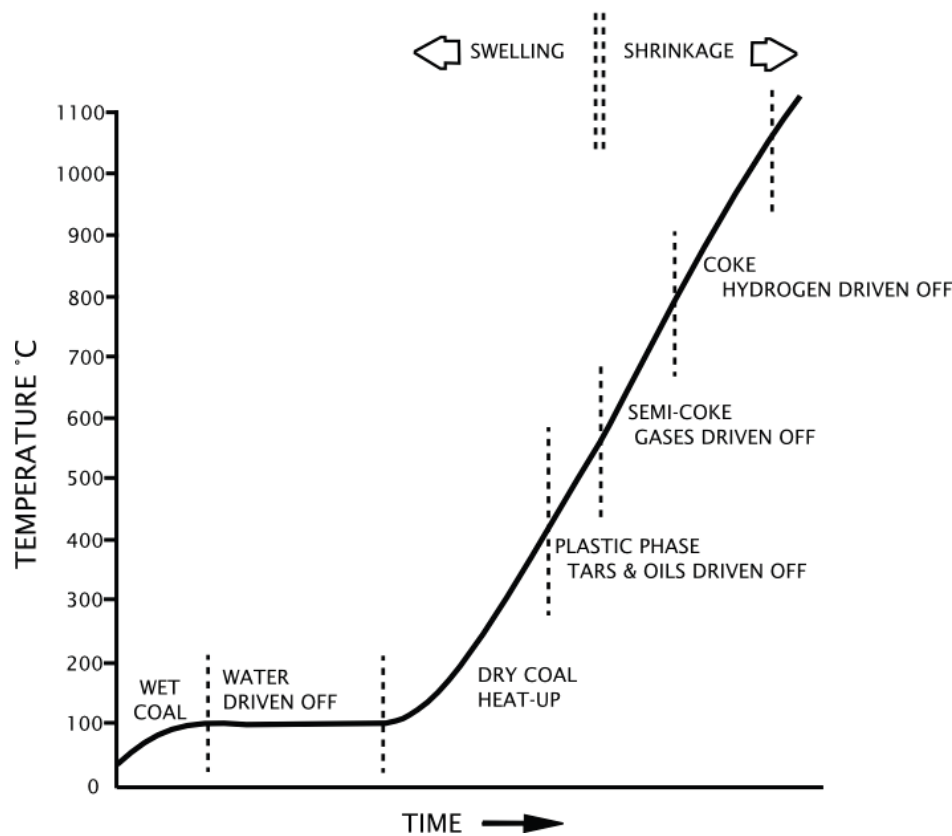
Although the oxides of iron may be reduced to metallic iron by many agents, carbon (directly or indirectly) is the reducing agent found to be best suited for the economical production of iron. Carbon of suitable reactivity and physical strength was at one time produced from wood by distillation, yielding wood charcoal; but for the operation of a modern large blast furnace the carbon required for the smelting of iron is obtained from the Coke, which is a high-carbon product, is usually produced in by-product coke ovens by destructive distillation of metallurgical coal. High-quality blast furnace coke should have low and consistent moisture content, low content of impurities, high strength, relatively low reactivity and a uniform size range. These properties are mainly governed by a proper mix of suitable coals and the optimization of the coking conditions such as coal size, bulk density and coke oven heating rate.

From a chemical point of view, coke has a rather simple composition, consisting of predominately carbon with some mineral matters and minor amounts of hydrogen, sulphur and oxygen. High ash content of the coke can be a problem, mainly for economy, since the blast furnace fuel rate increases and the productivity decreases. It affects the heat balance of the blast furnace in two ways: by lowering the carbon content in coke, which lowers calorific value, and by requiring additional slag former to melt the coke ash, which demands extra energy and increases the slag volume.

Coke-making process

Blast furnace cannot work without coke. Since coke is not a natural resource, coke making is an integral part of blast furnace iron making. There are three principal kinds of coke, classified according to the methods by which they are manufactured: low-, medium- and high temperature coke. Coke used for metallurgical purposes must be carbonized in the higher ranges of temperature (between 900°C and 1095°C). Blast furnace coke is produced by heating coking

coal to high temperature in the reducing atmosphere of the coke oven. The coal-to-coke transformation is illustrated in following figure. The coal mix is heated from the oven walls on each side of the coke oven and two plastic layers are developed on each side and converge eventually at the centre of the charge. First, coal near the wall loses water and becomes plastic at 350–400°C while leaving organic tars. The plastic layers solidify into a semicoke residue at 450–500°C while giving off methane and other volatile gases. It takes about 12–15 hours for the centre of the coke oven to solidify. The semicoke shrinks and fissures at elevated temperatures of 500–1000°C losing methane and hydrogen. The coke is pushed out of the coke oven at approximately 1000–1100°C, when the volatile matter is <1%. The coke-making process takes about 18–22 hours, but can vary depending on heating rate and width of the oven.



By product coke-oven

Metallurgical coke (coke used in blast furnace iron making) is produced in by product coke ovens. By – product coke ovens are flexible in treating different types of coal and to control the coke properties. Coke is charged batch wise into silica lined or refractory lined retorts and these

retorts are heated externally by burning gaseous fuels. Note that retorts are heated indirectly and coal is heated through heat transferred from the walls of the retort. Coal near the wall of the retort is heated faster than coal near the center. As a result, coke near the walls swells much earlier than coal at the center. Therefore, proper distribution of coal in the retort would be desirable. The volatile matter from the coal is collected in the byproduct recovery plant where by products are separated from each other. It takes around 18 hours to convert one batch of coal into coke. Coke is discharged from the other end of the retort through a mechanical hopper into a car, where it is wet quenched.

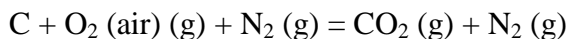
Note that hot coke so produced cannot be used directly in a blast furnace hence coke is cooled to room temperature. During wet quenching of coal a considerable amount of sensible heat is lost and pollutants are discharged into the atmosphere. The aspect of coke making is dealt separately in the next lecture with a quantitative illustration. The quality of coke depends both on temperature and rate of heating. Metallurgical grade coke is produced at a temperature higher than 1000°C.

Among the by-products, coke oven gas possesses both sensible heat and potential energy. Coke oven gas is used as a fuel to heat the furnace and also in heating the coke oven. The leakage of atmospheric air into the coke oven must be avoided as the air causes oxidation of C and results in a decrease in yields.

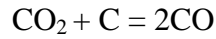
Functions of coke

- Source of thermal energy (Fuel)
- Produces and regenerates reducing agent (CO)
- Coke maintains permeability of the burden (Spacer)
- Reduces the melting point of iron

In the raceway (in front of the tuyeres) C in coke is combusted together with C from injected material such as oil or PC (pulverized coal) as follows:



N₂ is inert but carries heat which is released during the ascending of the gas. O₂ from hot blast is consumed and produces CO₂, which is unstable at temperatures above 1000°C in the presence of carbon, reacts further in the raceway with C of coke, and CO gas is formed by the following endothermic (solution loss) reaction



It is evident that the only materials which is solid till the bosh zone is nothing but the coke which acts as a permeable bed for the burden above it. Liquid metal and slag trickle down to hearth through the coke and also gases pass through it.

As we know carbon in iron decreases its melting point up to the eutectic composition about 4 wt%) the carbon content in pig iron/ hot metal is maintained around that composition to make the pyro-metallurgical extraction economic.

Strength reactivity, chemical composition etc. are the important properties

Coke consumption varies from 500 - 550 kg/ton of hot metal. Modern blast furnaces with pulverized coal injection work with 300 - 350 kg/ton of hot metal.

LECTURE-5

Testing of raw materials for blast furnace

Testing of coke:

The efficiency of a blast furnace is directly dependant on the permeability of the charge in the furnace. The quality of coke should be such that it gives minimum operational difficulties and maximum production rate .The properties of B/F fuel are

1. Chemical composition like fixed carbon, sulphur, phosphorus and other deleterious impurities.
 2. Reactivity
 3. Cellular and porosity
 4. Size range
 5. Thermal stability at high temperature
 6. Strength an abrasion resistance
1. **Chemical composition:** The chemical composition is important in determining the quality of coke. Chemically the useful component of coke is its fixed carbon and the balance is made up of ash volatile matter and other impurities. Ash usually contains refractory oxides like SiO_2 , Al_2O_3 , CaO , etc. and needs basic flux to maintain slag basicity. An increase in ash content of coke by 1% decreases the production by 3-6% and increase coke consumption by 4-5%.Coke is the prime source of sulphur so, the total amount of sulphur and phosphorus in coke is usually small. Proximate and ultimate analyses provide different data of coke chemical composition.
 2. **Reactivity:** It is defined as the rate at which the carbon atoms of coke react with oxidizing gasses like O_2 , CO_2 or H_2O . A highly reactive coke may react more readily with the CO_2 in the ascending gasses in the BF and a low reactivity may delay its burning. Reactivity is measured under standard conditions of experimentation. Knowing the physical properties of Coke is important as it predicts how coke will behave in a Blast furnace. The reactivity test for coke includes *Coke Reactivity Index (CRI) and Coke Strength after Reaction (CSR)* according to standard ASTM D 5341-93a.

Technique for determining lump coke reactivity in carbon dioxide gas at elevated temperatures and its strength after reaction in carbon dioxide gas by tumbling in a cylindrical chamber referred to as the 'I - tester'. When coke lumps descend in the blast furnace, they are subjected to reaction with countercurrent CO₂ and to abrasion as they rub together and against the walls of the furnace. These concurrent processes physically weaken and chemically react with the coke lumps, producing an excess of fines that can decrease burden permeability and result in increased coke rates and lost hot metal production. This test method is designed to indirectly measure this behaviour of coke in the blast furnace. Most blast furnaces will require a coke with a CSR greater than 60 and CRI less than 25.

A coke sample of 200 grams with a particle size between 19 - 22.5 mm is placed in a reactor and heated to 1100°C in inert atmosphere. Subsequently, the coke is degassed, i.e. reacting according to the Boudouard or the solution loss reaction, isothermally for two hours in 100% CO₂ gas atmosphere, and then cooled with nitrogen gas. After cooling, the coke is weighed and tumbled for 600 revolutions in an I-drum followed by sieving through screens with mesh sizes of +10 and - 0.5 mm. The weight loss of coke represents CRI, and the remaining coke on the +10 mm sieve represents the CSR.

3. **Porosity:** Apparent and true specific gravity, as determined by this test method, are influenced by the type of coals carbonized and the operating and preparational conditions of that carbonization, that is, charge bulk density, heating rate, and pulverization level. In turn, these properties directly influence the performance in processes using coke. This test method covers the determination of apparent specific gravity and true specific gravity of lump coke larger than 25-mm (1-in.) size and calculating porosity from the specific gravity data.
4. **Size range:** The characteristics of a good metallurgical coke are its strength and size range. The size range of coke is generally chosen to match the size of other raw materials to ensure maximum bed permeability for smooth furnace operation. The investigations have shown that the coke size has to be sufficient lumpy. It was found that a mean size of burden of 13 mm the mean size of coke should not be less than 53 mm .The optimum size of coke should be 3-5 times than that of iron bearing material.
5. **Thermal stability:** During its descent in the furnace coke gets progressively heated to 15-1600° C before it burns in front of the tuyeres . The temperature gradient in an individual piece of coke sets in differential contraction and expansion, particularly where large mineral matter

is present, resulting in high local stresses and its consequent tendency to degradation increases. The thermal stresses experienced by coke in the blast furnace are greater the larger is the lump size of the feed coke. The following factors are therefore believed to favor high thermal stability:

Absence of large lumps in feed coke

Uniformity of coke texture

3. Minimum inert inclusion of large sizes

4. High carbonization temperature and heat soak

5. Prior mechanical conditioning

6. low chemical reactivity

6. **Strength and abrasion resistance:** The blast furnace coke has to stand the rigorous of handling and which includes **drops** and **flows** at several places before landing in the furnace. In addition to this, coke has to stand high temperature and the nearly 20-25 m tall burden lying over it when it reaches the tuyere level. Height of modern furnace is controlled mainly by the strength of the available coke. It is an established practice to estimate the suitability of coke for blast furnace, in terms of its strength and abrasion resistance by measuring its **shatter** and **abrasion indices**. The shatter test gives resistance of coke to impact where as the **tumbler** test is designed to measure the resistance of coke to degradation by a combination of impact and abrasive forces. **Shatter tests** consists of dropping the obtained sample of certain fixed weight of coke from a standard height on a standard quality floor. The shatter index is expressed as percentage retained in various sieves of certain fixed sizes. The abrasion index is measured by putting a standard weight of sample in a drum and rotating it for a fixed number of revolution at a fixed speed.

Micum test: The most popular test is the Micum test as per the German standard DIN 51712 of 1950. In this test 50 kg. of air dried coke sample of +50 mm size is charged in a drum, one meter long and one meter in diameter and rotated for 100 revolutions at the rate of 25 r.p.m. The product is screened on 50, 40, 10 mm screens and results are expressed as M_{40} (% of total charge retained on 40 mm screen) and M_{10} (% of charge passing through 10 mm screen) values. M_{40} index gives the indication of hardness of coke and M_{10} its resistance to abrasion. The drum employed here in as longitudinal angle iron baffles from inside.

Testing of Iron Ore :

- a) **Reducibility:** The reducibility determines the ease with which oxygen can be removed from the iron oxide in the ore by reducing gases. This influences the productivity and quality of the product. The reducibility is inversely proportional to the time required reach some arbitrarily chosen degree of reduction.

The standard norm is represented by ,

$dR/dT = 0.5\%$ per minute ,minimum, which can be verified experimentally. Reducibility of the iore depends on size, shape and distribution of the ore. Hematite ore is chosen for DRI production, because it has better reducibility than magnetite ore.

- b) A tumble strength test measures two mechanisms of feedstock degradation, that is, the Tumble Index (TI) and the Abrasion Index (AI). It was carried out following the International Standard ISO 3271:1995(E) for determination of Tumble Strength for iron ore. Precisely, a 15kg test block sample was tumbled in a circular drum rotating at 25rpm for 200 revolutions. Subsequently, the ore was screened and fractions +6.3mm and -0.5mm were obtained. The percentage of the fractions in proportion to the feed weight is the value of the TI (+6.3mm) and AI (-0.5mm). The test was repeated four times and the average values for these tests represent the final TI and AI data.
- c) For estimation of a Shatter Index,a dried lump iron ore sample (10kg) of size -40 + 10mm was dropped 4 times from a height of 2m onto a cast iron floor (0.5 × 0.5 × 0.03m). Thereafter, the iron ore was screened and the shatter index expressed as the wt% passing through a 5mm sized screen (i.e., -5mm fraction). This procedure followed a test procedure, which were carried out on other ores.
- d) **Decrepitation :** When iron bearing materials are suddenly exposed to the exhaust gas temperature at the stock level on charging, breakdown may occur due to thermal shock. This is known as decrepitation.

Experimentally it is measured by dropping a known weight of material in a furnace previously heated to a temperature level of 400-600°C, under normal atmosphere, inert atmosphere or under mildly reducing conditions. After the charge attains the temperature it is removed, cooled and sieved to measure the breakdown.

In a typical test 500 g of 20-40 mm size undried ore is dropped in a furnace previously heated to a temperature level of 400°C and retained there for 30 min under a flow rate of 5000 litres of nitrogen per hour. The sample is then removed, cooled and the percentage of 0.5 mm and -5.6 + 0.5 mm material in the product is determined by sieving.

It is believed that ores with more than 10% porosity will not decrepitate

LECTURE 6

Burden Preparation-Agglomeration

Indian iron ores are characteristically friable in nature that results in generation of sizeable quantity of fines (-10 mm) every year during various stages of mining and processing in the country. Low grade iron ore, iron ore fines and iron ore tailings/slimes accumulated over the years at mine heads and generated during the existing washing processes, need to be beneficiated followed by agglomeration before using in domestic iron and steel industry.

We can't use iron ore fines as a feed material in blast furnace because:

1. It will fly away along with the blast furnace gas and additional load for gas cleaning unit.
2. Fines will block the gas passage and drastically decrease the bed permeability.

The process that has been under consideration can be classified as follows :-

- Briquetting.
- Nodulising.
- Vacuum extrusion.
- Sintering.
- Pelletizing.

Out of the above processes sintering on grate and pelletizing are widely used in preference to others, because of their technical & economic suitability for agglomeration of iron ore fines on a large scale.

Sintering

Sintering is the most widely used and economical agglomeration process for iron ore fines. Though it is a matured technology, there is a need to explore options to engineer sinter quality as per the changing needs. Sintering is essentially a process of heating of mass of fine particles to the stage of incipient fusion (temperature little below the melting or softening point) for the purpose of agglomerating them into lumps. In iron ore sintering the aim is to produce strong but porous agglomerate from a sandy (uncompacted) mass. Since coke breeze is normally available as an otherwise waste product in an integrated steel plant it is universally incorporated as a solid

fuel in the sinter mix. Down draught Dwight Lloyd sintering machine is usually employed for this purpose.

Pelletization

Pelletization is the prime focus of the agglomeration practices in the current scenario with the generation of high amount of ultra-ore fines (less than 100 mesh) at mines. The process of pelletizing combines mixing of raw material with or without binder, forming the green pellets of usually 7-20mm in size from moist mixture and a thermal treatment baking the soft green since the green balls lack necessary strength.

These two processes are there for discussed in detail while only a brief mention has been made about the other processes.

Sinter vs. pellet

Briquetting:-

Briquetting essentially consist of pressing of ore fines with or without a binder, into a block or briquette of suitable size and shape, and then subjecting it to a hardening process. A wide range of organic & inorganic binders like tar, pitch, cereal product, sodium silicate, ferrous sulphate, magnesium chloride limestone, cement, bentonite etc. have been tried with varying success.

On small scale briquetting does offer some advantages over other processes. In the early form of the process fine ore mixed with water was pressed into oblong blocks which were hardened in a tunnel kiln heated to about 1350°C.

Nodulising:-

In the nodulising process flue dust, pyrite residue or fine ore concentrate along with some carbonaceous material like tar are passed through a rotary kiln heated by gas or oil. The feed travels counter current to the gases. The temperature inside the kiln is just sufficient to soften the ore but not high enough to fuse the ore. The kiln is inclined slightly to the horizontal, rotates at 1-2 rpm & is nearly 30-60 mtr in length. The diameter at the feed end is about 2 mtr where as

that at the sinter zone is about double that at the feed end. Then it takes about 1.5-2.0 hours for the charge to travel through the kiln. The size of the nodules produced very considerably depending upon the tar content and the temperature.

The nodules varied considerably in composition & were too dense, slaggy & lacked the required porosity & hence this process could not find great favour. It has been largely replaced by either sintering or pelletizing.

Vacuum Extrusion:-

The process of vacuum extrusion otherwise used in ceramic industry, was adopted for producing iron ore compacts in the 1950's on a small scale. Moist ore with or without bentonite as a binder is fed in the deairing chamber & extruded into a cylindrical product which is cut into desired small sizes. The product is dried & fired before used in a blast furnace.

Blast furnace is the most established route for iron making. Blast furnace productivity has improved significantly over the years with the utilization of higher and higher proportions of quality agglomerates in the burden. Replacement of lumps by agglomerates in the burden has given advantage in terms of coke rate and high hot metal production.

LECTURE-7

Agglomeration of iron ores: Sintering

Introduction:-

Sintering is essentially a process of heating of mass of fine particles to the stage of incipient fusion (temperature little below melting or softening point) for the purpose of agglomerating them into lumps. Metal powder compacts are sintered to obtain desired solid shapes. In iron ore sintering the aim is to produce strong but porous agglomerate from a sandy mass. Compacting of ore fines prior to sintering, as is done for powder metallurgical articles, is therefore unnecessary. Similarly heating the entire bed of fine particles to the temperature required for sintering, as is done in powder metallurgy, is neither practicable nor economical. Sintering is therefore carried out layer by layer in a mass of charge.

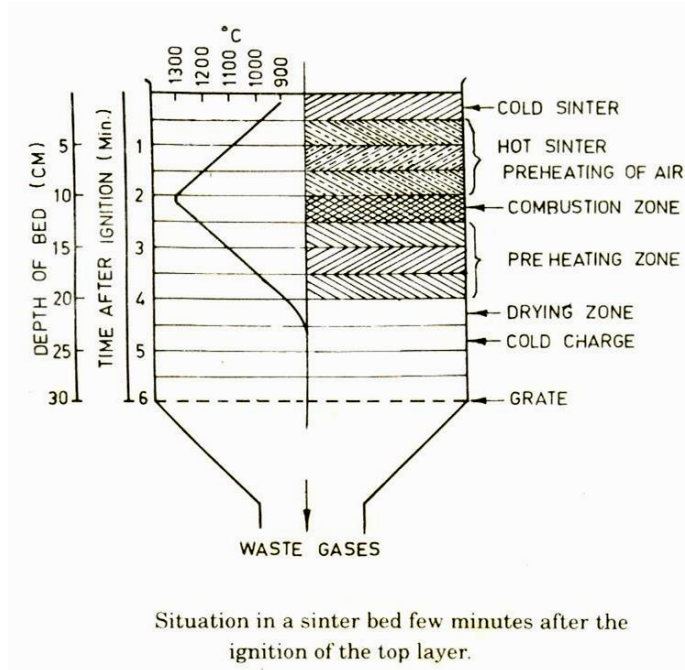
Principle of Sintering:-

Iron ore sintering, in essence, is carried out by putting mixture of iron bearing fines mixed with solid fuel on a permeable grate. Since coke breeze is normally available as an otherwise wasteful product in an integrated iron and steel plant, it is universally incorporated as a solid fuel in the sinter mix. It is equally true that this is the best choice among the available solid fuels. The top layer of the sinter bed is heated to the sintering temperature (1200-1300°C) by a gas or oil burner & air is drawn downwards, through the grate with the help of exhaust blower as connected, from underneath, to the grate. The narrow combustion zone is developed initially at the top layer travels through the bed raising the temperature of the bed layer by layer, to the sintering level.

The cold blast drawn through the bed cools the already sintered layers & thereby gets itself heated. The heat content in the blast is utilised in drying & preheating the lower layers in the bed. In advance of combustion therefore each layer gets dried and preheated by the heat transfer from the upper combustion zones. Much of the heat in the gas is absorbed by the lower portion of the bed.

In the combustion zone bonding takes place between the grains and a strong porous aggregate is formed. The process is over when the combustion zone has reached the lowest layer of the bed,

the sinter cake is then tipped from the grate in hot condition or after partial cooling. It is broken, screened & cooled to produce desire fraction. The under size is recycled and the oversize is fully cooled and send to the blast furnace.

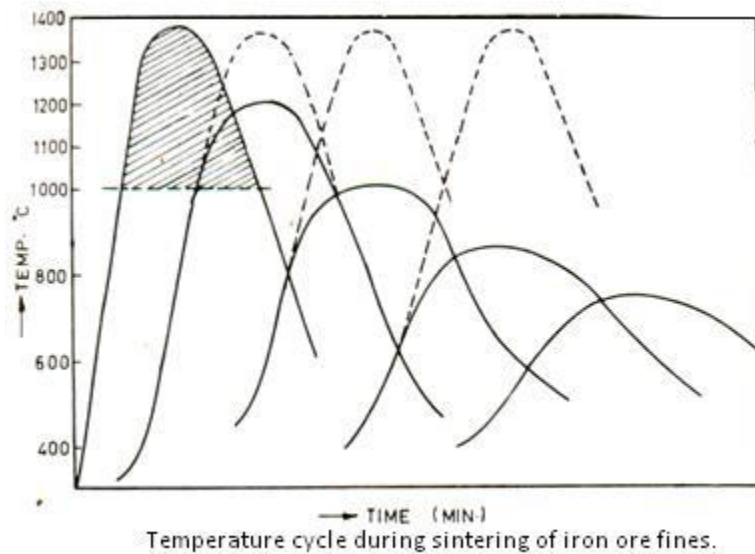


The process described above is known as down draught sintering since the air blast is drawn through the sinter beds downwards. In contradiction to this some non-ferrous ores are sintered by up draught sintering. In the up draught sintering, the initial small layer of the charge on the grate is ignited before making of the remaining bed thickness. Iron ores are however sintered by down draught sintering alone.

Process variables:-

During sintering the aggregate of particles, constituting the charge is heated layer by layer to the sintering temperature & also similarly cooled within a matter of some minutes (10-15). The temperature cycle, which any layer of the charge has to pass through, is qualitatively & ideally shown in the fig which is given below. The particle reaches the pick temperature of incipient fusion, so that, at the point of contact of particles, with each other, fusion bonding takes place. For a given mix it is most unlikely that the bonds of sufficient strength will be formed below a certain temperature level. Bonding of grains is therefore a high temperature phenomenon

& is a function temperature to which & the time for which the mass is maintained at the high temperature level. In other words, the area under the time- temperature curve as shown the fig essentially determines the nature & strength of the bonds developed during sintering of given mix. The nature of this graph is of paramount importance and depends upon rate of heating and cooling. The area under the curve above a certain temperature (1000° C for iron ores) is the effective factor in deciding the extent of sintering rather than the whole area under the curve.



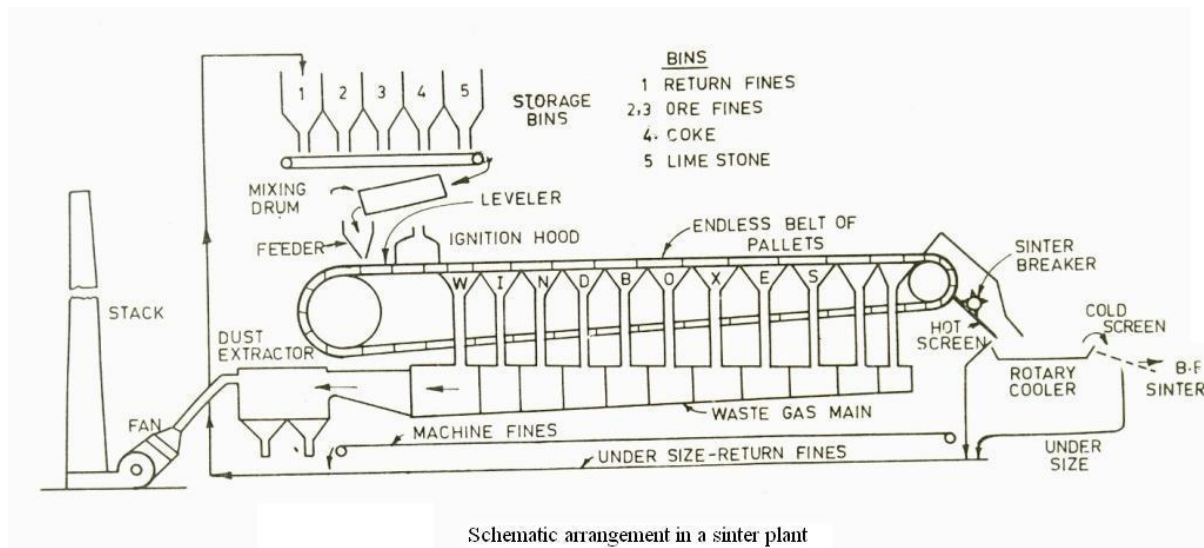
The variables of the sintering process are broadly as follows:

1. Bed permeability as decided by the particle size & shapes of the mix.
2. Thickness of the bed.
3. Total volume of air blast drawn through the bed for its sintering.
4. Rate of blast drawn through the bed during sintering.
5. Amount & quality of solid fuel incorporated in the charge.
6. Amount & type of carbonates present in the charged.
7. Amount of moisture in the charge.
8. Nature of ore fines i.e. its chemical composition.
9. Any non-uniformity in the bed composition or in the process of sintering.

Dwight Lloyed sintering machine:-

The design of an endless band of pallets moving over static wind boxes was originally set out by Dwight and Lloyed in the beginning of the present century in Mexico. The Dwight-Lloyed sintering machine is essentially an endless band of pallets moving over rails. Stretched across and over to huge pulleys, one of which is driven by a motor through a reduction gear system. The raw materials are loaded at one end of the machine and the top layer is ignited as it immediately comes under a fixed ignition hood. As the pallets move the ignited portion comes over a series of stationary wind boxes connected to an exhaust blower. Sintering of the charge is completed by the time the pallet travels over the nearly the whole useful length of the machine.

The sintered cake dropped out at the other end when the pallets turn upside down. The cake is broken, screened and the oversized is cooled. The under size usually -9 mm, is return to the machine for re4sintering where the oversize after rescreening goes to the blast furnace as charged. From the discharge end the pallets return to the feed end in inverted overhung fashion on rails. The exhaust gases from the wind boxes are let off into the atmosphere through a chimney after dust extraction.



The important parts of the machine and its accessories that make the complete plant are follows:-

- (a) Storage bins, mixers, feeders, etc.
- (b) Charge traveler
- (c) Ignition hood
- (d) Band of pellets and rails for its movement
- (e) Drive machine

- (f) Sinter breaker
- (g) Spillage collector
- (h) Wind boxes, dust extractor, exhaust fan, chimney, etc.

Storage Bins, Mixers and Feeders:-

Raw materials like iron ore fines, coke breeze, reclaimed flue dust , etc. are generally used for sintering. These may be stored in separate overhead bins from where weighed amount of these are delivered to the mixer to prepare the feed mix. Alternatively the raw materials may be laid in layers to prepare a huge pile of blend which is reclaimed and delivered to the mixer.

The proportioned raw materials are moistened and thoroughly mixed in the mixer to prepare a desired homogeneous feed for sintering. The mixer is a tromel or a drum fitted with paddles. After initial mixing the moist charge is rolled into a drum to nodulise the fines and there by decrease the size range of particles and improve consequently the permeability of sinter bed.

Charge Leveller:-

For smooth working it is necessary to have uniform thickness of the bed on the pallets. A flat plate is fitted immediately after the feeder to level off the charge. The cut off plate is generally adjusted to give a fixed thickness of the bed in the range of 300 – 450 mm. The importance of this, apparently simple device, should not be underestimated.

Pallet:-

The pallet has to support the mix during its travel from the feed end to discharge end. It must also stand high temperature developed during sintering without in any way bending or getting damaged whilst maintaining an efficient seal with the stationary wind boxes. The grate bars laid at the bottom of the pallet should provide maximum openings without allowing excessive fine materials being pulled through into the wind boxes during suction. The grate bars are generally made of high chrome cast iron & in order to avoid distortion three or four rows of bars per pallet are provided.

The side walls of the pallet have a maximum depth of about 600mm. The flanged rollers are attached on each side of the pallet, on the sides ,to make them run smoothly on the rails.

Ignition Hood:-

If proper ignition of the top layer is not achieved, both output and quality of the sinter will suffer adversely. For each mix proper ignition conditions are established by experiments. The main part of the hood ignites the top layer as usual & a series of gas fired burners are fitted in the hood extending over a substantial length of the stand to provide additional heat. Proper ignition depends on the nature, amount and distribution of the solid fuel in the mix.

Generally there are two types of ignition hoods:-

(a) Combustion chamber type:-

Two burners are located on the discharge half side of the hood and hence heating takes place partly directly & partly by the radiations from the hot roof.

(b) Nozzle type:-

One or more rows of burners are fitted into the hood over the entire width of the pallet to obtain ignition time of nearly a minute, the flame directly impinges on the charge & radiation plays a relatively minor role.

Product Handling:-

Because of the straight line movements of the pallets from the ignition to the discharge end the sinter cake is practically a one whole piece . At the discharge end when pallets change the direction of their movement, one by one, the sinter cake breaks in slabs of nearly the same dimensions as that of the pellet. These big pieces slide away from the pallet on the crash deck & the sinter breaker. The sinter breaker has a series of prongs or claws mounted on the shaft & these pass between fixed breaker bars. The lumps are broken by bending rather than shearing.

The sinter is broken, crushed, screened & oversized is cooled fully so that it could be carried to blast f/c even on rubber belt conveyors.

Although breaking & screening of sinter produce poses no unsurmountable technical problem, the abrasive nature and the hot condition of the product create problems in the maintenance of the system making the operation more expensive & time consuming.

Spillage Collector:-

The feed is expected to be laid on the pallets & the sinter cake to slide off the pallets on the crash deck. In spite of all care the materials do spill over & these must be collected & reused again.

The machine is covered on the sides & a longitudinal bunker is installed below the bottom track such that the spillage from the top track & sinter fines or lumps sticking to the grate even in its inverted position can be directed to the apron conveyor which conveys all these to the sinter screen for further processing.

Waste Gas System:-

Depending upon the thickness of the bed & bed quality feed, ignition condition, amount & quality of the feed, ignition condition, amount & quality of fuel incorporated in the mix, etc. an exhaust blower of suitable size is provided with the machine. The waste gas system mainly included the down comers, the dust hoppers, the waste gas main, the dust extractor, the exhaust blower & finally the chimney. The entire design should ensure as far as possible:-

1. Uniform suction across the width of the machine.
2. Collection of dust & charge particles falling through the grate in dust hopper itself.
3. Low velocity of waste gases to keep down abrasion due to dust laden gases.
4. Control on suction of individual wind box.
5. Minimum distortion due to thermal expansion when the machine comes to its peak performance, by providing expansion joint.

The waste gas main may be designed either for low velocity by making the cross section large or for high velocity by making it small. The latter is relatively cheaper but the abrasion problem can be serious.

Drive Mechanism:-

The pallets move from the feed end to the discharge end on the upper rail track & after discharging the sinter cake return back to the feed end in over hung fashion on the lower rail track. The drive mechanism is expected to effect smooth changeover of their movements from lower to upper rails and vice versa at the ends.

The main drive normally operates through a spur wheel & pinion arrangement to achieve the desired speed reduction. The most critical part of the machine is the discharge end & the major

problem in its design is how to accommodate the thermal expansion of the pallets as they move from the feed to the discharge end.

Efficiency of Sintering Machines:-

The efficiency of a sintering machine can be assessed in terms of the following parameters :

1. Productivity in tonnes of useful sinter per square meter of the working area per hour. Bigger is the figure, more efficient is the unit.
2. Effective suction as determined by the effectiveness of leak proof seals measures in terms of length of seals per square meter working area. Smaller is the length better is the efficiency.
3. The quantity of air drawn through the bed per unit time. The higher is the value better is the efficiency.

Types of Sinter:-

1. Acid sinter:-

The sinter mix does not contain flux at all. Flux is added in the furnace separately. The production of this type is fast declining and most modern practices produce self-fluxing or super flux sinter.

2. Fluxed sinter :-

The amount of flux added in the mix is such that the basicity of the mix is equal to that of the slag to be produced in the furnace. Separate addition of flux would be required only in proportion to the amount of natural lumpy ore charged in the furnace. In case of fluxed sinter, the replacement of lime should be done by incorporating dolomite in the sinter itself since the use of any dolomite will be associated with a sufficient slag fluidity and maximum possible desulphurisation by MgO, On the other hand, with high CaO, a limy slag will be formed with increase in viscosity and if Al_2O_3 is incorporated in slag the case will be worse.

3. Super fluxed Sinter :-

The entire amount of flux required to be otherwise charged in the furnace, when run on 100% natural ore charged is added in the mix. The basicity in the mix would be obviously more than that of the slag in the furnace, since a part of the charged would

neither be natural lumpy ore or sinter of lower basicity than that of the normal blast furnace slag.

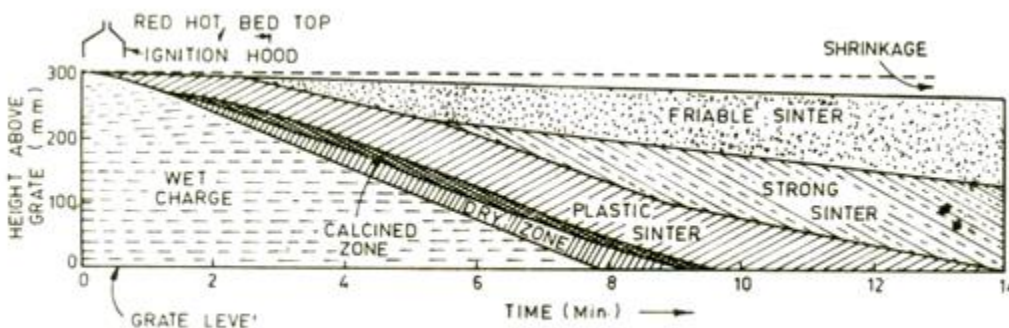
Mechanism of Sintering:-

The addition of flux in the sinter mix to produce fluxed sinter is beneficial because to that extent lime stone addition in the BF is reduced & the heat required for its calcinations inside the BF is also reduced proportionately.

The same thermal energy can be better utilised inside the BF for iron oxide reduction there by getting more productivity. The addition of flux in sinter – mix is therefore a universal adopted practice. This addition can go the extent that no flux will be added in the BF at all. It means that the actual basicity in the sinter will be much more than what would be maintained inside without loosing reducibility is what is desired.

The strength of sinter goes up with increase in the flux addition, but it simultaneously reduced the reducibility of the sinter because of the formation of a slag layer on the surface of the particles or the pore surface.

During sintering on a static bed various zones that are formed have shown in the figure. Each layer below the ignited top layer undergoes changes in order:



Vertical section through the sinter bed over the whole length showing the various zones

Wet ore → drying → calcining → preheating → combustion → cooling.

The same order of changes take place on a moving bed. The downward movement of each zone & forward movement of the pallet during sintering results in the structure, which is vertical section through the bed over the entire effective length of the strand. Each zone is extended

diagonally over the length. The chemical composition changes gradually across the various zones. The proportion of ferrous iron is more in the zone of calcination & combustion but it decrease in cooling.

Two types of bonds may be formed during sintering:-

1. Solid state bond:-It is formed as a result of recrystallization of the parent phase at the point of contact of two particles in solid state.
2. It is formed as a result of formation of low melting slag or glass at the point of contact of the two particles, depending upon the mineral constitution, flux addition.

The sinter can have three different constituents:-

- (a) Original mineral which has not undergone any chemical or physical change during sintering, more at the core of individual particles.
- (b) Original mineral constituents which have undergone changes in their physical structure without any change in chemistry.
- (c) Secondary constituents formed due to dissolution or reactions between two or more of the original constituents i.e. formations of ferrites, spinels, silicates, etc.

The proportion of each physical and chemical change during sintering depends upon the time temperature cycle of the process. The higher is the temperature more will be the proportion of new constituents by way of solution & interactions whereas lower is the temperature & longer is the duration more is the process of recrystallization in solid state.

The more is the slag bonding, stronger is the sinter but with less reducibility but less is the strength.

The amount and quality of solid fuel incorporated in the mix has a great bearing on the nature & extent of chemical changes occurring during sintering.

Factors Affecting Efficiency of Sinter:-

Size:-

If all other things are similar the strength of sinter is directly related to size distribution of the charge mix.

If the size will be large the contact area will be less & the strength of the sinter will be low & conversely if size is too small the contact area of particle will be large & the strength will be less.

Fuel content:-

The proportion of fuel in the charge has a bearing on the constitution of the sinter. Fuel content is usually in the range of 6 – 8%.

With the progress of sintering the air gets preheated to an increasingly higher level so that the fuel required to achieve the same sintering temperature decreases in the lower region of the bed.

The use of extended hood, gas sintering have been developed to decrease the solid fuel requirements in sintering.

Moisture:-

It maintains proper permeability in the bed during sintering. The rate rise of temperature at higher temperatures levels is more rapid than dry mix.

The presence of moisture increases the specific volume of air required for sintering.

Circulating load:-

For higher output of the sinter strand the circulating load should be low. A low circulating load reduces the permeability of the bed & consequently the input rate to the strand has to be reduced.

The circulating load the permeability & the bed thickness should in fact be related to the fan size, amount of leakage before the optimum values are established for efficient operation.

The top layer is generally more friable & the return fine arises from the portion. The circulating load can thus be reduced by decreasing the proportion of the top layer in the overall thickness by just increasing bed thickness.

Concluding Remarks:-

Sintering process of this type is one of the most efficient & compact design of heating iron ore charge on such a large scale. In fact such a system of direct heating should be used for other useful applications in metallurgical industries such as calcinations of limestone & the like.

LECTURE-8

Agglomeration of Iron Ores: Pelletisation

Introduction:

Pelletisation essentially consists of formation of small green balls by rolling a fine iron bearing material with a critical amount of water and to which an external binder or any other additive may be added if required. These green balls of nearly 5-20 mm size are then dried, pre-heated and fired, all under oxidizing condition, to a temperature of around 1250-1350°C. Bonds of good strength are developed between the particles at such high temperature. The sensible heat of the exhaust gases is recovered and is fed back in the induration operation.

The process therefore produces pellets in a highly oxidized state as against the sintering process in which solid fuel is incorporated in the charge and which finally results in a 5-20% ferrous iron.

Steps in the Process of Pelletisation:

- 1) Feed preparation
- 2) Green ball production and sizing
- 3) Green ball induration
 - a) *Drying*
 - b) *Preheating*
 - c) *Firing*
- 4) Cooling of hardened pellets

Theory of Balling:

The observation on ball formation that eventually led to the development of the theory of balling are as follow:-

- 1) Dry material does not pelletise and presence of moisture is essential to roll the powder into balls. Excessive water is also detrimental.

- 2) Surface tension of water in contact with the particles plays a dominant role in binding the particles together in green condition.
- 3) Rolling of moist materials leads to the formation of balls of very high densities which otherwise is attainable by compacting powder only under the application of very high pressure.
- 4) The ease with which material can be rolled into balls is almost directly proportional to the surface area of particles, that is its fineness.

From fundamental studies it has been concluded that there are three different water particle systems:-

1) **The pendular state**

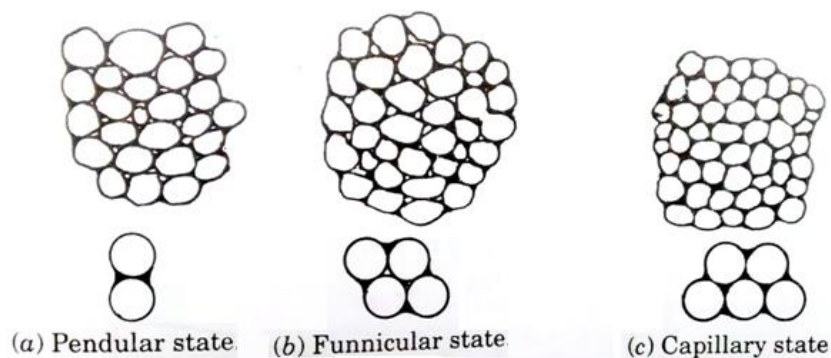
When water is present just at the point of contact of the particles and the surface tension hold the particle together as shown in figure (a)

2) **Funicular state**

When some pores are fully occupied by water in an aggregate system as shown in figure (b)

3) **The capillary state**

When all the pores are filled with water but there is no coherent film covering the entire surface of the particle as shown in figure (c)



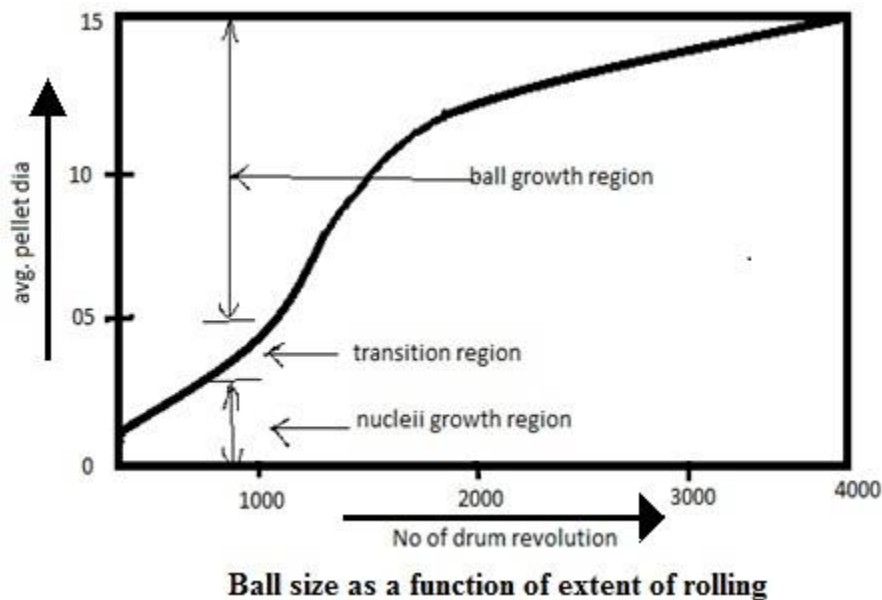
Water particle systems

Maximum strength of a green ball produced from a given material will be obtained by compacting the material to the minimum porosity and with just sufficient water to saturate the voids. The water content is quite critical to obtain good quality balls. The critical moisture can be assessed by determining the porosity of an aggregate.

Mechanism of Ball Formation:

The ball formation is a two stages process that is nucleation or seed formation and growth. If moisture is less than the critical amount its distribution tends to be non-uniform. If the moisture level is more than the critical value, growth rate is more but the balls produced are liable to deformation because of their plasticity. Seeds are formed only if critical moisture level is maintained and without which the process cannot proceed properly. The seeds generally have slight excess of water which imparts a certain degree of plasticity which is necessary for its growth.

Growth takes place by either layering or assimilation



General nature of the variation is shown in figure in which three regions can be clearly observed:

- 1) *Nuclei formation region*
- 2) *Transition region*
- 3) *Ball growth region*

1) *Nuclei formation region:-*

Several such particles initially join during rolling to form a highly porous loosely held aggregate and crumbs which undergo re-arrangement and partial packing in short duration to form small spherical,

stable nuclei. The growth rate of nuclei appears to be related to their specific surface and to the specific surface area of the feed material.

2) Transition region:-

After nuclei are formed they pass through a transition period in which the plastic nuclei further re-arrange and get compacted to eliminate the air voids present in them.

3) Ball growth region:-

The plastic and relatively wet granules grow if they are favorably oriented. In this process some granules may even break because of impacts, abrasion, etc. Growth takes place by two alternative modes.

i) Growth by **assimilation** is possible when balling proceeds **without** the addition of fresh feed material.

ii) Growth by **layering** is possible when balling proceeds **with** the addition of fresh feed material.

Growth by Assimilation:

If no fresh feed material is added for balling the rolling action may break some of the granules, particularly the small ones, and the material coalesces with those which grow. The bigger the ball the larger it will grow under these conditions. Since smaller granules are weaker they are the first victim and growth of bigger balls takes place at their expense. Growth continues till such size when the torque tending to separate balls exceeds the bond strength of coalescence. Balls with higher percentage of moisture therefore grow to larger size but the strength of the ball is low. It has been shown that after the initial growth by coalescence further growth may occur by assimilation, that is joining of two granules without poor fragmentation of either of them.

Growth by assimilation is more predominant in drum pelletiser, at least beyond the feed zone.

Growth by Layering:

Growth of the seeds is said to be taking place by layering when the balls pick up material while rolling on a layer of fresh feed. The amount of material picked up by the balls is directly proportional to its exposed surface, that is the increase in the size of the balls is independent of their actual size. In practice fresh material is always supplied to the pelletiser continuously while withdrawing equivalent amount of balls from the circuit. The conditions are, therefore, more

suitable for growth to takes place by layering. However, it has been observed that if growth by layering is pre-dominant the size range of the balls gets more narrower than that produced by assimilation mechanism.

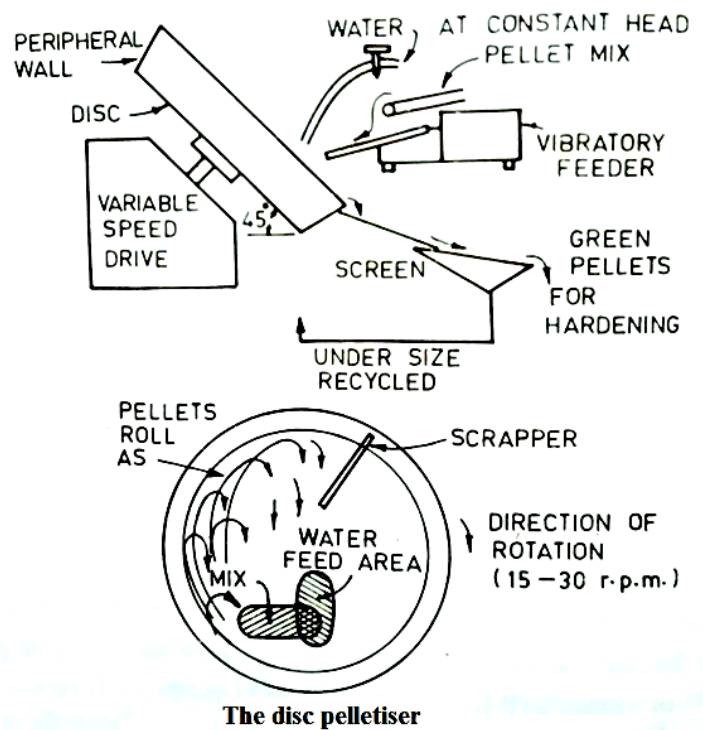
Growth by layering is more pre-dominant in the disc pelletiser .

Green Ball Production:

Green ball agglomerates are economically produced in balling drum or discs and rarely in other specialized equipment.

Disc pelletiser:

It is essentially a disc, with an outwardly sloping peripheral wall, which is rotated, around its own centre, in an inclined position to horizontal as shown in figure. These disc resemble flying saucers and are normally 3.6 - 5.6 m in dia. These are inclined at about 45° to the horizontal. The material to be pellatised is generally fed directly onto the disc and moisture level is made up with the help of water sprays. Scraper is provided to prevent buildup of moist material on the disc. It can also control the material flow pattern on the disc.



COMPARISON OF DISC AND DRUM PELLETER:

ITEM		DISC	DRUM
1.	Extent of mixing	Acts nearly as a perfect mixer	Not to the same extent of disc
2.	Classification of product	Acts as a classifier also.	Not like disc
3.	Nucleation	In the wet zone	In the wet zone
4.	Growth	Mainly by layering	More by assimilation
5.	Specific production rate	20 t/m ² of the pelletiser area	7-12t/m ² of the pelletiser area.

6.	Space requirements and capital cost	30% less than that required for equivalent capacity drum and proportionately less capital cost.	More than that for equivalent capacity disc
7.	Circulation load	Less and hence less power is required to run the disc.(=15% of the feed rate)	Both more 50-100% of the feed rate
8.	Flexibility of operation	More flexible with respect to rate of nucleation and growth.	Less flexible
9.	Product quality	Close sized product	Wide range of product size
10.	Necessity of screening of green balls	Adopted only if narrow size range product is required.	A must
11.	Size	For the same output the size of disc much larger than drum.	-----
12.	Supervision	More	Less

Additives:

Many times the iron bearing by itself may not roll into balls and may not produce green balls or required strength just by the addition of even the right amount of water. Under such circumstances addition of small amount of certain binders have successful in improving the balling operation as well as the green ball strength. Use of organic substances like starch, dextrin, alginate products, etc. has been beneficial in the preparation of green balls. But their use many times does pose insurmountable problems during induration of these balls and hence in general these are not preferred. Inorganic binders like ferrous sulphate, alkali chlorides, alkali and alkaline earth carbonates, etc. have been successfully used even practice. Bentonite to the tune 0.5-1.0% is generally added in the feed during its preparation. Excess of addition can lead to undesirable properties. Use of lime or dolomite as an additive is beneficial since it ultimately produces slag bonds during induration and which is exhibited as better strength of the pellet.

Induration of Pellets:

These green balls from the pelletisation plant are sent to the induration plant where these are subjected to drying, pre-heating, firing and cooling. For this purpose the pellets are subjected to a continuous heating and cooling cycle depending upon the variable of balling process, the mineralogical nature of the raw materials, the type of bonds to be developed and so on. The balls

attain a maximum temperature during the firing operation. The time for which the balls are subjected to this depends mainly on the quality of the pellets ultimately required. The heat recovered during cooling of the pellets is invariably utilised in drying the pellets.

The different types of heat hardening equipments are in commercial use at present time:

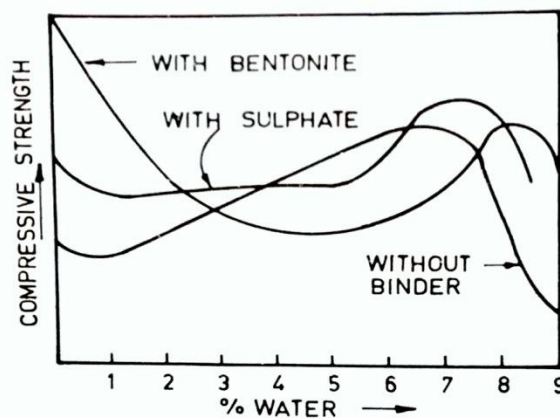
- 1) The vertical shaft furnace
- 2) The travelling grate
- 3) The grate kiln

In general green pellets are fed at one end of these modern machines and, hardened and cooled pellets are withdrawn from the other. In other words these are continuous induration units.

Drying:

The green balls are dried by passing hot air through the bed of the bed of the balls laid on a grate. The temperature of the hot blast and volume required for drying are largely a function of the moisture content of the green balls and the nature of the ore from which the balls are made. Specular hematites decrepitate

temperatures in excess of nearly 300°C and hence these should be dried as well as below this temperature. On the contrary natural magnetites may safely be dried at even 475°C . Use of binders increases the highest safe permissible temperature for drying. The depth of has to be suitably fixed to avoid condensation of moisture from gases in the cold zone. The strength of dried balls may be due to the mechanical interlocking obtained earlier during the rolling operation. Adequate dry strength of the balls is necessary to stand the pre heating and handling operation without giving away. The compressive strength of pellets varies with its moisture content during drying as shown in above figure.



Variation in compression strength of pellets as a function of moisture content during the drying cycle.

Pre-heating and Firing:

The hardened pellets should not only possess the optimum mechanical properties in the oxidised state but more so while these are being reduced in the blast furnace. The firing cycle should aim in this direction in developing the required nature and strength of the bonds. The role of additives has to be viewed from this point of view. As in sintering solid state bond (crystal growth) and the slag bond (due to melt formation) may be formed during firing. In the pre-heating stage partial oxidation of magnetite and sulphides, if present, occurs along with the calcination of carbonates and hydroxides. The solid bonding and grain growth commences in the pre-heating period. These are completed in the firing stage. Liquid formation by chemical combination of various acid and basic constituents and consequently the developments of slag bonds occurs pre dominantly during firing. The strength of the pellets is developed initially at the shell and progresses towards the centre with time at the firing temperature 1250-1350⁰C. Higher temperatures lead to excessive slag formation and less porous pellets a quality detrimental to blast furnace operation. The actual firing temperature is decided in terms of the basic feed material, the additives, the holding time at the maximum temperature, the size of pellets and the ultimate quality of pellets required. It should be clear that heat requirements for hardening hematite pellets are more than those for magnetite and similarly those for goethite, limonite, etc. are more than that of hematite.

Firing units:

1. The Shaft Kiln:

The green or dried pellets are fed vertically downwards in a central shaft of usually a rectangular cross section. fuel is burned in two fire chambers, one on each long side of the shaft, and the hot gases are allowed to enter the main shaft through multiple fuels. it is in a way counter-current gas solid heater. The fired pellets are cooled in the lower portion and if any chunks are formed these are broken by the chunk breakers and cooled pellets are discharged from the bottom. cooling air, introduced from below, gets preheated and is either taken out to burn the fuel in the chamber or to make available preheated gas for completing the combustion in the firing zone of the shaft. The temperature in the combustion chamber is around 1300⁰C. The furnace contains nearly 200t of pellets and production rate of hardened pellets is around 1000-2000t per day

2. The Grate Machine

These are essentially the modified adoptions of Dwight-Lloyd sintering machine in which the green balls are fed at one end on the continuous travelling grate and the hardened and cooled pellets are discharged from the other end. The length of the grate is divided into four zones viz drying, preheating, firing and cooling zone of suitable relative lengths. The hot air from the cooling zone is circulated in a complex manner to carry out drying, dehydration, preheating, etc. Down draught, up draught or a combination of the two is suitably employed in the design to carry out the preliminary operations before actual firing. Oil or gas burners are used in the firing zone to generate the necessary temperature. No solid fuel is incorporated along with the pellets.

3. The Grate Kiln:

It is combination of grate and a kiln and hence the name. The drying and pre-heating is carried out in this machine on the straight travelling grate, as in the grate system, and firing is completed in long rotating kiln wherein the fuel is burned to generate the necessary temperature. Hot gases from the kiln are carried over the grate, where drying and pre-heating is carried out under down-draught conditions or in combination with up-draught movement as well. Atypical scheme is shown in figure. The travelling grate in this case is not subject to very high temperature and hence the life of the grate is very high. It is one single factor that makes the machine a costly operation.

Concluding Remarks:

The process of pelletisation is not a competitor of sintering. On the contrary these are complementary to each other as fine are sintered and ultra-fines are pelletised.

LECTURE- 9

Blast Furnace Burdening and Distribution

1. Introduction:

The regulation of the proportions of ore, pellets, sinter, flux, coke and miscellaneous materials charged into the blast furnace is called burdening. Uniformly permeable burden offers uniform resistance to passage of up-going gases in the counter current reactor i.e the in the B/F. the production rate of blast furnace is directly determined by the two factors viz:

- i. The rate of reduction of iron oxide
- ii. The rate of heating of the burden

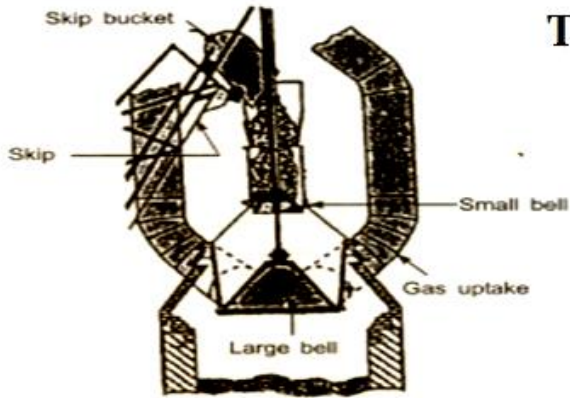
These two factors are related to the quantity of blast that goes through the furnace and depends upon the degree and time of contact with the gases with the burden. For efficient reduction and heating of burden, the burden should have uniform and good bulk permeability throughout the cross section to offer uniform resistance to the blast and increases its residence time. This also enhances CO utilization in other words decreases coke consumption.

Burden distribution is a generic term used to denote radial ore/coke distribution as well as particle size distribution in a blast furnace. The initial distribution of charge in the throat usually persists throughout the furnace shaft so, it is of much importance in the bed permeability.

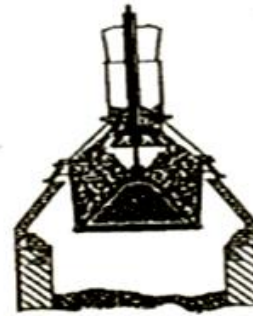
2. Charging systems

- a) Bell Charging
- b) Bell less Charging

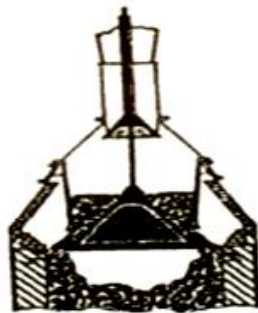
Two Bell Charging System



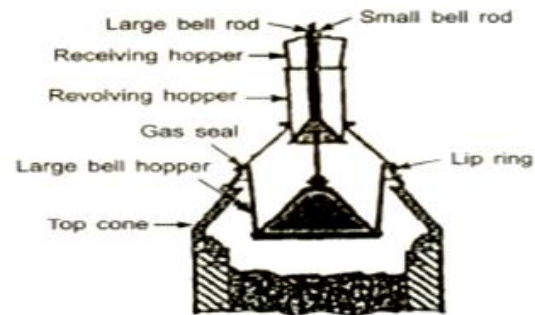
(a) Small bell and large bell both closed; skip bucket tipped to dump charge in hopper above small bell. Gas flowing from top of furnace through uptakes located in dome (top cone).



(b) Large bell remains closed while small bell opens to admit charge to large bell hopper.



(c) Small bell closed to prevent escape of gas to atmosphere and large bell open to admit charge to the furnace.



(d) Both bells closed, ready to repeat charging cycle. Note that the rod supporting the large bell passes through a hollow rod supporting the small bell, permitting independent operation of bells.

Bell less Charging



Compared with traditional double bell top charging facility, bell-less top charging facility has following advantages:

- High furnace top pressure: it is beneficial to maintain BF top operation pressure in a high level in long term and good for improving the BF operation index.
- Compact design, low space requirements
- Time based discharge control
- Operation and control of the BF can be improved.
- Quality of pig iron will be improved.
- Control of gas current can be optimized under high PCI ratio.
- Material distribution is flexible
- Maintenance and overhaul is easy and simple and cost of maintenance is also lower than the double bell charging facility.
- Low investment costs

3. Factors affecting Burden Distribution

A) Furnace Design

- a. Angle and size of the BIG bell
- b. Additional mechanical devices
- c. Speed of lowering the BIG bell

B) Physical properties of burden

- a. Size range of the burden
- b. Angle of repose
- c. Density

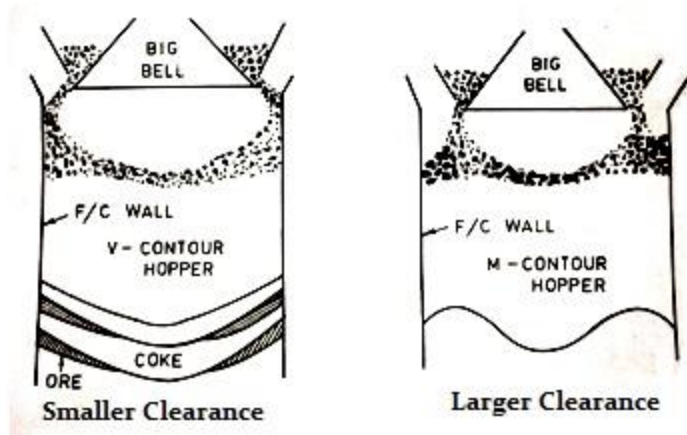
C) Control system

- a. Distribution of charge on the BIG bell
- b. Stock level
- c. Order of charging

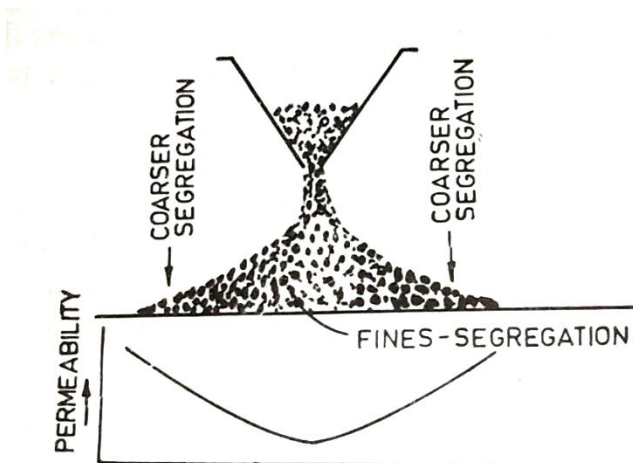
Angle and size of the BIG bell

Angle: 50-53° from the horizontal for rapid discharge & preventing sticking of wet materials

Clearance between big bell and f/c wall



We know that dense small and irregular particles (like iron ore) remain substantially where they fall whilst light large smooth particles (like coke) rolls into troughs.



More clearance leads to formation of crest (peak/heap) of fines further away from the f/c wall and formation of M contour(hopper) of stockline , less clearance means more fines at the wall forming V contour. The permeability of the whole improves with a crest reasonably away from the wall. In other words it accounts for keeping the clearance between the BIG bell and the f/c wall as large as possible or having a larger throat to bell dia ratio.

The normal clearance is nearly 800-1000mm or a good throat-bell dia ratio seems to be 1.36

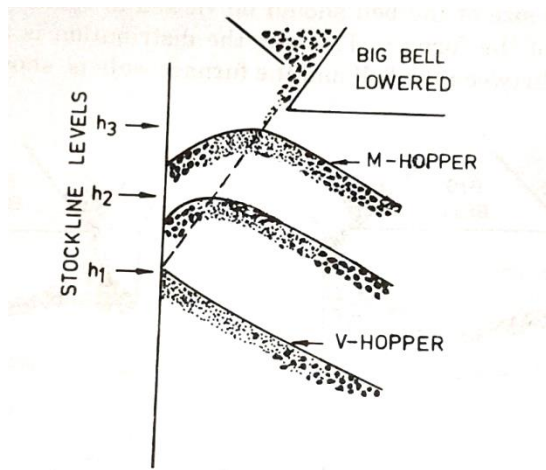
Stock line level

Stock line: The distribution pattern at the top. Charge or stock level in the furnace throat

Zero stock line: Horizontal plane formed by bottom of big bell when closed.

For example 6ft stock level for instance located 6ft below zero stock line.

As b/f is a continuous process the stock line goes on descending and the hopper alters as per figure.



It is important to maintain stock line at optimum level. Therefore stock line bars are used which indicate for timely charging. These are suspended from the top and rest on the top of the burden to indicate the stock level. During charging these are lifted up thereafter they are again placed.

Speed of lowering the BIG bell

Speed at which big bell is lowered alters the contour as faster speed throws the burden away towards the wall and slower speed leads to segregate the fines. Therefore lower speed should be maximum but speed of raising the big bell should not be high enough as it may strike the seat too hard.

Density of charge materials

Densities of ore(5-6gm/cc), coke(1.5) and flux(3-3.5) are quite different. Since density values cannot be altered, the sizes may be so chosen that their differential rolling tendencies are offset to some extent. From this point of view agglomerates are more preferable to lump ores.

Angle of repose

The base angle of the conical heap of the particles formed on a horizontal plane when allowed to fall gently is called the “angle of repose” of that material. Angle of repose depends upon the particle size, size distribution, shape, surface characteristic, moisture content etc. of that material.

Material	Size distribution	Avg. size	Angle of repose
Iron ore	10 – 30mm	18mm	33-35°
Coke	27-75mm	45mm	35-38°
Sinter			31-34°
Pellet			26-28°

Surface wide size range, roughness, moisture, clay content increases angle of repose. Higher the angle of repose higher the ridges.

Size and size range

Fines (<5mm) maximum of 5%

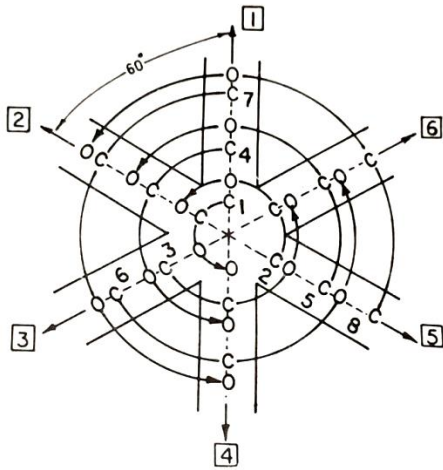
Permeability decreases with increases in size range % of fines (generally <10mm, sometimes <5mm).

Top size of iron ore should be as low as possible. Because-The rate of reduction of iron ore decreases drastically (perhaps exponentially) with increasing size.

Distribution of charge on BIG bell

Two skip cars are employed to haul the charge materials to the small bell hopper. The McKee revolving distributor ensures good mixing of the materials. For a charging sequence CCOO↓ the small bell hopper is rotated through 60° after every skip is dumped on the small bell and the small bell is lowered after every four skips to dump the charge on to the BIG bell. When 8(eight) such charges that means 32 skips are deposited on the BIG bell, it is lowered to allow the whole

material to fall in to the furnaces. By virtue of this operation there will be alternate layer of ore and coke and the mixing is improved further when it is dropped in to the furnace.



Additional mechanical devices

Paul-Wurth –Bell Less –Top(PW-BLT) proved to be beneficial for high top pressure in modern f/cs. Tata steel practiced Moving Throat Armour (MTA) and proved to be coke saving.

Order of charging

Proper sequencing of charge accounts for uniform gas flow in the bed. With the usual M-contour a large vol of gases passes through the peripheral ring. Maximum loading this space with iron ore leads to better gas utilization and less erosion of wall lining by gas velocity. But excessive loading may restrict the uniform gas flow.

$C_n O_m \downarrow$: commonly used

$O_m C_n \downarrow$: gradual overloading of centre and periphery with ore

$O_m \downarrow C_n \downarrow$: overloading of centre but not the periphery with ore

$C_n \downarrow O_m \downarrow$: overloading of periphery but not the centre with ore

Uniformity index: this index of distribution quantifies the degree of uniformity ore and coke layer thickness in radial direction

LECTURE- 10

Blast Furnace Profile

The reduction of iron oxide is a high temperature process and the thermal requirements of the process are met by burning coke, injections if any, and the enthalpy of the hot blast. The maximum temperature is therefore generated in front of the tuyeres where hot blast burns preheated coke. This temperature may be as high as 2000°C . The temperature inside the furnace decreases on either side of the tuyere level. The exhaust gas temperature is as low as 200°C and the temperature in the hearth may be around $1300\text{--}1500^{\circ}\text{C}$ depending upon

the chemistry of the slag ; a typical temperature being 1350°C . The temperature variations inside the furnace along the vertical axis are shown in Fig. 10.1. The pressure and gas composition profiles are shown alongwith the temperature profiles. The horizontal variations in temperature are shown on the vertical profile of the blast furnace. The gas composition and temperature vary smoothly over the vertical as well as the horizontal cross section of the furnace. The chemical processes are therefore continuous from bottom to top and throughout the volume of the furnace.

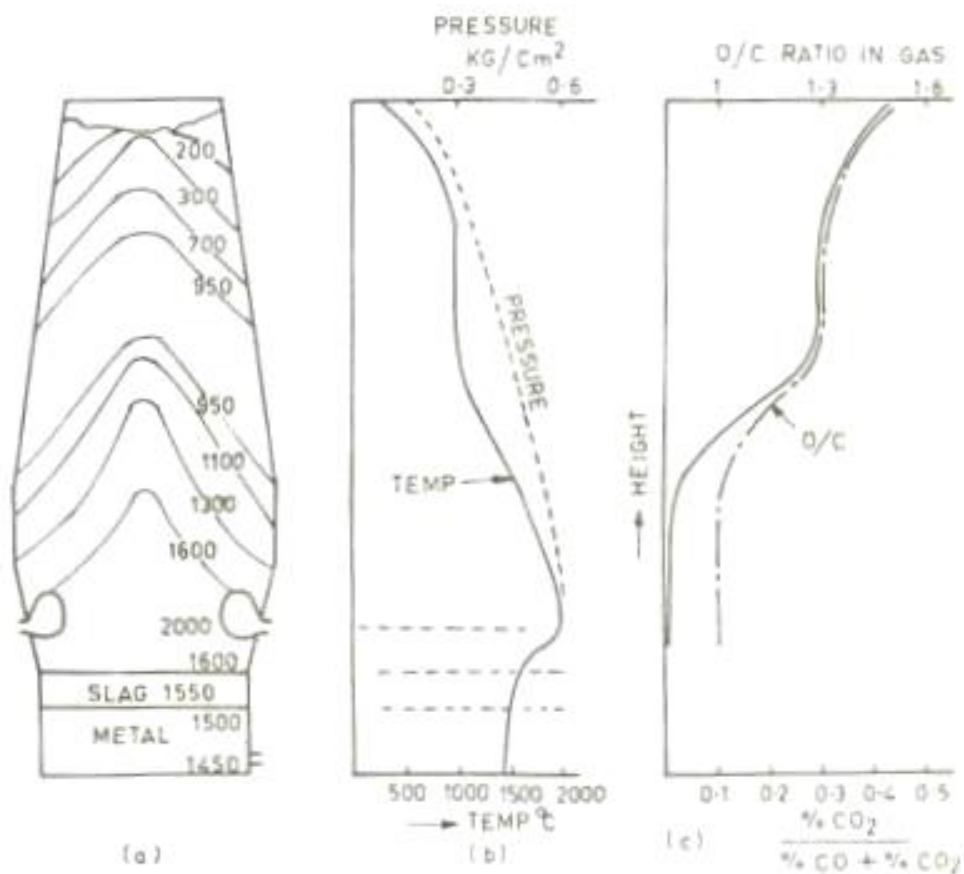


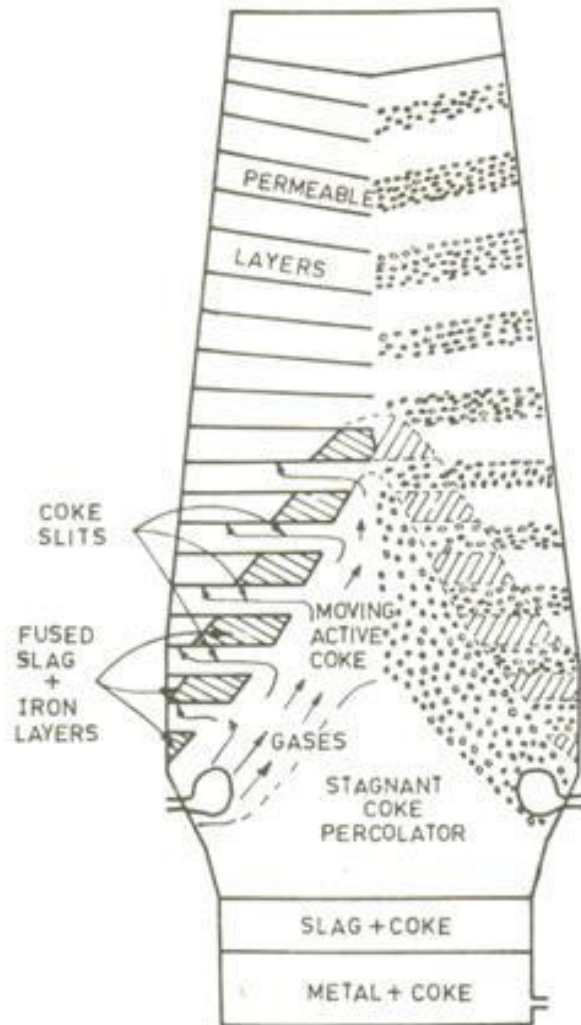
Fig. 10-1. Temperature, pressure and gas composition profiles in a blast furnace.

(a) Shows the profiles of temperatures in °C in a vertical section, (b) and (c) show the profiles along the vertical axis of the furnace.

10-1-2. Physical Structure

The hot blast enters the furnace through the tuyeres at a velocity of around 200–300 m/sec and at a pressure of 2–4 atmos. Such a high pressure is necessary to push the reducing gases through the solid burden and to overcome the top pressure, if any, applied from top. These conditions create a raceway of 1–2 m in front of each tuyere and which

can be easily penetrated by a bar pushed through a tuyere. Each raceway at its sides and below is bound by rather firm regions of lump coke which has bypassed oxidation during its descent through the furnace. Studies of quenched blast furnaces indicate that this coke extends downwards into the iron pool and perhaps even to the hearth bricks. This is what has been shown in Fig. 10-2. as stagnant coke percolator.



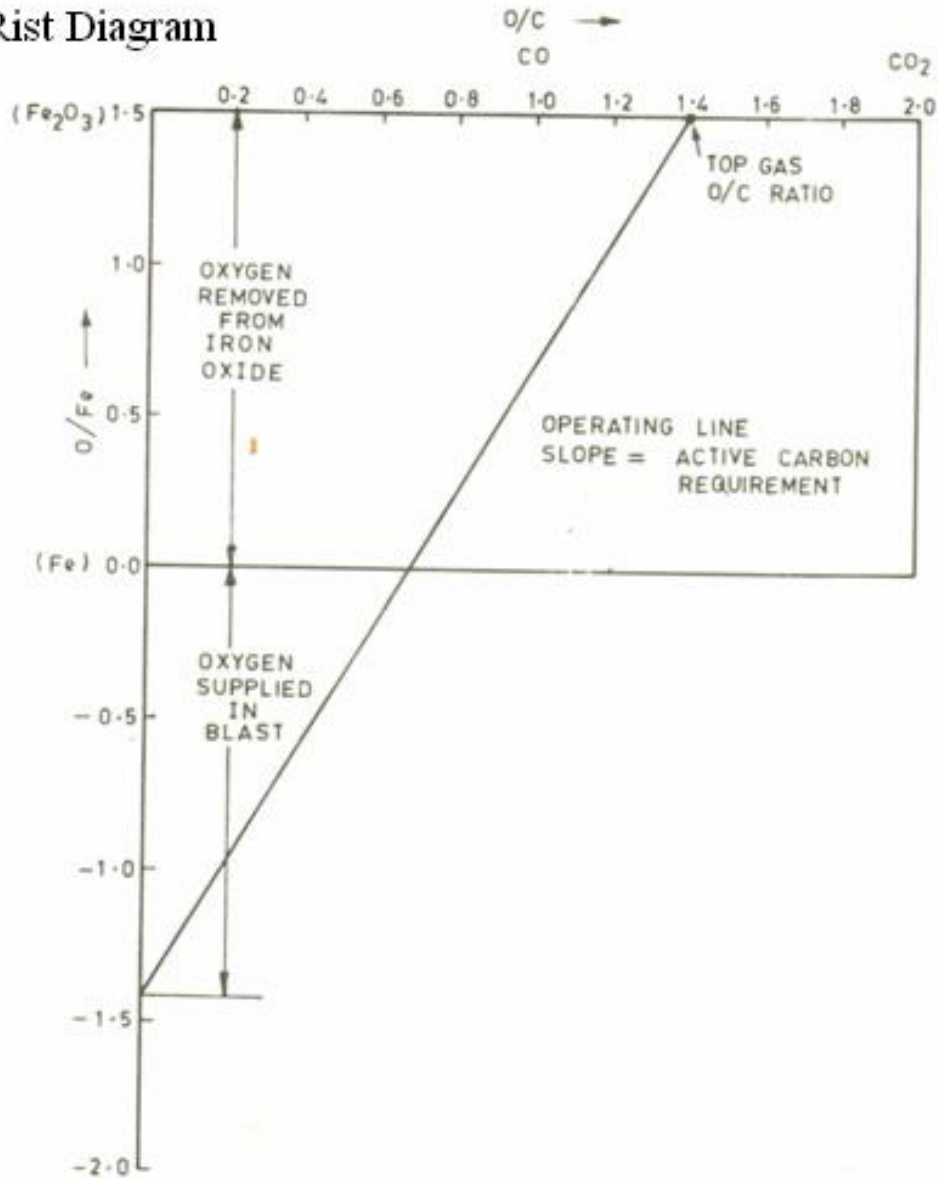
Idealised internal structure of a blast furnace in operation. The most notable feature of this structure is the fusion zone with its alternate layers of 0.5-1.0 m thick of (i) coke and (ii) fused slag plus iron. The moving active zone of coke through which the gases ascend and the stagnant coke bed through which liquid slag and metal percolate are also shown⁴⁸.

The top of raceway region is bound by loosely packed lump coke since gas has to flow upwards through it. The bottom most pieces in this region periodically fall into the raceway and are consumed by the

hot blast. The coke bed thus, by and large, slowly moves down, gets consumed and is replenished from top. The above description indicates that almost all the solid material in the hearth and bosh is coke alone. Liquid iron and slag percolate through this and collect in the hearth. The hot gases rising from the raceways heat up the descending coke and, liquid slag and metal. During percolation reduction of iron oxide is completed and it gets saturated with carbon. Partial reduction of other oxides like those of phosphorus, silicon, manganese, etc. also occur alongwith and that makes the metal impure.

Studies of commercial quenched blast furnaces have shown that the region of loose packed coke above the raceways is bounded on top by inverted 'U' shaped fusion zone consisting of alternate layers of coke and other charge materials as developed at the stockline level by way of charging sequence. The structure is shown in Fig. 10-2. The inverted U shape is characteristically an important feature. It supports the furnace burden, aided by the gas pressure from below. Until all the charge is solid entire burden is permeable to ascending gases. In the fusion zone the ore, flux, etc. melt and the permeability of the bed is only due to the existence of alternate coke 'SLITS' as shown in Fig. 10-2. The gas flow therefore tends to become horizontal and the coke quality must ensure this permeability during the operation. Any breakdown of coke above the fusion zone therefore tends to effect permeability, and consequently the operation, adversely. As the charge melts in the fusion zone it trickles down through the coke bed and the coke slits slowly become a part of the moving active zone. The slag while percolating through the moving active zone of coke absorbs the ash released from the coke on burning. Any iron oxide contained in the slag is reduced fully during percolation through the stagnant coke percolator. The structure of the burden above the fusion zone is practically the same as is obtained at the stockline level by way of charging sequence. The reduction of iron oxide there is mainly by gas *i.e.* indirect reduction.

The Rist Diagram

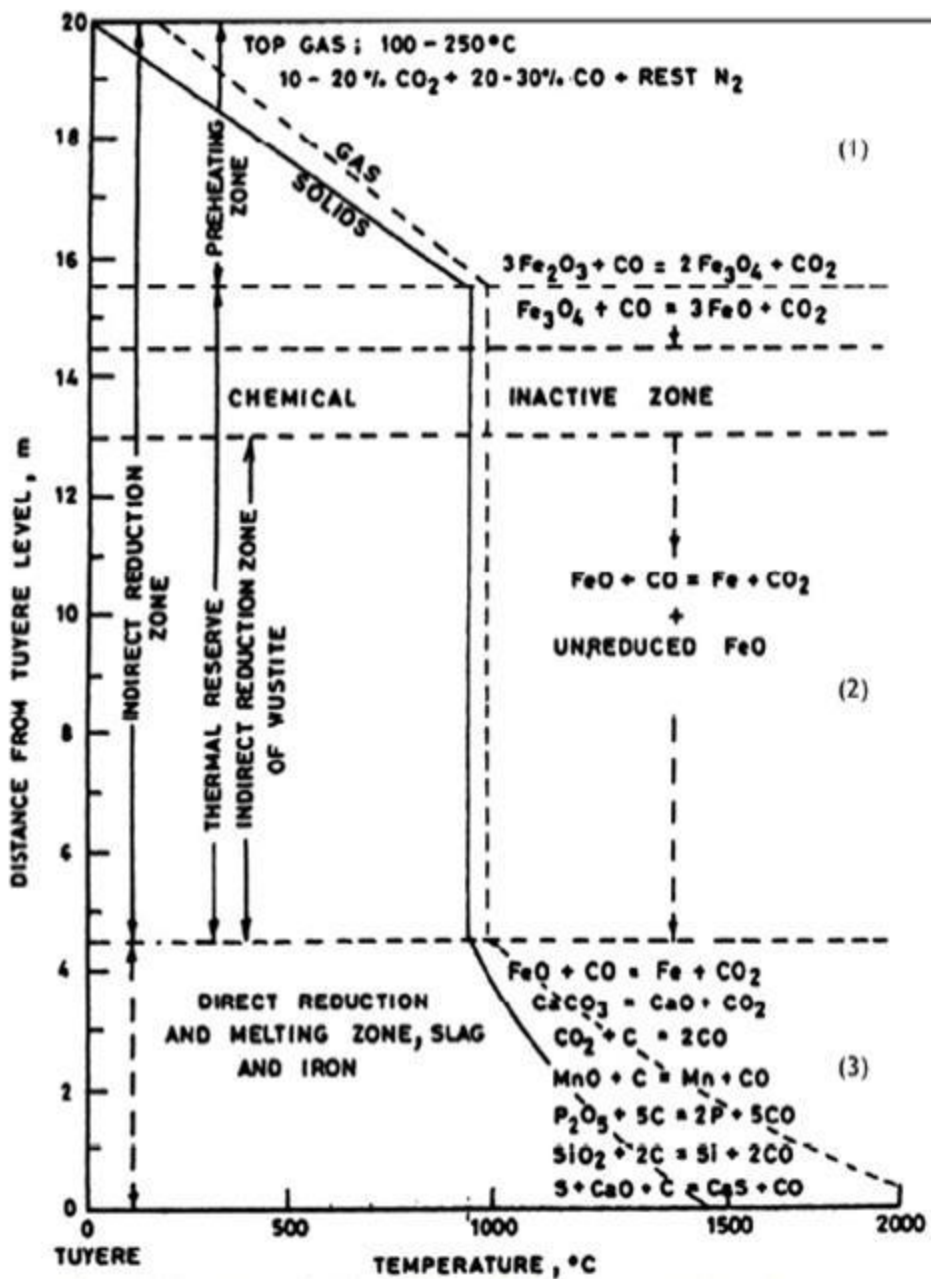


The Rist Diagram showing the operating line for a simplified blast furnace operation defined by Eq. (10-10).

In words it means :

$$\left. \begin{array}{l} \text{No. of moles of oxygen} \\ \text{in the blast} \\ + \\ \text{No. of moles of oxygen} \\ \text{in oxide} \end{array} \right\} = \left\{ \begin{array}{l} \text{No. of moles of active carbon} \\ \text{multiplied by} \\ \text{No. of moles of oxygen in gas} \end{array} \right.$$

(all per mole of product Fe).



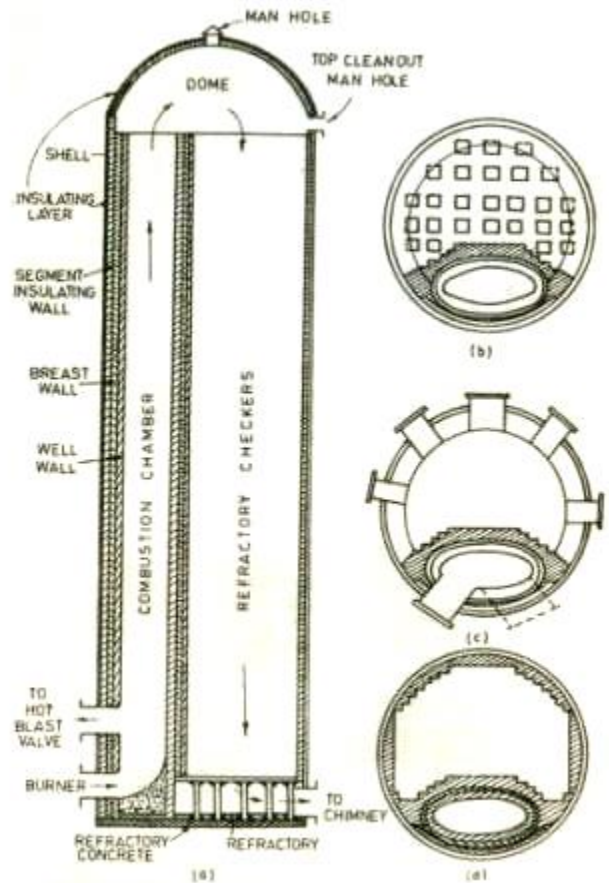
LECTURE-11

Blast Furnace stove, Gas Cleaning Unit & Instrumentation

Hot Blast stove

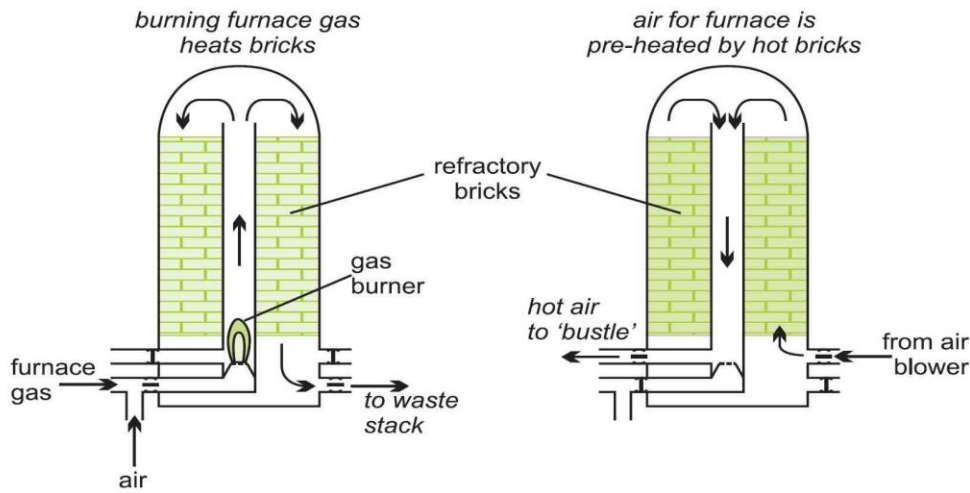
Introduction:

The blast required for running a blast furnace is first heated in what are commonly known as hot blast stoves. It essentially consists of a tall cylindrical structure comprising of a combustion chamber and a heat generator unit of checker bricks. The clean blast furnace gas is burnt in the combustion chamber and the hot products of combustion later heat up the checker bricks. In this case the stove is said to be on **'on gas'** and is maintained on gas until the checker bricks are heated to a certain temperature. Firing is stopped and cold blast is passed through checkers which impart the heat stored in them and there by produce preheated blast. The stove is said to be **'on blast'**. The stove is again put on gas and the cycle is repeated.



The thermal efficiency of the stove varies between 75-90%. For an uninterrupted steady supply of blast at specified temperature a battery of at least three stoves is necessary. A two stove system is quite unsatisfactory and hence three or four stove system is preferred. The modern trend is to manage with a three stove system of efficient design to last practically the whole

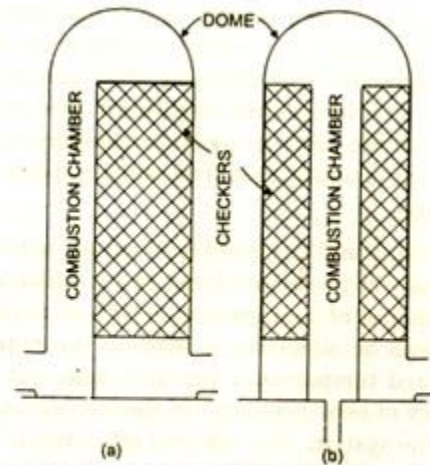
furnace campaign without any major intermediate repairs .In a three stove system one is on blast and the other two are on gas at any time.



Construction of stove and its accessories

The stove is a tall cylindrical steel shell with height ranging between 20-36 m and diameter 6-8 m. has a dome shaped design. It is lined from inside thoroughly with good insulation. The wall separating the combustion chambers from the checkers is also heavy duty firebrick construction. The dome is also lined with heavy duty firebricks. The checkers are refractory construction of special brick shapes. The bricks should have maximum weight with maximum surface areas of flues i.e. maximum openings to allow free passages of gases .The top 3-6 m of the checkers is made up of higher alumina bricks or semi-silica bricks while the remainders are of good quality firebricks.

In a commonly adopted design the products of combustion rise up wards in the combustion chamber and then flow downwards through the checker only once. This is called two-pass design. There are other designs, in one of which the checkers are divided into two chambers and the gases have to pass once downwards and then upwards before leaving the stove. Such a design is known as three pass design. Modern stoves



Hot blast stove design with respect to location of the combustion chamber.

(a) side-combustion type. (b) central-combustion-type

invariably have two-pass design. Stoves differ in yet another major feature i.e. the location combustion chamber. In one type the combustion chamber is located at the centre and the checkers are all around it in an annular shape. This is called central combustion type. In the other type the combustion chamber is one side and oval in cross section. This latter design is more popular than the former and is known as side combustion type.

Gas Cleaning Units

The modern blast furnace produces an effluent gas containing a significant proportion of carbon monoxide. Although various uses of blast furnace gas, including recovery of by products, have been suggested, it has not been possible to use the gas for any other purposes other than just a lean fuel.

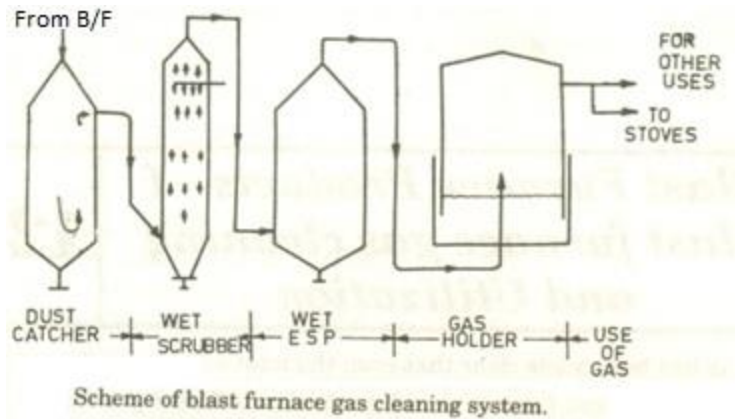
Problems associated with blast furnace gas

- Contains a significant amount of dust.
- May lead to accumulation of dust and wear in the equipment.

The gas is therefore cleaned before use, but sensible heat is lost.

The chemical heat of the cleaned gas is what is utilized.

In general cleaning is carried out in three stages viz. coarse, semi-fine and fine cleaning. A typical scheme is shown in the figure. Wet methods are generally preferred to dry methods for their better efficiency and smooth working.



3 stage gas cleaning system

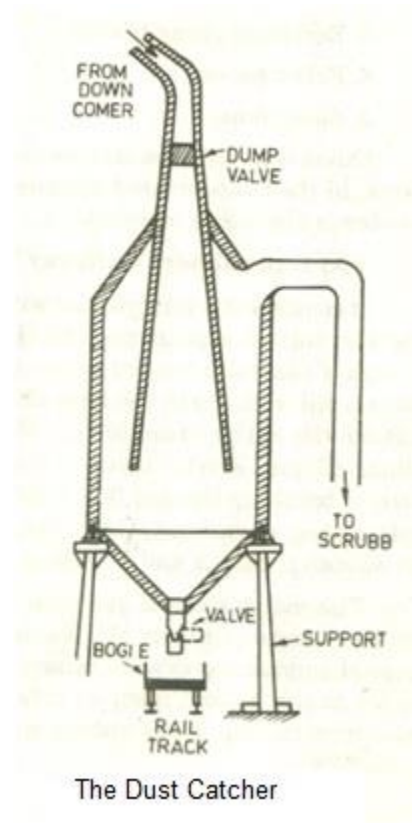
1. Coarse Cleaning (Dust Catcher)

- It operates on the principle that the dust is contained in the gas by the virtue of its velocity, and hence, if the velocity is zero the dust particle will be eliminated from the gas.
- The state of “**zero velocity**” is achieved in the dust catcher by simply reversing the direction of the gas flow, from vertically downward to vertically upward direction
- Efficiency is usually 60-75%
- Coarse sized particles are cleaned.

2. Semi-fine or primary cleaning

The gas from the dust catcher is led into primary or semi-fine gas cleaning system which consists of any one of the following units

- Scrubbers or spray towers
- Ventury washer
- Revolving spray-towers
- Feld washers

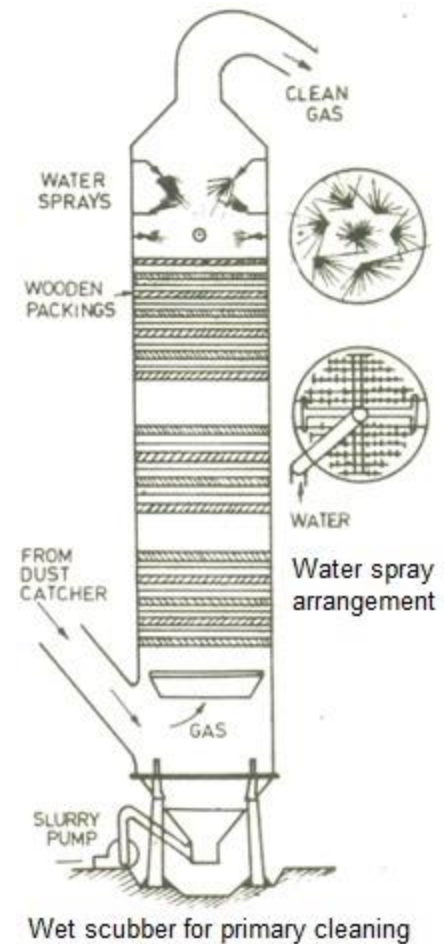


- Spray fans

Out of these, spray-towers are most commonly used

- **Scrubbers or Spray towers**

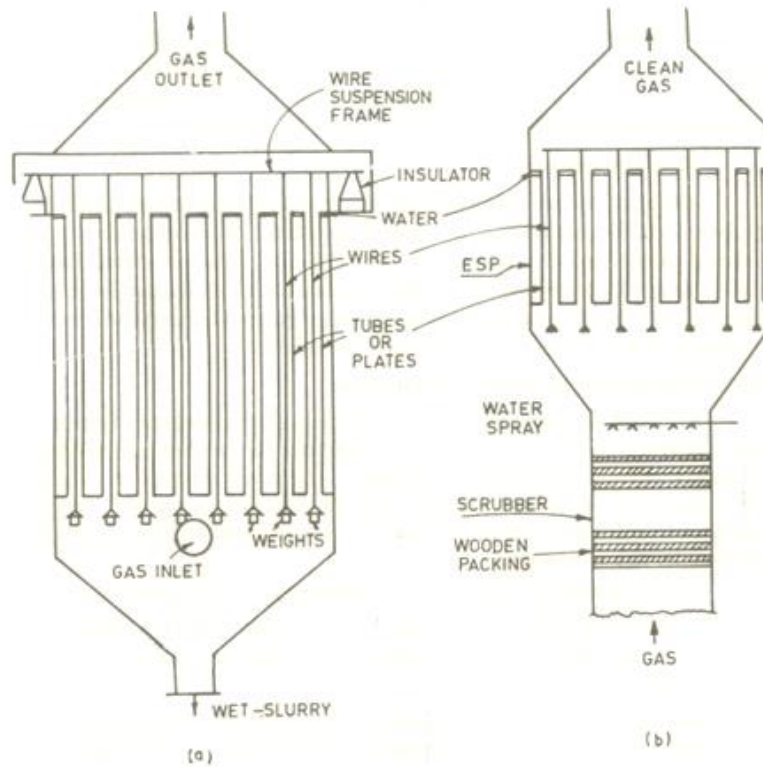
- Consists of a tall cylinder with conical top and bottom.
- The gas enters in this at a point near the bottom of the cylinder and leaves through a centrally located outlet at the top as shown in the diagram
- The coarse cleaned gas's velocity is reduced considerably because of sudden increase in volume and as a result some coarse dust settles at the bottom itself as it becomes wet
- Finer dust is washed down from the top.
- The gas is also cooled to around 30-35° C.
- Nearly 80-90% of the dust contained in the incoming gas is removed by this method with about 3.5-4.5 litres of water per cubic metre of gas.



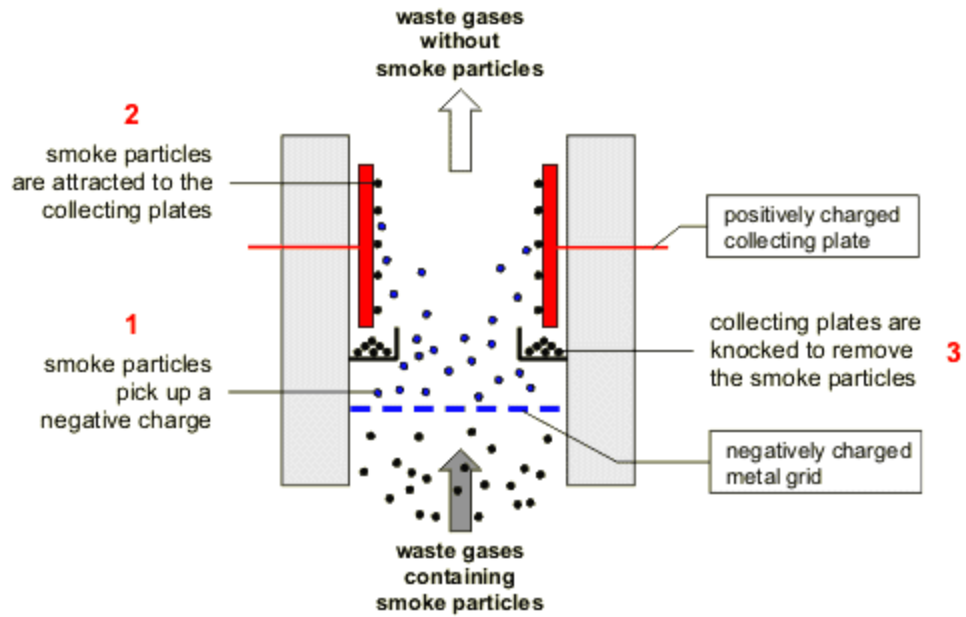
3. Fine or secondary cleaning

- Carried out in both wet and dry conditions
- Wet – type cleaners are either electrostatic precipitators, high speed disintegrators or Theissen disintegrators.
- **Electrostatic Precipitator**
 - It is essentially based on the principle that under an action of high applied voltage the dust particles in a gas phase acquire electrostatic charge and are attracted to an electrode of opposite polarity where they are collected and washed out
 - The corona effect also helps this behaviour
 - The dust particles on ionisation get attracted to these wet surfaces and are washed down.
 - The precipitator is usually divide into two zones, one of which is working while the other is under repair, if required.

- The ESP is used widely for cleaning the dry effluent gases before they are left off into the atmosphere as per the pollution control legislation.
- Total dust is cleaned out.



Wet ESP used as an independent unit (a) or in combination with scrubber(b)



Conclusion:

In spite of all the improvements in blast furnace processing the effluent gas still contains carbon monoxide. Therefore a fairly efficient cleaning system is devised and after cleaning of the gas is used primarily as blast in blast furnace and the rest is used somewhere else as lean fuel.

LECTURE-12

Refractories used in blast furnace and stove

Introduction:

Refractory is a material having the ability to retain its physical shape and chemical identity when subjected to high temperatures. **Refractory** bricks of various shapes are used in lining of blast furnaces and hot blast stoves. Refractories are inorganic, nonmetallic, porous and heterogeneous materials composed of thermally stable mineral aggregates with a binder phase and additives. Refractories are heat resistant materials used in almost all processes involving high temperatures and/or corrosive environment. These are typically used to insulate and protect industrial furnaces and vessels due to their excellent resistance to heat, chemical attack and mechanical damage. Any failure of refractory could result in a great loss of production time, equipment, and sometimes the product itself. The various types of refractories also influence the safe operation, energy consumption and product quality; therefore, obtaining refractories best suited to each application is of supreme importance.

Requirements of Right Refractory

The general requirements of a refractory material can be summed up as:

- 1) Its ability to withstand high temperatures and trap heat within a limited area like a furnace;
- 2) Its ability to withstand action of molten metal , hot gasses and slag erosion etc;
- 3) Its ability to withstand load at service conditions;
- 4) Its ability to resist contamination of the material with which it comes into contact;
- 5) Its ability to maintain sufficient dimensional stability at high temperatures and after/during repeated thermal cycling;
- 6) Its ability to conserve heat.

1) Classification Based on Chemical Composition

Refractories are typically classified on the basis of their chemical behaviour, i.e. their reaction to the type of slags. Accordingly the refractory materials are of three classes - Acid, Basic & Neutral.

Acid Refractories: Acid refractories are those which are attacked by alkalis (basic slags). These are used in areas where slag and atmosphere are acidic. Examples of acid refractories are:

- 1) Silica (SiO_2),
- 2) Zirconia (ZrO_2), and

Neutral Refractories: Neutral Refractories are chemically stable to both acids and bases and are used in areas where slag and atmosphere are either acidic or basic. The common examples of these materials are:

- 1) Carbon graphite (most inert)
- 2) Chromites (Cr_2O_3)
- 3) Alumina

Out of these graphite is the least reactive and is extensively used in metallurgical furnaces where the process of oxidation can be controlled.

Basic Refractories: Basic refractories are those which are attacked by acid slags but stable to alkaline slags, dusts and fumes at elevated temperatures. Since they do not react with alkaline slags, these refractories are of considerable importance for furnace linings where the environment is alkaline; for example non-ferrous metallurgical operations. The most important basic raw materials are:

- 1) Magnesite (MgO) - caustic, sintered and fused magnesite
- 2) Dolomite ($\text{CaO}\cdot\text{MgO}$) - sintered and fused dolomite
- 3) Chromite -main part of chrome ore

Chemical characteristics of the furnace process usually determine the type of refractory required. Theoretically, acid refractories should not be used in contact with basic slags,

gases and fumes whereas basic refractories can be best used in alkaline environment. Actually, for various reasons, these rules are often violated.

2) Classification Based on Method of Manufacture

The refractories can be manufactured in either of the following methods:

- a) Dry Press Process
- b) Fused Cast
- c) Hand Molded) Formed (Normal, Fired or chemical bonded)
- e) Unformed (Monolithic – Plastics, Ramming mass, Gunning, Cast able, Spraying

3) Classification Based on Physical Form

Refractories are classified according to their physical form. These are the *shaped* and *unshaped* refractories. The *shaped* is commonly known as refractory bricks and the *unshaped* as “monolithic” refractories.

Shaped Refractories:

Shaped refractories are those which have fixed shaped when delivered to the user. These are what we call bricks. Brick shapes maybe divided into two: standard shapes and special shapes.

Standards shapes have dimension that are conformed to by most refractory manufacturers and are generally applicable to kilns and furnaces of the same type.

Special shapes are specifically made for particular kilns and furnaces. This may not be applicable to another furnaces or kiln of the same type. Shaped refractories are almost always machine-pressed, thus, high uniformity in properties are expected. Special shapes are most often hand-molded and are expected to exhibit slight variations in properties.

Unshaped Refractories:

Unshaped refractories are without definite form and are only given shape upon application. It forms joint less lining and are better known as **monolithic** refractories.

These are categorized as Plastic refractories, ramming mixes, castables, gunning mixes, fettling mixes and mortars.

4) According to their refractoriness:

Brick	%age SiO ₂	%age Al ₂ O ₃	Other Constituents	PCE °F
Super Duty	49-53	40-44	5-7	3175-3200
High Duty	50-80	35-40	5-9	3075-3175
Medium Duty	60-70	26-36	5-9	2975-3075
Low Duty	60-70	23-33	6-10	2770-2900

Iron and Steel industry is the major consumer of refractories with about 70% of refractories being consumed by them. The different areas of blast furnace are exposed to different temperatures, slag and sulphur gases.

Special refractories:

Graphite Blocks

(Application Areas: blast furnace bottoms, hearths next to the cooling plate and the area of the copper cooling plate at middle and lower stack)

Graphite Blocks utilize low ash content, high quality petroleum coke and pitch coke as primary materials, along with high thermal conductivity additives. The bricks are molded under high pressure, impregnated and baked multiple times, graphitized and then machined to specification. Graphite Blocks feature excellent thermal conductivity that leads to better cooling while efficiently reducing lining erosion and hot metal infiltration.

Specifications

Bulk density g/cm ³	1.61 to 1.72
Apparent porosity %	24.70 to 16.5
Compression strength MPa	25.70 to 35.80
Bending strength MPa	7.80 to 12.50

Thermal conductivity (room temp.) W/m.k 42 to 128

Ash % 0.48 to 0.28

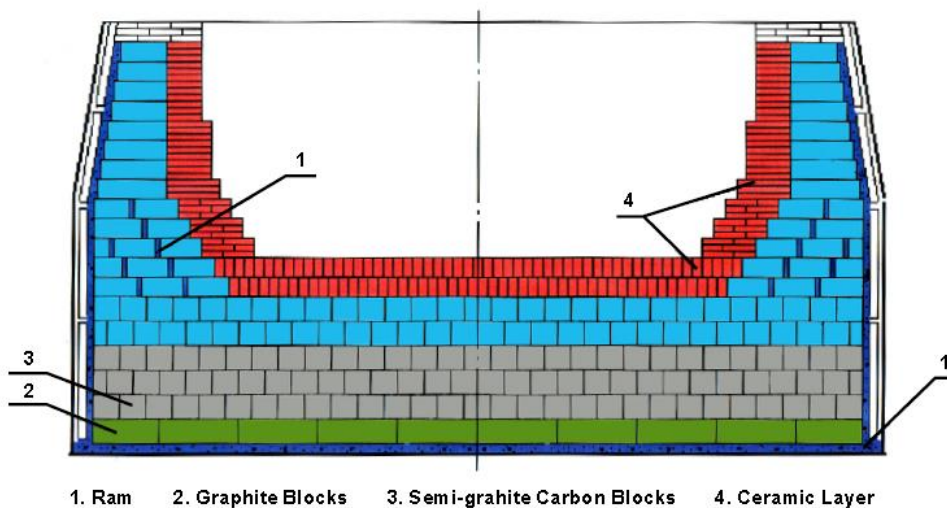
Semi-graphite Carbon SiC Blocks

(Application Areas: blast furnace slag formation area, hearth and bottom)

Semi-graphite SiC blocks use calcined anthracite coal and silicon carbide as primary materials, along with small amounts of other metal powders. The production process includes high-speed vibration, high-pressure compression, high temperature baking and machining to specification. The advantages of semi-graphite SiC blocks include good thermal shock resistance, good thermal conductivity and excellent slag erosion resistance.

Specifications	Units	Index	
Bulk density	g/cm ³	1.65	
Apparent porosity	%	15.97	
Crushing strength	MPa	35.82	
Bending strength	MPa	11.01	
Thermal conductivity	Room temp.	W/m.k	6.92
	300 °C		11.05
	600 °C		12.21
Permeability	mDa	18.50	
SiC	%	18	

1800m³ Blast Furnace Bottom and Hearth Structure



LECTURE-13

Blast Furnace Operations

BLOWING IN:-

The Process of Starting a B/F Operation is known as blowing in. There is no particular blowing in operation. It depends upon the local conditions and raw materials.

The blowing in operation is divided into four categories:

- (1) Drying
- (2) Filling
- (3) Lighting and operating until routine process in establishing

1. Drying:-

The newly lining B/F contains significant proportion of moisture. This must be slowly and completely removed before the operation of B/F. This process is also known as drying.

The B/F drying can be done by any one of the following three techniques:

- (1) Supply of hot blast from stove.
- (2) Use of wood and coke fire on the hearth.
- (3) Use of a dutch oven as an auxillary furnace to generate and supply hot gases.

2. Filling :-

At the end of drying, it depends upon the method of use for drying, the furnace is cleared off all the things used for filling.

Filling of the furnace means filling of hearth with light wood and oil upto the tuyers level and laying over this a scaffold of old timber slippers.

Coke is charged above the timber scaffold from the top upto the bosh level. Certain quantity of limestone charge along with the coke after the initial coke blank is set.

A small amount of old B/F is slag added with coke after the coke level rises beyond the mantle level.

Then the B/F is ready for ignition. Hence seeing the gravity of the process, this step should be carefully maintained.

3. Lighting and Operation until routine practice is established:-

After filling the furnace, the bells are opened and the dust catcher dump valve is closed. The air in the dust catcher is purged out with the help of steam.

The furnace is lighted either by inserting red-hot bars through the tuyers or slag hole and iron notch. Sometimes gas torch is also used. This kind of natural burning is continued for 24-36 hours.

Sufficient amount of gas produced from the furnace top, the bells are closed and the dust catcher dump valve is slowly opened to conduct gases through the gas cleaning system.

Tap holes are kept open for hot gases to escape out during the early period. Once coke burning and slag formation starts furnace crew are vigilant in observing the tap holes.

Sudden decrease in the outcoming gas through the tap hole is taken as an indication of beginning of slag production in the hearth and the tap hole is immediately closed thereafter. They are again opened after 3 to 8 hours

After two days, the ratio of iron ore increase in the burden.

After the 1st case is obtained a proper amount of slag and metal flowing out of the furnace is correlated to the charge schedule and proportion in order to assess the progress of blowing-in operation.

After the 1st case is over, charging and tapping schedules are established and strictly adhered to until routine production is established.

It takes about a week to establish normal routine process to obtain desired quality of metal and slag.

BANKING:-

Sometimes a furnace has to be shutdown for an extended period during campaign for variety of reasons like labour trouble, shortage of raw materials transport, strike or a breakdown in B/F.

During these troubles, the furnace has to be shutdown temporarily and these troubles of B/F should be adjusted in such a way that less amount of loss is occurred. This temporary shut down of B/F is known as “Banking”.

It means reduced combustion rate. This is done by taking some percentage of blast from the fire and covering the fire with excess coke.

The amount of blank coke charge depend upon the length of shutdown period.

The blank coke charge is followed by light charges, 50-60% of the normal burden.

The longer is the anticipated shutdown, the lighter is the burden.

The stock level is observed everyday when the furnace is banked. A slow downward movement of stock indicates slow consumption of coke. A rapid drop in the stock level can lead to deficiency of coke. At that time furnace is restarted.

The furnace is restarted by replacing the tuyers and notches and charging sufficient coke to fill up space created by sinking of the stock.

BLOWING OUT:-

The process of stopping the furnace operation at the end of its campaign is known as blowing-out.

It is accomplished in two ways.

- (1) Charging is stopped and stack is allowed to sink until minimum of it remains inside. As the stack sinks, the blast is reduced and the top of the stack is cooled by water sprays inserted through the top. The furnace becomes cold in about 2 hours.
- (2) In this method blank coke charge is followed by clean silica gravels of +25mm and - 50 mm size is used.

At the beginning, the stack line is kept at the normal level but laterally it is allowed to sink. The stack line temp. decrease by water sprays. The time required for blow-out is very short, about 6-8 hours.

BLOWING-DOWN:-

If business conditions deteriorate to the extent production is no longer required, the decision may be made to blow-out a furnace. Starting the furnace again under conditions approaching those of starting a new furnace is generally accomplished faster and with less effort than starting from a bank. However, the costs in connection with blowing out, raking out and cleaning preparatory to starting is likely to exceed the cost of banking. When a furnace has reached the end of its campaign (that is, the lining has worn out), it was usually blown-out except under most unusual circumstances. Modern practice is to blow the furnace down.

In the 1970s it became apparent to blast furnace operators that the procedures of banking and blowing-out blast furnaces were expensive. Often banking resulted in difficult startups because of water problems, or any number of unforeseen reasons. Also there was enough technical understanding of the process to allow operators to blow the furnace down. Blowing a furnace down means running the furnace without charging until the burden level in the furnace is reduced to approximately the tuyere level. There are several reasons why this technique is preferable to banking or blowing-out a furnace. Some are when a furnace is blown-down it can be thoroughly inspected for leaking coolers or staves, and no effort is required to stop air infiltration because the furnace is empty. And, if the furnace is to be relined it is faster and less expensive to tear apart an empty furnace than one containing burden.

The blow-down technique varies from operation to operation but basically the procedure is as follows. Atomizing water sprays are installed at the top of the furnace on a shutdown prior to the blowdown, somewhere in the vicinity of the 3–5 metres (10–15 ft.) stockline level. There are usually four to six spray nozzles installed equally spaced around the furnace. The purpose of the water sprays is to control the top temperature. Depending on the type of top and its condition, emergency water sprays are sometimes installed in the event of a grease fire during the blow down. Steam sprays are also installed on one to three levels to provide some cooling and to maintain furnace pressure. If one level is used it is generally placed low in the stack at roughly the 45–50 ft. stockline level. If additional levels are used they are placed roughly equidistant

between the top water sprays and the bottom steam sprays. The steam injectors are activated once the burden descends below them. Some companies will also run nitrogen to the steam sprays as backup in case the steam pressure would be lost. Nitrogen injection is provided for purging the furnace at the end of the blow down. Usually the nitrogen is introduced through the bustle pipe; if nitrogen is used as backup to the steam injection system, that system can also be used to purge the furnace when the blow-down is complete. Provision is made for process offgas analysis, often this simply requires recalibration of the top gas analyzer. Hydrogen and oxygen analyses are required. Hydrogen levels must be kept low (below 15%) and essentially no oxygen should be detected. Stockline measurement is required, and should be capable of extending well into the furnace.

The wind rate and hot blast temperature are reduced as the burden descends to control the top temperature and hydrogen and oxygen content of the top gas. Typically the maximum allowable top temperature is in the range of 315–425°C.

FANNING:-

Occasionally during the campaign of a blast furnace situations arise when the full productive capacity of the furnace is not required for a period of time. When this occurs, the problem can be solved by shutting the furnace down or curtailing the operation of the furnace by reducing the quantity of blast being blown. The blast/wind rate is usually reduced until the hot-blast pressure at the tuyeres is very low. A positive pressure must be kept in the hot-blast system to assure that there is no danger of gas coming back from the blast furnace into the blower system. The technique of reducing the wind volume to less than 20–25% of normal is known as fanning. Fanning has the advantages of keeping the gas system pressurized and furnishing a small quantity of blast furnace gas for use as fuel, and enables a resumption of near full operation on relatively short notice. This technique is used for emergency situations or short periods only. Prolonged use, such as eight hours out of every 24, or on weekends, often results in a hearth buildup and frequently promotes inwall scab formation.

BACK DRAUGHTING:-

Occasionally it is necessary to take the furnace off blast for short periods, often less than two hours, to perform various maintenance functions such as the replacement of tuyeres or repairing skip cables. In such instances, the furnace is not banked but is back draughted/ back drafted. In

this operation, as soon as the wind is stopped, the bustle pipe is put under negative pressure. This is done usually by opening the chimney valve and the hot-blast valve to a stove that has already been prepared by heating it to temperature and then shutting off the gas valve. As the furnace gas is drawn back into the stove, air is admitted through the peep sights and stove burner, and the operator makes certain that the gas burns in the stove. During the operation, the bleeders at the top of the furnace also are opened to pull some of the furnace gas out through the top. At several furnaces, a special back draft stack is installed so that it is not necessary to draw the furnace gas back through a stove. This stack is connected to the bustle pipe or to the hot-blast main. In some instances, it is closed by a water-cooled gate valve at the level of the bustle pipe and in others it is closed by an uncooled cap valve at the top of the stack. Opening the valve allows the furnace gas to draft to the atmosphere where it burns without difficulty.

TAPPING:-

The metal and slag are removed from the B/F periodically. The removal operation is generally known as “Tapping”.

The time interval between the two taps is also known as tapping time.

For production of 1 ton of pig iron, the amount of slag production is approximately 300-700 kg.

The density of slag is one-third of metal and hence the volume of slag is three times the metal of the same weight. Hence volume wise the slag volume will be nearly twice that of the metal in a commercial blast furnace.

It is generally seen that metal is tapped every 4-6 hours but tapping times for slag is 2 hours.

After tapping metal, they are collected in ladles. Then they are carried to the cast house for shaping them. Hence the ladles and cast house should always be ready for operation.

If the same molten pig iron is used for casting then the cast house should be nearer to the B/F.

DISPOSAL OF SLAG:-

When slag was considered as waste product produced to obtain useful molten pig iron. It was transported from the B/F to slag dumping yard in slag ladles.

The slag was tipped on the heaps in molten condition and was allowed to be air cooled.

This air cooled slag was used to make mineral wool, ceramic ware or used as filler material on Railways.

B/F slag as a raw material for the manufacturing of Portland cement.

For this the slag needs to be granulated. B/F slag as it flows out from the furnace through channels in the cast house is allowed to fall in stirred water held in a steel ladle and where it gets granulated.

In this form, it is easy to transport it to cement manufacturing sites and finally in cement making kiln.

LECTURE- 14

Material Balance

Example-1: In a furnace, iron ore is reduced according to the following reaction: $\text{Fe}_2\text{O}_3 + 6\text{CO} = 2\text{Fe} + 3\text{CO}_2 + 3\text{CO}$. Coke of composition 94% C is used to produce CO by combustion with air at the bottom of the furnace. Of the coke charged, 3.5% C is absorbed by iron and 90.5% C burns to CO only. No CO_2 is produced by combustion of coke.

Calculate:

- Volume of CO to produce 1000kg iron.
- Weight of coke required to produce 1000kg iron.
- Volume of air to burn the coke amount determined in (b)
- Volume and % composition of gases formed in combustion.
- Volume and % composition of gases resulting in combustion and reduction.

Solution:



Moles of CO = $\frac{6}{2} \times \frac{1000}{56} = 53.6$ kg moles.

a) Volume = $53.6 \times 22.4 = 1200\text{m}^3$ (1atm, 273k)

b) Coke required = $\frac{53.6 \times 12}{0.905} = 710\text{kg}$

volume of air $\text{C} + \frac{1}{2}\text{O}_2 = \text{CO}$

Moles of $\text{O}_2 = \frac{710 \times 0.905}{12 \times 2} = 26.8$ kg moles

Volume of air = 2858.7m^3

Gases formed in combustion:

CO -> 53.6 (34.7%)

N_2 ->100.8 (65.3%)

Gases formed in reduction:

CO -> 26.8 (17.35%)

CO₂-> 26.8 (17.35%)

N₂-> 100.8 (65.3%)

Example-2: Blast furnace produces pig iron of composition Fe 94%, Si 2%, Mn 0.5%, and C 3.5% by reduction smelting of iron ore, coke, and limestone. The analysis is as follows:

Iron Ore :Fe₂O₃ 78%, SiO₂ 8%, Al₂O₃ 5%, MnO 2%, H₂O 7%

Coke: 86% C and 10% S and 4% Al₂O₃. Amount is 600 kg per ton of pig iron.

Limestone: Pure CaCO₃ to produce a slag of 45% CaO

Calculate:

- Amount of ore/ton of pig iron
- % of total SiO₂ and of the MnO reduced in the furnace
- Amount of slag/ton of pig iron and its % composition.

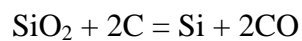
Solution:

Amount of ore = xkg

Fe in ore = Fe in pig iron

$$78 \times 112x / 100 \times 160 = 940$$

$$X = 1721.6 \text{ kg}$$



$$\text{Total SiO}_2 = 8/100 \times 1721.6 \text{ kg} = 137.7 \text{ kg}$$

$$\text{SiO}_2 \text{ reduced} = 31\%$$

$$\text{MnO reduced} = 18.75\%$$

$$\text{Si in ore} = \text{Si in slag} + \text{Si in pig iron}$$

$$\text{Si in slag} = 44.3 \text{ kg}$$

$$\text{SiO}_2 = 94.9 \text{ kg}$$

$$\text{Mn in ore} = \text{Mn in slag} + \text{Mn in pig iron}$$

$$\text{Mn in slag} = 21.76 \text{ kg}; \quad \text{MnO} = 28 \text{ Kg}$$

$$\text{Al}_2\text{O}_3 \text{ in slag} = 0.05 \times 1721.6 + 4/100 \times 600 = 110.1 \text{ kg}$$

$$\text{SiO}_2 + \text{Al}_2\text{O}_3 + \text{MnO} = 233 \text{ kg}$$

$$\text{Weight of slag} = 233/0.55 = 423.6 \text{ kg}$$

$$\text{SiO}_2 = 94.9 ; 22.4\%$$

$$\text{Al}_2\text{O}_3 = 110.1; 26 \%$$

$$\text{MnO} = 28; 6.6\%$$

$$\text{CaO} = 190.6; 45\%$$

Example-3: Haematite ore of 80% Fe_2O_3 is reduced in blast furnace using coke of 85% C. The reduction equation is: $\text{Fe}_2\text{O}_3 + m\text{CO} = 2\text{Fe} + 3\text{CO}_2 + n\text{CO}$. It is required to produce exit gas of composition $\text{CO}:\text{CO}_2=7:4$. The pig iron analyzes 94% Fe and 4% C and ignore rest.

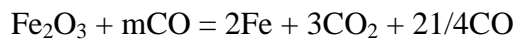
Determine:

- Reduction equation, balanced with whole numbers
- Amount of coke/ton of pig iron
- Amount of air required/ton of pig iron to burn C of coke to produce CO
- % composition of gas resulting due to combustion and reduction

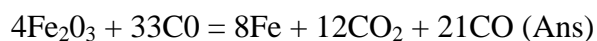
solution:



$$\text{CO}/\text{CO}_2 = 7/4 ; \text{CO} = 21/4$$



$$m = 3 + 21/4 = 33/4$$



Pig iron = 940 kg Fe = 16.8kg mole/ ton kg iron

CO needed = $33/8 \times 16.8 = 69.3$ kg mole / ton of pig iron

Carbon in pig iron = 40 kg

Total C = $40 + 69.3 \times 12 = 871.6$ kg

Coke = 1025 kg/ ton of pig iron

Amount of air = $69.3/2 \times 1/0.21 \times 22.4 = 3696\text{m}^3$; $\text{C} + 1/2\text{O}_2 = \text{CO}$

% Composition of gas

In reduction,

$\text{CO}_2 = 12/33 \times 69.3 = 25.2$ (by moles) , 36.4%

$\text{CO} = 21/33 \times 69.3 = 44.1$ (by moles), 63.6%

% Composition of gas on combustion

$\text{C} + \frac{1}{2}(\text{O}_2 + 3.76 \text{N}_2) = \text{CO} + 1.88 \text{N}_2$

$\text{CO} = 69.3$ (by moles); 34.79%

$\text{N}_2 = 130.3$ (moles); 65.29% (ans)

Self Assesment Problem: An iron blast furnace produces pig iron of composition C 3.5%, Si 1.5%, Mn 0.8%, and rest is iron. The burden of blast furnace consists of the following materials:

Iron ore: Fe_2O_3 76%, SiO_2 14%, Al_2O_3 9%, MnO 1%

Coke: C 88% and SiO_2 12 %; Amount 1100kg per ton of pig iron. Limestone: pure CaCO_3

The exit gas analyzes 26% CO, 13% CO_2 , and 61% N_2 . Assume no loss of iron in slag.

Required:

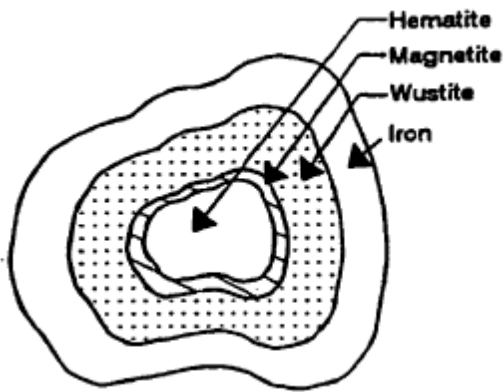
- Weight of iron ore/ton of pig iron
- Weight of limestone/ton of pig iron, required to produce slag containing 36% CaO
- Volume of gases
- Volume of blast based on O_2 and N_2 balance

LECTURE-15

Reduction of iron ores

Iron oxides may exist in three forms, i.e. hematite (Fe_2O_3), magnetite (Fe_3O_4) and wustite (FeO) depending on temperature and oxygen potential of the system. Although the non-stoichiometric wustite is usually written as FeO or Fe_xO , the actual oxygen content in wustite has a wide range from 23.1 to 25.6 wt%. The value of x in Fe_xO is less than unity and close to 0.95 when it co-exists with metallic iron.

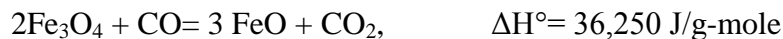
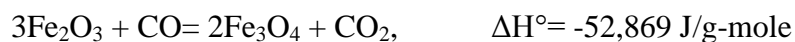
The reduction of hematite to produce iron takes place step-wise, with porous magnetite and wustite (at a temperature above 570°C) as intermediates as shown in Figure. At temperatures below 570°C wustite will not form.



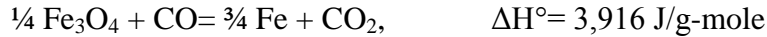
[Cross-section of a partially reduced dense iron ore particle showing topochemical type of reduction]

Equilibrium compositions between iron oxides and reducing gases:

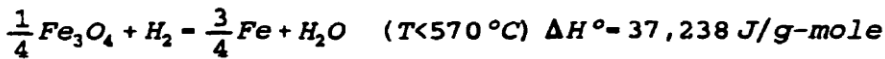
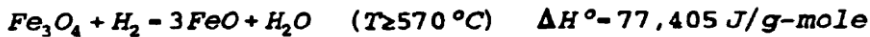
Carbon monoxide and hydrogen are the most common reducing agents used in commercial processes. Some thermodynamic data are given below as examples. Heat of reactions for the stoichiometric equations (per gram moles at 25°C) are included.



At a temperature below 570°C, magnetite is reduced directly to metallic iron,



The reduction of iron oxides by hydrogen and the heat of reactions at 25°C are listed below



The 1st reaction is exothermic in nature but rest reactions are endothermic.

Salient kinetic features that need to be noted are as follows:

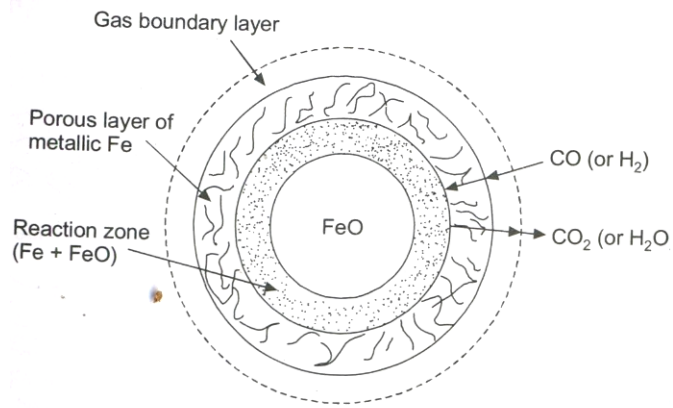
- Lump ore, sinter and pellets are porous solids.
- Reduction is characterized by the formation of a porous product layer.
- Fe₂O₃ is reduced in stages viz, Fe₂O₃ → Fe₃O₄ → Fe_xO → Fe
- Additional porosity develops during reduction owing to density differences of the product solids. The relative volume per unit mass of Fe are:

$$\text{Fe} : \text{Fe}_2\text{O}_3 : \text{Fe}_3\text{O}_4 : \text{FeO} = 1 : 1.79 : 2.08 : 2.14$$

As a result, when haematite is reduced to magnetite, additional porosity develops, enhancing further reduction of haematite.

- A measure of rate of reduction (r_0) is dF_0/dt , where t is the time after reduction starts.
- The higher the reducibility of an iron oxide bearing solid, the faster is the reduction of reduction.

Mechanism of Reduction:



Reduction of FeO in any ore takes place as in the diagram above

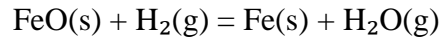
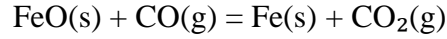
The reduction of a single oxide particle:

The monitoring of reduction kinetics of a single oxide particle is usually carried out by suspending a spherical sample from a balance in a flowing-gas stream of known composition and temperature for continuous measurement of weight loss. In order to analysis the weight-loss data, the experiment is usually designed for: (1) isothermal conditions; (2) single solid/ solid interface where gas/solid reaction takes place; and (3) dense specimens. A sketch showing the reduction of a dense wustite sphere by CO is given in Figure.

The system is made of three phases: the gaseous phase, the porous layer of solid product, and the un-reacted pore free solid reactant. The removal of oxygen from wustite takes place at a sharp iron/wustite interface, which shrinks parallel to the initial outer surface of the pellet, and results in weight-loss. The removal of oxygen proceeds through the following steps occurring successively during the reaction.

The kinetic steps involved are:

1. Transfer of reactant gas to the solid surface (CO or H₂) across the gas boundary layer around the piece of solid.
2. Inward diffusion of reactant gas through the pores of the solid.
3. Chemical reactions occurring are:



4. Outward diffusion of the product gas (CO_2 or H_2O) through the pores.
5. Transfer of the product gas from the solid surface into the bulk gas across the boundary layer.

Each step may contribute resistance to the completion of the overall chemical reaction. The slowest step among these five which are connected in series will provide the largest resistance to the overall reaction. If a step is much slower than all others so that resistances attributed to the other steps become relatively insignificant, this step is called rate-limiting or rate-controlling.

The rates of the above reactions are comparable. The overall rate depends on:-

1. Temperature
2. Gas Composition
3. Size of the particle
4. Nature of the solid in terms of its structure and composition.

The pattern of ore reduction has been classified into:-

I. Uniform internal reduction

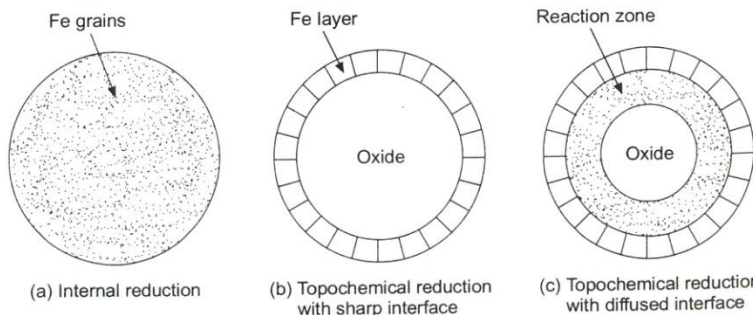
Chemical Reaction < Diffusion through pores

II. Topochemical reduction with a sharp interface

Chemical Reaction > Diffusion through pores

III. Topochemical reduction with diffused interface

Chemical Reaction \approx Diffusion through pores



Sketches of sections of partially reduced iron ore pellets, showing three patterns^S (Fe_2O_3 , Fe_3O_4 and FeO have not been shown separately).

N.B: In a blast furnace, stage wise reduction occurs throughout, i.e,

Upper part of stack → Internal Reduction

Lower part of stack → Topochemical Reduction with diffused interface

The overall kinetics of iron ore reduction is:

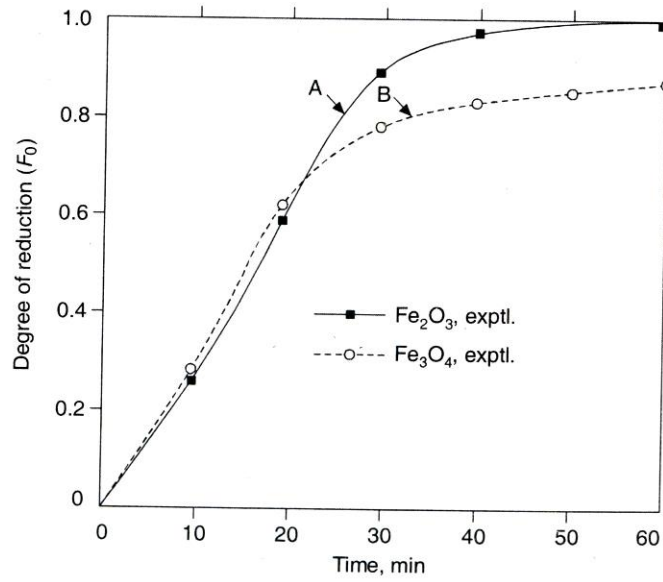
- Dependence of the rate on the structure of the solid, especially its porosity and pore structure.
- Changes in overall structure of the solid during reduction on account of swelling/contraction, recrystallization and grain growth.
- Comparable rates of chemical reaction, pore diffusion and in many cases, boundary layer mass transfer.

Standard Free Energy of formation (J/mol) of some compounds as a function of temperature

Reaction	Temperature, K	A, J/mol	B, J/mol	$\Delta G^\circ = A + BT$ (at 1500 K)
$C(\text{gr}) + (1/2)O_2(\text{g}) = CO(\text{g})$	1500 – 2000	-111700	-87.65	-243175
$C(\text{gr}) + O_2(\text{g}) = CO_2(\text{g})$	1500 – 2000	-394100	-0.84	-395360
$H_2(\text{g}) + (1/2)O_2(\text{g}) = H_2O(\text{g})$	1500 – 2000	-251877	58.33	-164382
$Fe(\text{l}) + (1/2)O_2(\text{g}) = FeO(\text{l})$	1809 – 2000	-238070	49.45	-163895
$Fe(\text{s}) + (1/2)O_2(\text{g}) = FeO(\text{s})$	298 – 1642	-259600	62.55	-165775
$3Fe(\text{s}) + 2O_2(\text{g}) = Fe_3O_4(\text{s})$	298 – 1642	-1102720	307.5	-641470
$2Fe(\text{s}) + (3/2)O_2(\text{g}) = Fe_2O_3(\text{s})$	298 – 1800	-814512	250.7	-438462

- Reduction by H_2 is 5-10 times faster than by CO . (Since, upward slope in Ellingham Diagram, i.e unstable).
- In the stack region, CO and H_2 can reduce only the oxides of iron.
- Degree of Reduction (F_0):

$$F_0 = \frac{\text{loss of mass of the ore due to removal of oxygen}}{\text{total mass of removable oxygen in ore}}$$

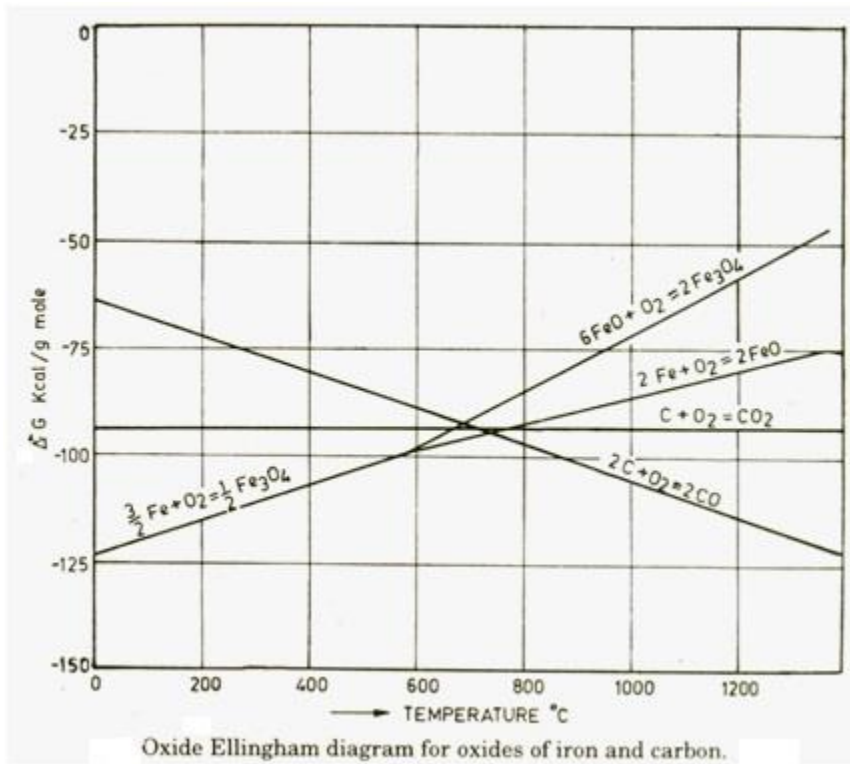


F_0 vs. t plots for reduction of Fe_2O_3 and Fe_3O_4 in H_2 at 900°C .

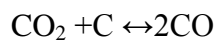
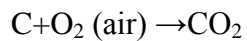
LECTURE-16

Fe-C-O and Fe-O-H Phase equilibria

Blast furnace is a counter current reactor where carbothermic reduction takes place using CO as reductant mainly.

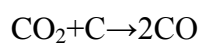


Boudouard Equilibrium



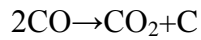
The second reaction is reversible one and depends upon the temperature and pressure of the process.

When temperature increases as shown in the figure CO becomes more stable and the above reaction shifts towards forward direction i.e.



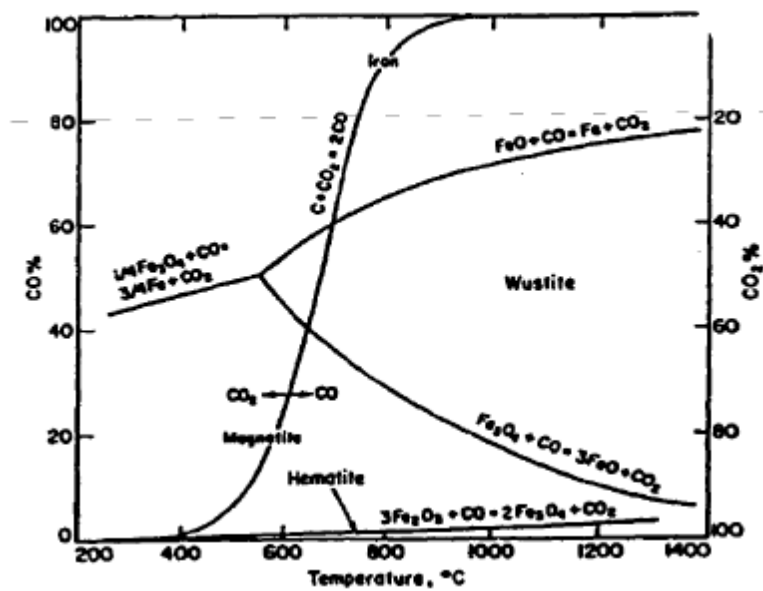
This forward direction reaction is called solution loss reaction or carbon gasification reaction which regenerates reducing agent. This reaction is highly endothermic and has a very large activation energy for chemical kinetics; therefore, it proceeds at an appreciable rate only at sufficiently high temperatures for a given reactivity of carbon.

When pressure increases the Boudouard reaction shifts towards backward direction i.e.



This backward direction reaction is called Naumann's reversion reaction which is responsible for carbon deposition at lower temperature zones of blast furnace.

The equilibrium phase diagrams (super-imposed with that for carbon-carbon oxides system) is shown in Figure



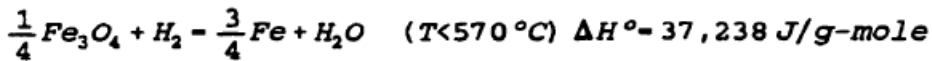
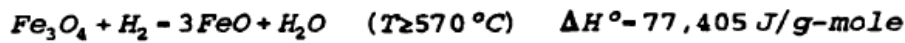
It is evident from the above figure that Fe_3O_4 is directly reduced to Fe below 570°C as follows



In the presence of solid carbon, the following reaction due to humidification of blast may restore the reducing power of local gas to sustain the reduction of iron oxides at the expense of heat.



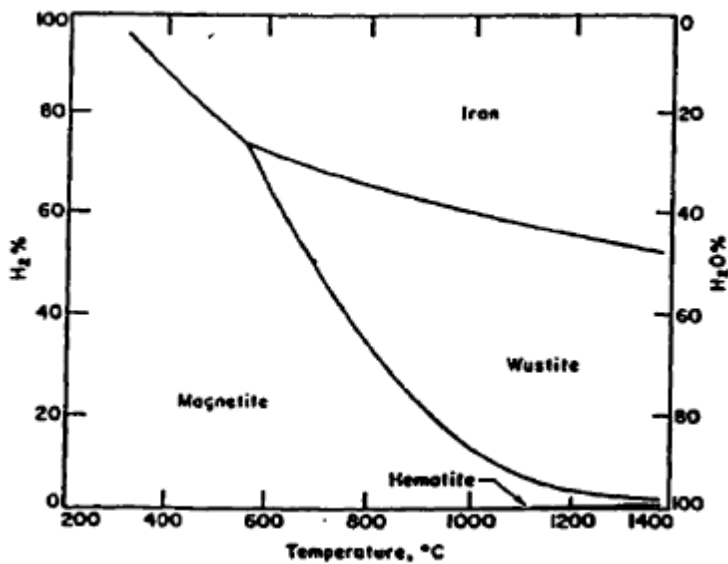
The reduction of iron oxides by hydrogen is endothermic for the following reactions. The heat of reactions at 25°C are listed below



Hematite to magnetite reduction is exothermic



The phase diagram of the H-O-Fe system is shown in Figure



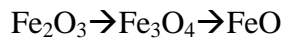
LECTURE-17 & 18

Blast furnace reactions

SIX DIFFERENT ZONES OF BLAST FURNACE UNDER WORKING STATE (top to bottom):-

1. Granular zone

It is the top most zone of blast furnace which contains the coke, iron ore and flux like limestone. The iron-bearing oxides charged gets reduced to wustite and metallic iron towards the lower end of the granular zone. Following reaction takes place



2. Cohesive zone

As the burden descends further, and its temperature rises on account of contact with the ascending hot gases, softening and melting of the iron-bearing solids takes place in the so-called cohesive zone (mushy zone). The cohesive zone has the lowest permeability. Further down the furnace, impure liquid iron and liquid slag are formed. The absorption of carbon lowers the melting point of iron drastically. For example, an iron alloy containing 4 wt. % carbon melts at only 1185°C..

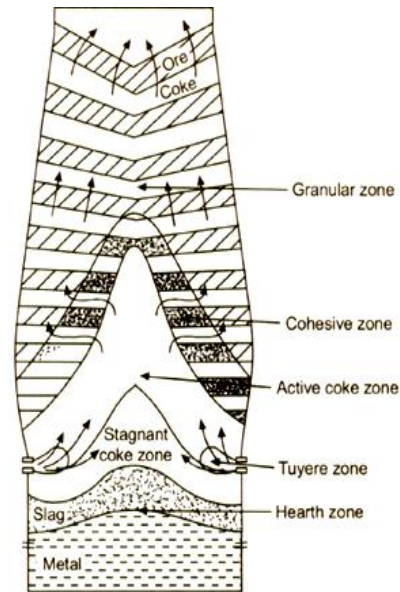
Also the melting of primary slag (containing more of FeO) starts from at this zone.

3. Active coke zone

In this zone reduction of FeO by C takes place from the slag and so slag composition starts to change from high FeO content to low FeO content.

4. Stagnant coke zone (or deadman zone)

Here the coke column either floats on liquid iron in the hearth region or reaches as far as hearth floor. Some slag and iron entrapped in the interstices of the coke pieces are drained out during tapping.



Schematic sectional diagram of the internal zones in a blast furnace.

5. Tuyere zone

Coke is the only material of the blast furnace charge which descends to the tuyere level in the solid state. It burns with air in front of the tuyeres in a 1-2 m deep raceway around the hearth periphery.

During combustion coke pieces swirl around before they get gasified, and then fresh pieces descend to take their place. The temperature is highest out in this zone of blast furnace. The continuous consumption of coke and the consequent creation of an empty space permit the downward flow of the charge materials.

- The combustion zone is in the form of a pear shape, called '**RACEWAY**' in which the hot gases rotate at high speeds carrying a small amount of burning coke in suspension.

Raceway

The raceway is a vital part of the blast furnace since it is the heat source in a gigantic reactor and at the same time a source of reducing gas.

The salient features of Combustion zone are summarized below:

- ▶ The force of the blast forms a cavity the roof of which is formed of loosely packed or suspended coke lumps and the wall more closely packed.
- ▶ The CO₂ concentration tends to increase gradually from the centre and reaches a maximum value just before the raceway boundary where most of the combustion of coke occurs according to:



- ▶ The temperature of the gas rises as the coke consumption proceeds and reaches a maximum just before the raceway boundary. Thereafter, it falls sharply as the endothermic reduction of CO₂ by C proceeds;



- ▶ The concentration of CO₂ falls; rapidly from the raceway boundary and the gasification is completed within 200-400 mm from the starting point of the reaction.

- ▶ Blast, injectants and coke are converted to hot reducing gas. This gas reduces the ore as it moves counter currently towards the top of the furnace.

RAFT 'Raceway Adiabatic Flame Temperature'

This is the highest temperature available inside the furnace. There is temperature gradient in vertical direction on either side of this zone. This temperature is critically related to the hearth temperature known as operating temperature of the furnace. It is equally related to the top gas temperature such that the hot raceway gasses have to impart their heat to the descending burden to the extent expected and leave the furnace as off-gases at the desired temperature.

RAFT Calculation

$C + 1/2 O_2 = CO$ Heat of combustion = 2300 kcal/kg.C

For normal blast containing 21% O_2 by volume and 79% N_2

Carbon+Blast=CO+Nitrogen

$C + (21\% \text{Oxygen} + 79\% \text{Nitrogen}) = CO + 79\% \text{Nitrogen}$

$C + 0.5(O_2 + 79/21N_2) = CO + 0.5*(79/21N_2)$

$C + 0.5O_2 + 1.88N_2 = CO + 1.88N_2$

Volume of Air required = $0.5 + 1.88 = 2.38$ moles per mole of Carbon

Heat of combustion of C to CO (kcal/kg C) + heat content of carbon (kcal/kg C) + vol. of air ($Nm^3/kg. C$)*heat capacity of air (kcal/ $Nm^3. ^\circ C$)* blast temperature ($^\circ C$) = heat capacity of gas(kcal/ $Nm^3. ^\circ C$)*RAFT

6. Hearth zone

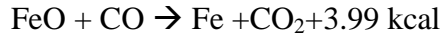
It is the zone where everything is liquid except coke bed which holds the liquid iron and slag upon it. The slag floats over the molten iron as slag lighter/less denser than molten metal. It is less viscos than molten metal.

REACTIONS IN UPPER ZONE:

1. REDUCTION OF IRON OXIDES:

Following indirect reduction reactions occurs in this zone of blast furnace





The extent of actual reduction depends upon the size and nature of ore and intensity of gas flow in the radial cross section of the zone.

2. VOLATILIZATION:

The water left in the coke, ore, limestone and other charge materials are very helpful for lowering the top gas temperature. Also water is deliberately added to decrease dust loss in the top gas for fine ores. The volatilization of water consumes about 380 kcal/kg of water.

A good variety of coke contain about 1.5-2 % of volatile matter. But volatilization of this doesn't affect much on the top gas composition.

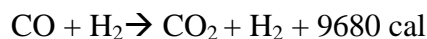
3. DECOMPOSITION OF HYDRATES:

Apart from its physical presence as moisture content in the burden it is also present as the water of crystallization or hydration. Heat energy is required for decomposition of hydrates.

Iron oxides lose the hydrates at about 300°C and $\text{Al}_2\text{O}_3 \cdot 2\text{SiO}_2 \cdot 2\text{H}_2\text{O}$ does so at much higher temperatures and so requires high amount of heat energy.

The physically combined water escapes as steam without contributing much to the reactions but the hydrated water which is released above 300-400°C may react with CO to produce CO_2 and H_2 .

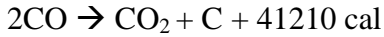
4. WATER GAS SHIFT REACTION:



The equilibrium constant of the reaction is 2.46 at 900°K and 0.8 at 1200°K. It is, therefore, thermodynamically possible for CO to reduce H_2O in upper furnace to certain extent while the reaction would be kinetically more favorable at middle region where temperature between 800-1000°C.

The reaction is influenced by catalysts and freshly reduced iron itself possesses catalyzing properties.

5. CARBON DEPOSITION :



Because of exothermic nature of the reaction, the formation of carbon should be preferred at low temperatures but the reaction velocity possesses the opposite tendency, i.e. it increases with increasing temperature. The deposition occurs predominantly in a narrow temperature range 450-600°C where the reaction velocity attains reasonable rate.

The presence of freshly porous Fe and its oxides catalyzes the reaction.

The deposited carbon consists of graphite and cementite. Carbon deposition ceases when most of the iron is converted into cementite.

The C settles in the pores of the charge descending and at high temperature direct reduction of iron oxide takes place to produce CO, a gas and this sudden expansion leads to breaking of lump and new surface are created for better reduction of ore.

The deposition being exothermic helps in the rapid heating of charge material.

But due to their fine size they might get lost in the high velocity of upward moving gas. The top pressure also helps in this respect by lowering the kinetic energy of the gas. On the other hand it will lead to more of deposition reaction to occur.

Disadvantages:

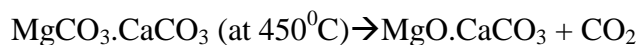
The C moving down with the ore when comes in contact of CO₂ leads to solution-loss reaction at relatively lower temperature as it would have been with C from coke. So, major portion of the reducing gas is lost and demand for more of CO leads to more of fuel consumption.

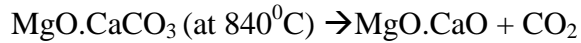
It is also deposited on the brick works of the shaft and can cause its disintegration, dislocation as well as formation of bridges, accretions, etc.

6. DECOMPOSITION OF CARBONATES:

Metal carbonates with the ore as well as the flux which are deliberately to the blast furnace charge are source of CO₂ formation.

Flues added such as dolomite (MgCO₃.CaCO₃) releases CO₂ as follows



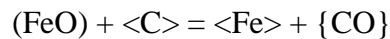


which all occurs in the stack.

The dissociation pressure (p_{CO_2}) of carbonates increases with increases temperature, the decomposition temperature being that when $p_{\text{CO}_2} = 1 \text{ atm}$.

REACTION IN BOSH

Here direct reduction of iron oxide (FeO) takes place as follows:



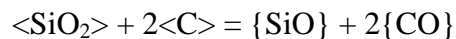
The iron reduced is still in solid form because of its high melting point. But it starts picking of carbons a trickles down through the bosh region and its melting point thereby decreases and its becomes molten.

The initial slag of high FeO content is acidic but it picks up CaO and achieves the desired basicity and viscosity as it passes through the cohesive zone.

P_2O_5 , SiO_2 , Cr_2O_3 , and MnO which are part of slag get reduce in this region.

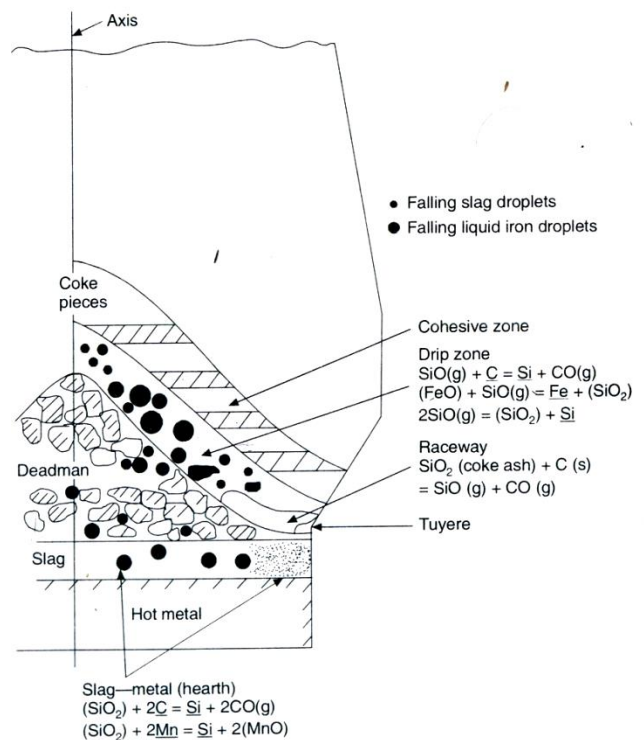
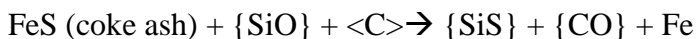
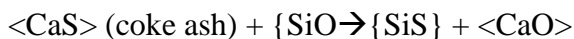
The iron is carburized it melts and picks up Si, S, Mn, Cr, and P as per the extent of reductions

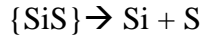
SiO gas is formed during the burning of coke as per the following reaction:



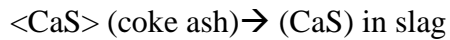
Which during its ascent contribute to *siliconisation* of metal.

Sulphur is released from the fuel on its burning. Following mechanism is followed by sulphur transfer:





And partially bosh slag also absorbs sulphur as



The sulphur in the iron melt decreases and for slag increases during the passage through the bosh zone upto the slag surface in the hearth.

QUESTION: Why is the basicity of bosh slag higher than that of hearth slag?

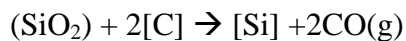
REACTION IN HEARTH:

Hearth is the part of the furnace where molten part of blast furnace process accumulate and stay there until these are tapped out.

Here only two separate layers of slag and molten metal are formed. Slag is above the molten metal because of its lower specific gravity.

Since the volume of slag is twice that of molten metal, it is tapped twice as compared to molten metal periodically.

For silicon, the slag metal reaction is:



For which, $K_{31} = \frac{[\%Si][f_{Si}]}{(a_{SiO_2}) \cdot (p_{CO}/a_C)^2}$

It has been further observed that the Si-Mn reaction, which is a slag metal reaction, also occurs in the blast furnace hearth i.e.



LECTURE-19

Blast furnace slag

Slag is the molten oxide by-product of smelting process.

When raw materials viz. iron ore, sinter, pellets coke and different types of fluxes are charged in a blast furnace, hot metal is produced and the impurities in the charge get separated as slag. Thus, slag is a product formed by chemical combination of fluxes (usually lime and magnesia) with aluminates, silicates and other gangue constituents in iron oxide and coke ash.

Slag = Gangue + Flux

The molten slag is insoluble in hot metal and is lighter than the liquid iron, resulting in a separate layer of its own. Basic flux is essential to lower the melting points of the acid constituents and thus obtain a liquid and sufficiently fluid slag. Basic flux also takes part in the slag-metal reactions and controls the metal quality. Apart from the four main non-ferrous oxides (lime, magnesia, alumina and silica) which form about 95-96% of the total, there are other compounds present in minor quantities depending upon the raw materials and the type of iron smelted in the blast furnace viz. FeO, MnO, FeS, CaS, alkali silicates, etc.

Functions of Blast Furnace Slag

A proper slag should perform various functions in order to achieve maximum fuel efficiency and maximum furnace performance.

1. The slag should have low viscosity so that it flows freely and rapidly at operating temperatures. The coke rate influences this flowing temperature and hence this affects the eventual fuel economy of the process.
2. The slag should facilitate free movement of stock and furnace gases.
3. It should also have a high sulphur retention potential i.e. (S)/[S].
4. The slag also controls the metal temperature, metal quality and homogeneity.

5. The slag should tolerate and cushion the fluctuations in the temperature and physical and chemical properties of the burden.
6. Finally, the slag should be suitable for use in cement manufacture, road ballast, etc.

On the other hand, the formation of slag is a subject to the operating conditions and other factors like **flame temperature, distribution of materials at the top, ore reducibility, burden yield, sinters and pellets, softening temperatures of the ferrous charge, coke ash content, furnace lines**, etc. When all the operating conditions are conducive to a smooth and regular furnace movement, the use of a slag of optimum composition may result in a maximum production and minimum of slag bulk, metal-sulphur and fuel costs.

It is generally accepted that for smooth blast furnace operation, the basicity ratio, **CaO/SiO₂ or (CaO + MgO)/SiO₂, should be maintained at about 1.2 or 1.4** respectively and the alumina content of the slag should not exceed 18% and neither fall below 12%. High alumina slags have high viscosity, which hampers desulphurization. In such a situation, MgO is normally added to reduce the viscosity but it results in an increase in slag volume which lowers the furnace productivity and increases the coke rate. A study of the viscosities of blast furnace slags reveals that acceptable viscosities are obtained with either **5% or 10% MgO in the slag, for slags containing up to 20% alumina.**

The sequence of slag formation in a blast furnace can be divided into three zones:

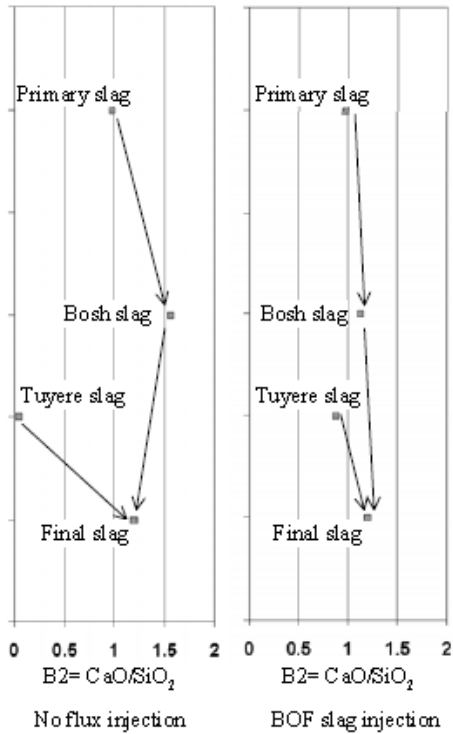
- (a) Primary slag fusion zone
- (b) Bosh slag fusion zone
- (c) Final slag or hearth slag zone

In raw ore and limestone burden, the calcinations and assimilation of the latter occur slowly and are complete only when very high temperatures are reached, i.e. in the bosh, tuyere and hearth zones. **The slag formed in the cohesive zone is the primary slag formed with FeO** as the

primary fluxing constituent; Thus, the initial slag formed is acidic in nature as it consists mostly of unreduced FeO and gangue silica and alumina with some MnO, the fusion temperature depending on the FeO content. If the ore is lean and highly reducible, the FeO content in the slag is less and the fusion temperature is high (1200-1350°C) and if the ore is rich with low slag bulk, the FeO content is comparatively higher even if its highly reducible and the fusion temperature is low (about 1200°C). The %FeO decreases due to direct reduction and the fusion temperature rises as the primary slag flows down. The presence of MnO and absorption of lime and magnesia keep the slag molten during its passage to the hearth.

The absorption of lime may be slow but it is continuous and is complete only in the hottest parts, i.e. at the tuyere level. The reduction of FeO in the lower, hotter zones is very fast and is almost complete above the tuyeres. The resulting bosh slag is much more basic than the final slag because the acid constituents of the coke ash are only absorbed when released at the tuyeres. Bosh slag composition may vary widely depending on the furnace movement as well as on the individual materials charged, namely, iron ore, manganese ores, sinters or fines, or limestone. Iron oxide is usually low (5 percent) but maybe as high as 10-20 percent. The bosh slag basicity is higher with richer ore, greater coke rate, greater sulphur load, greater metal-silicon, and greater coke ash content.

The hearth slag is formed on dissolution of lime which was not incorporated in the bosh and on absorption of coke ash released during combustion. The formation is more or less complete in the combustion zone. This slag runs along with the molten iron into the hearth, accumulates there and forms a pool with the liquid metal underneath. During the passage of iron droplet through the slag layer, the slag reacts with the metal and a transfer of mainly Si, Mn and S occurs from or to the metal, tending to attain equilibrium between themselves. The nature, composition and bulk of the hearth slag ultimately control the composition of the pig iron and the productivity of the blast furnace. The most important purpose of slag control is the control of sulphur, since sulphur is the key to iron quality. The use of optimum slag composition would give minimum sulphur in metal and minimum slag volume, resulting in maximum production and minimum fuel costs.



Basicity of slags at different locations in the blast furnace.

The primary slag is formed from the gangue content in the pellets. The bosh slag is considered to be formed by the primary slag, all additives and fluxes and some ash from the coke that is consumed in the shaft by solution loss reaction. The ash released in the raceway, when coke and coal are combusted, forms the tuyere slag, which also contains the flux constituents, when they are injected. The bosh slag and tuyere slag are mixed and after some additional reduction the final slag is formed. As can be seen, the excessive basicity of the bosh slag and the formation of a very acid tuyere slag can be avoided, by the injection of fluxes.

Slag Viscosity:

Slag viscosity is a transport property that relates to the reaction kinetics and the degree of reduction of the final slag. Slag viscosity also determines the slag-metal separation efficiency, and subsequently the metal yield and impurity removal capacity. In operation, the slag viscosity is indicative of the ease with which slag could be tapped from the furnace, and therefore relates

to the energy requirement and profitability of the process. The slag viscosity is sensitive to its ionic and molecular structure.

The blast furnace slag should be a **free flowing** one at the operating temperatures to ensure efficient slag-metal separation. It must have appreciable affinity for the gangue constituents to facilitate the production of metal of choice. Yet, from the available heat (in the hearth) point of view the blast furnace slag should be neither very viscous nor very fluid. The viscosity of the blast furnace slag governs the reaction rates in the furnace by its effect on the diffusion of ions through the liquid slag to and from the slag metal interface. It also affects the operation efficiency of the blast furnace by its influence on the aerodynamics of the furnace since the flow pattern of the molten slag significantly influences the gas permeability and heat transfer in the furnace.

A process of **depolymerisation** lowers the viscosity of the slag. **An increase in basicity decreases the viscosity** of the blast furnace slag breaking the three dimensional silicate network into discrete anionic groups thereby causing depolymerisation. However, the blast furnace slag viscosity cannot be described only by the enhanced degree of depolymerisation. Beyond certain level of basicity the viscosity actually increases, despite the enhanced degree of depolymerisation as such an increase in the basicity may also result in an increase in the chemical potential of some primary solid phases.

Slag Liquidus Temperature:

The liquidus temperature influences the effective positioning and width of the cohesive zone which directly controls the productivity of the blast furnace operation. The positioning and width of the cohesive zone controls the gas permeability, Si content in the hot metal and the extent of indirect reduction inside the furnace. Thus the determination of liquidus temperature of the slag is of vital importance.

The liquidus temperature is the temperature at which the first crystal forms during cooling down the melt or the temperature at which the last crystal disappears during heating.

In industrial melts, *i.e.*, when a pyrometallurgical process is carried out in a furnace, it is the fusion behaviour of the non-metallic melts (*i.e.*, the mixture of oxides which would combine to make the slag) which is more important than the exact fusion temperature. The fusion behaviour is described in terms of four characteristic temperatures.[5] These are; the initial deformation temperature (IDT), symbolising the surface stickiness; the softening temperature (ST), symbolising the plastic distortion; the hemispherical temperature (HT), which is also the liquidus temperature, symbolising sluggish Flow; and the flow temperature (FT), symbolising liquid mobility. Industrial Standards 51730 has been defined as the hemispherical temperature, *i.e.*, the temperature at which a small mass of the slag assumes the shape of a hemisphere on heating. The heating microscope method is adopted to measure this hemispherical temperature.

Shortness of a Slag:

The softening and melting phenomena of blast furnace slag is of vital importance in BF operation as it has a direct say over the productivity and efficiency of the process. The characteristic temperatures obtained from the flow characteristics of blast furnace slag includes; Initial Deformation Temperature (IDT), Softening Temperature (ST), Hemispherical Temperature (HT) and Flow Temperature (FT). The ST and FT rheologically signify the plastic distortion and liquid mobility of the blast furnace slag respectively. The difference between the flow temperature and softening temperature is termed as the 'shortness' of the slag. In a blast furnace operation, a short-slag is desirable as it trickles down the furnace as soon as it is formed thus exposing further reaction sites for better slag-metal reaction rates. Shortness of the slag has a direct impact over the furnace productivity as it controls the slag-metal reaction rates.

Alumina Problem

Often Al_2O_3 is regarded as acid oxide but the behavior of alumina is more complex, although it forms only two minerals which are really aluminosilicates of lime. These are:

1. Anorthite ($\text{CaO} \cdot \text{Al}_2\text{O}_3 \cdot 2\text{SiO}_2$)- Base (Replaces one molecule of CaO)
2. Gehlinit ($2\text{CaO} \cdot \text{Al}_2\text{O}_3 \cdot \text{SiO}_2$)- Acid (Replaces one molecule of SiO_2)

In both the cases the minerals contain 37% Al_2O_3 but the CaO/SiO_2 ratio is 0.47 and 1.87 respectively. Therefore when:

CaO/SiO_2 ratio is low, Anorthite ($\text{CaO} \cdot \text{Al}_2\text{O}_3 \cdot 2\text{SiO}_2$) is favoured

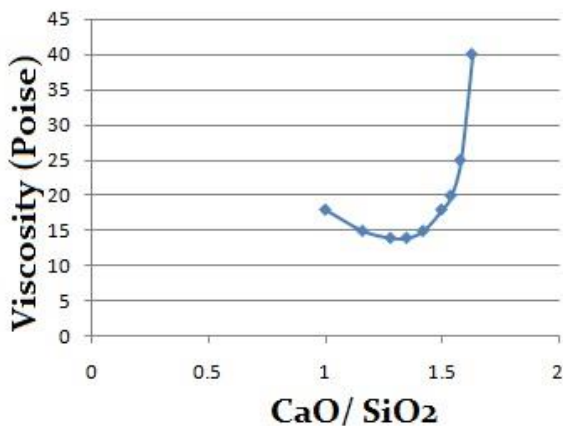
CaO/SiO_2 ratio is high, Gehlinites ($2\text{CaO} \cdot \text{Al}_2\text{O}_3 \cdot \text{SiO}_2$) is favoured

When the Al_2O_3 content of a slag is between 15-18% it can be regarded as a neutral constituent and ignored while calculating the burden.

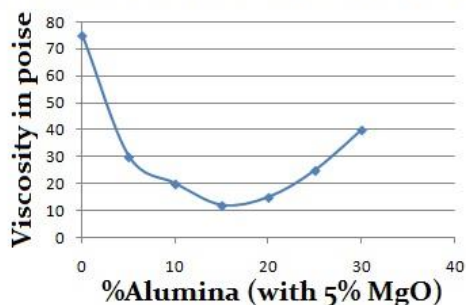
When less than 15% it behaves like an acid. As Al_2O_3 falls below 10% large quantities of sesquisilicate ($3\text{CaO} \cdot 2\text{SiO}_2$) are formed which affects the melting point and viscosity of slag to a marked degree.

If the Al_2O_3 content exceeds 18% then the Al_2O_3 acts as a base and a reduction of CaO in burden is necessary.

Viscosity Vs Basicity at 1400 Degree C



Effect of Alumina on Slag viscosity at 1400° C



Composition of minerals formed in blast furnace slags ratio

Mineral	Composition	Analysis in%				Ratio (CaO+MgO) SiO ₂	Ratio CaO/SiO ₂	Melting Point in °C
		SiO ₂	Al ₂ O ₃	CaO	MgO			
Calcium Ortho-Silicate	2CaO.SiO ₂	34.88	*	65.12		1.87	1.87	1700
Calcium Sesquisilicate	3CaO.2SiO ₂	41.66	*	58.34		1.40	1.40	1700
Calcium Bi- Silicate	CaO.SiO ₂	51.72	*	48.28		0.93	0.93	1530
Anorthite	CaO. Al ₂ O ₃ .2SiO ₂	43.19	36.65	20.16		0.47	0.47	1540
Gehlinit	2CaO. Al ₂ O ₃ .SiO ₂	21.91	37.18	40.91		1.87	1.87	1580
Akermanite	2CaO. MgO.2SiO ₂	44.07	*	41.14	14.79	1.27	0.93	1350- 1400
Monticellite	CaO. MgO.SiO ₂	38.39	*	35.84	25.77	1.6	0.93	1500

LECTURE- 20

Blast Furnace Performance and Control

The principle reason for the improvement of blast furnace performance includes:

- The increase in the average inner volume of blast furnace.
- The increase in the productivity (per unit volume) of the furnace.

The increase in inner volume of blast furnace is accomplished by increase in hearth diameter and not the height.

$$Productivity = \frac{Rate\ of\ production\ (R)\ (THM\ per\ day)}{Working\ volume\ (m^3)}$$

Productivity increases from 1.5 to 3.0

Efficiency:

- Coke/fuel rate (Lower is the coke rate the better is the furnace performance)
- Silicon content(Silicon content should be low from about 1% to 0.3%)
- Campaign (Lining to relining of a f/c is called its campaign. This should be longer)
- Fettleing (Temporary shutdown/repair of a f/c. It should be minimum)
- Irregularities (Random fluctuations by statistical variations in the composition and properties of input material)

Process variables:

In a modern blast furnace the following variables may be measured for effecting process control:

- 1) Stockline level with respect to certain fixed point.
- 2) Charging chute position, angle, rotation and material flow rate.
- 3) Top off gas temperature, pressure, flow rate and composition.
- 4) Charge material weight, rate of addition, moisture analysis, etc.
- 5) Blast furnace pressure drop.
- 6) Temperature distribution over the circular cross section at the top.
- 7) Shaft gas temperature distribution.
- 8) Stave and lining temperatures.

- 9) Hot blast temperature and pressure.
- 10) Slag weight and rate.
- 11) Tuyere cooling water temperatures and water volume throughout.
- 12) Blast volume, humidity content and oxygen enrichment.
- 13) Amount of injectants blown through tuyeres.
- 14) Hot metal tap rate, its analysis, particularly silicon content.
- 15) Hearth side wall and bottom temperatures.
- 16) Temperatures at any point inside the furnace.
- 17) Cooling water volume, its outlet temperature at all cooling plates.

There are independent as well as dependent variables:

Independent variables

Physical and chemical characteristics of raw materials as follows:

1. Size distribution

Fines are detrimental to furnace operation no matter where they are generated. The actual size of the ore used as a burden depends on its crushing strength, reducibility, etc. In deciding size of any charge material the relative size of ore, agglomerate, coke, flux, etc have to be taken into account to obtain best performance of the furnace.

2. Agglomeration

The remarkable improvement in furnace performance achieved by using sized lumpy ore as charge led to the development of agglomeration of ore fines as a routine method of burden preparation. Sintering also provides pre reduced ores.

3. Reducibility of iron ore

Reducibility parameters illustrate the possibility for removal of oxygen combined with Fe from the iron ore. A high reducibility is desirable, since it decreases the amount of reducing gas required to realise a given amount of metal from a given amount of ore. The rate at which the iron oxide can be reduced to metallic iron is the primary determinant of the rate of production in any reduction process

4. Reduction degradation index (rdi)

Lump ores, sinter and pellets disintegrate into smaller pieces during their downward travel through the blast furnace owing to the weight of the overlying burden, as well as abrasion and

impact between the burden materials. This tendency get aggravated when the oxides are in a reduced state.

5. Reactivity of coke

The Coke Reactivity test is a highly regarded measure of the performance of coal. This test has two components; the Coke Reactivity Index (CRI) and the Coke Strength after Reaction (CSR). A coal which, when coked, achieves a low CRI value and a high CSR value is highly regarded in the market, primarily because this test has been related to blast furnace performance, particularly fuel rate and permeability of the burden.

6. Chemistry of burden

Richness of iron ore, gangue content and chemistry, type of flux used, carbon and ash content of coke along with input blast chemistry viz degree of oxygen enrichment, humidification, temperature and pressure of blast can alter the process of counter current reactor.

Dependent variables

1. Top gas analysis
2. Blast pressure
3. Heat loses to cooling water
4. Rate of production of metal and slag
5. Chemistry of metal and slag
6. Temperature of metal and slag
7. Distribution of charge at the throat
8. Top gas temperature
9. Top gas pressure

Modelling:

The process modelling of blast furnace involves the nonlinear constitutive and conservation equation. All developments of mathematical process models are directed towards a better understanding of what happens within a blast furnace.

The three models:

1. One dimensional steady state models - Based on kinetics
2. Two dimensional steady state models – Based on Rate and transport phenomena, solid flow, phase changes, and other coupled forces.
3. Three dimensional dynamic process model – Based on internal state and process behaviour of a furnace.

Purpose of blast furnace models

- Overall model – High rates of injection, top gas recycling, etc.
- Burden distribution model - Charging programs of the sequence of charging various solids.
- Wall temperature distribution model – Detection and visualisation of patterns.
- Top gas distribution models – First principles model with parameter estimation based on the classified probe reading.
- Dead man model –Motion of the dead man zone.
- Silicon prediction models – Prediction of silicon content of hot metal.
- Freeze line model – Detection of accretion formation/melting.
- Tuyere state model – Flow of gases to and from the tuyere level.
- One dimensional/ two dimensional shaft simulation models – Dynamic of the thermal state in the shaft and gas flow in the shaft.

Types of model available

1. Fundamental or mechanistic model
2. Empirical model
3. Population-balance models

Important process models

1. Burden distribution model

The Burden distribution model describes the trajectory of these solids as a function of the chute design and the VTA position (the angle at which the throat armour plate is inclined).

For bell less top, the model predicts the burden profile after a given number of revolutions of the chute at any given angular positions.

2. Thermochemical model

The model can be used to predict the internal state of a furnace. It involves two dimensional mapping of the temperature and chemical composition profiles of gases and solids from the stock line to the raceway periphery, including the cohesive zone.

3. Model of the raceway

The raceway is the zone around the tuyeres of a blast furnace where the combustion of the coke and supplementary fuels (like pulverised coal, natural gas , etc) takes place in the presence of the pre heated air blast. This generate most of the heat required by the process.

The performance of a blast furnace can be increased through adopting improved technologies/ modern trends in blast furnace practice. Some of the recent developments are as follows.

- 1 .Big blast furnace
2. High top pressure
3. Oxygen enrichment of blast
4. Pulverised coal injection
5. Bell less charging system
6. Humidification of blast

LECTURE- 21

Recent developments in Blast furnace operations

Introduction:

In recent years many advances have been made in blast furnace design and practice to

- (i) Reduce coke consumption
- (ii) Increase productivity

The various improvements made in the furnace design are:

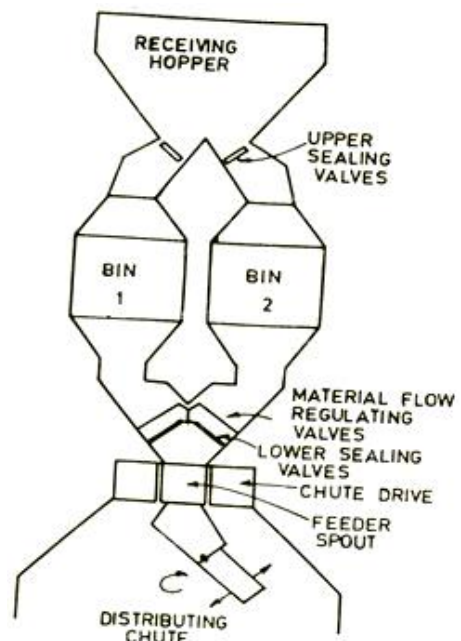
- (i) large capacity blast furnace
- (ii) better prepared burden
- (iii) high top pressure
- (iv) efficient burden distributor
- (v) better coke quality
- (vi) auxiliary fuel injection through tuyeres
- (vii) humidification of blast
- (viii) oxygen enrichment of blast
- (ix) higher blast temperature through better stoves
- (x) lime dust injection through tuyeres

BELL LESS TOP CHARGING:

In this design large bell is replaced by a distributor chute with no sealing function. A rotating chute is provided inside the furnace top cone. All the materials are charged via, holding hopper, with seals at its top and bottom which are charged and discharged alternately.

The advantages of this design are:

- (i) Greater charge distribution flexibility
- (ii) Easy access to any part of the system

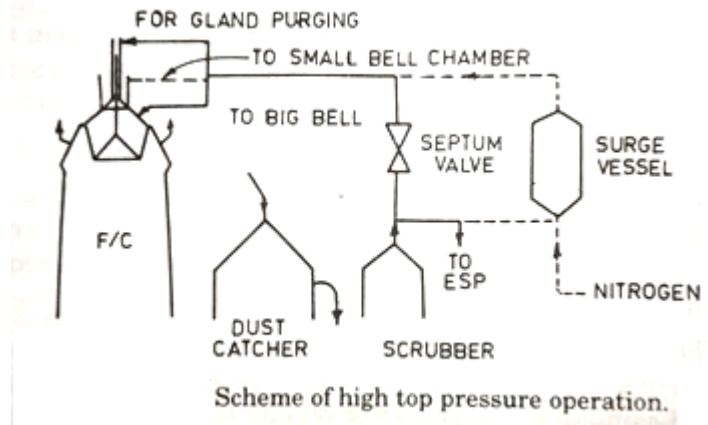


Paul-Wurth bell-less top

- (iii) Less maintenance cost
- (iv) Total overall height can be much less
- (v) Less investment for top construction
- (vi) More operational safety and easy control over varying charging parts

HIGH TOP PRESSURE

The increase in B/F top gas pressure by throttling back of the discharge gas is known as High Top Pressure (HTP). The throttling device is generally located after the second stage of gas cleaning system (scrubber), due to which the furnace along with the uptakes, down comer, dust catcher and scrubber are in increased pressure.



Difficulties in adoption in existing furnaces

In bell less charging systems it is easier to maintain High Top Pressure, but in case of existing furnaces with two bell charging systems a differential pressure is created across the big bell. As the B/F inside pressure is higher than the pressure of the big bell hopper the bell is difficult to be lowered down. So pressurizing the big bell hopper is needed for charging the furnace. Again depressurizing the big bell hopper becomes necessary to lower the small bell. Therefore an additional system of pressurizing and depressurizing the big bell hopper is required in an old furnace which is capable of withstanding the high top pressure for feasible adoption of the technology. Generally a part of the semi cleaned blast furnace gas after scrubber is by passed through a septum valve to the big bell hopper to increase the inside pressure. Further sealing of these bell charging systems becomes critical during adoption of HTP.

Benefits of HTP

1. Increased production rate due to available higher contact time of gas-solid inside the furnace.

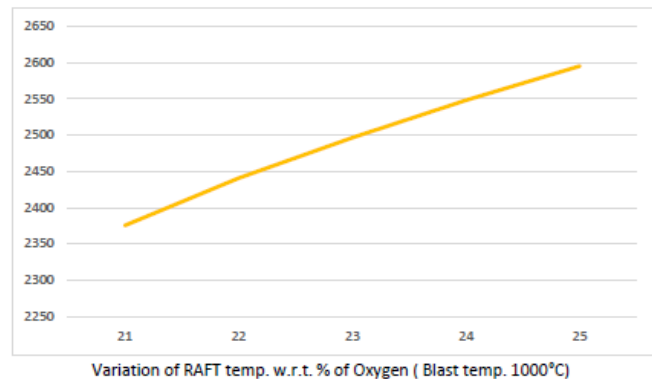
2. Boudouard equilibrium $\text{CO}_2 + \text{C} \leftrightarrow 2\text{CO}$ shifts to left leading to coke saving.
3. $\text{SiO}_2 + \text{C} \rightarrow \text{SiO} + \text{CO}$ at bosh level is hindered results in decreased further reduction of SiO gas and less Si in hot metal.
4. HTP also minimizes channeling (a blast furnace irregularity) and dust loss through out going B/F gas.

OXYGEN ENRICHMENT AND HUMIDIFICATION OF BLAST

In blast (air) presence of about 79% Nitrogen by volume restricts the temperature generated in the combustion zone as Nitrogen only acts as heat carrier. This temperature can be increased by decreasing Nitrogen content i.e increasing the Oxygen content in the blast which is called Oxygen enrichment of blast.

Advantages

1. Increases f/c temperature in turn decreases coke rate
2. Less amount of flue gas so reduces the loss of sensible heat
3. Increases productivity about 3-4% with 1% increase in oxygen.



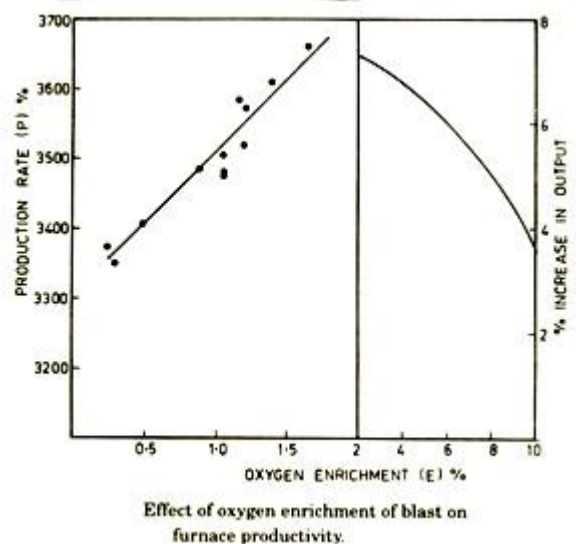
Disadvantages

1. Higher temperature leads to higher Silicon content in pig iron, only beneficial for foundry iron.

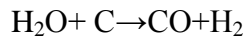
At high temperatures $\text{SiO}_2 + \text{C} \rightarrow \text{SiO} + \text{CO}$ reaction occurs and the SiO is in gaseous state which is again reduced to metallic Si and joins the hot metal.

2. Higher enrichment may cause refractory failure and tuyere burning

Use of oxygen enrichment upto 25% oxygen in the blast has been found to advantageous if it is



balanced by adequate humidification of blast. Addition of steam is the corrective addition because of its endothermic reaction with carbon:



Blast produces double the volume of reducing gases per mole of carbon burnt.

For humidification steam is introduced in the cold blast before it is preheated in the stove. If it is introduced in the hot blast, since the steam temperature will not be as close to that of hot blast, it will have a cooling effect which is not desirable. Humidification eliminates the day to day changes in humidity level and thereby eliminates one major variable that affects the blast furnace operation markedly.

AUXILIARY FUEL INJECTION:

Because of the increasing cost and rising scarcity of coke, it should be replaced by other cheap and readily available fuels without affecting the furnace efficiency. Solid fuels coal, either pulverized or in the form of slurry, liquid fuels either light oil, naphtha, or heavy oil, or gaseous fuel like natural gas or coke oven could be readily injected in the tuyeres. The choice of type of fuel to be injected depends on its availability, economics of its use and feasibility of injection. Pneumatic transport and injection of oils and natural gas is relatively easier than pulverized coal. Due to ease of use, oil, followed by natural gas, were the popular injectants in the sixties and seventies. With the oil crises, many companies stopped injecting oil into the Blast Furnaces and turned to alternative sources of fuel. Coal preparation and pneumatic conveying had become proven technologies by that time and this encouraged iron maker to consider coal as a suitable injectant. Most iron makers inject coal into their Blast Furnaces at present, the vast majority since 1980, due to its relatively low cost and beneficial effects on the Blast Furnace process. Replacing the use of coke with alternatives has the following additional benefits:

- Coke purchased from external sources could be minimized or eliminated altogether for plants with a coke shortage. The cost of building a new coke plant is about three times more than that for a PCI plant, for example. Even a rebuild of an existing coke plant can be twice as expensive when compared to the cost of a new PCI plant
- The useful lifespan of existing coke batteries could be prolonged by running the

batteries at reduced throughput, thereby obviating the need for costly rebuilds

- Old, environmentally unsound coke plants could be shut down
- Reducing the output of an existing coke battery could improve the quality of the coke produced by using the extra process room created due to the lower production rates

However, as coke provides physical support and gas permeability in the BF, its complete substitutions is not possible. The amount of coal that can be injected will depend on the coal and coke quality, furnace geometry, and operational practices. The fuel injection has resulted in 6-10% saving in coke and increase in productivity by about 0.5-2.0%.

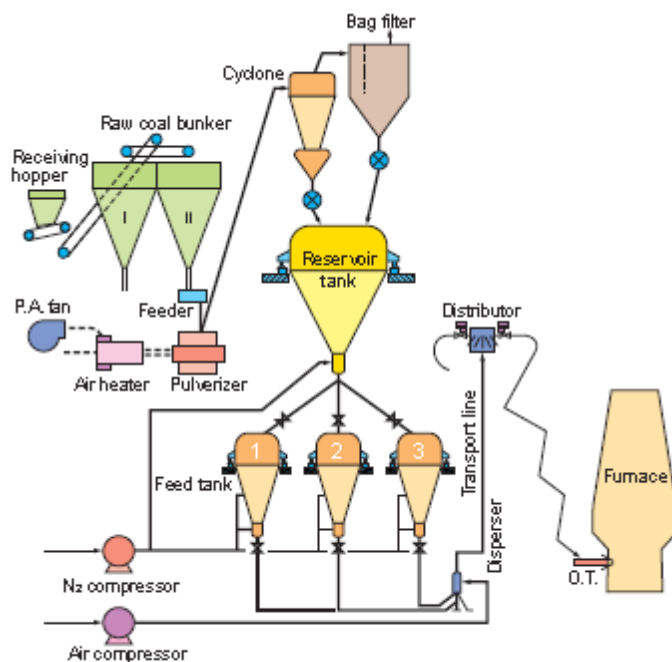


Fig. 1 Process flowchart of pulverized coal injection facility

These injections affect the following furnace parameters:

- (i) Flame temperature
- (ii) Bosh gas composition and volume
- (iii) Top gas temperature and CO/CO₂ ration and hence efficiency
- (iv) Operational efficiency
- (v) Slag volume and its basicity
- (vi) Overall production efficiency

Within the boundaries posed by process parameters such as flame temperature and pressure

differential over the burden, tuyere injectants also benefit the Blast Furnace process enormously. The cooling effect of the injectants in the raceway enables the use of higher hot blast temperatures and lower moisture additions, thereby reducing the total fuel consumption. The same cooling effect also permits the use of a higher concentration of oxygen in the hot blast, thus reducing the raceway gas volume.

Pulverized coal injection technology has the potential to spur innovations to blast furnaces: recycled materials, such as waste plastics and biomass, as well as recycled ores can be injected with pulverized coal into the furnaces via tuyeres. Thus, the technology is expected to be developed as core blast furnace technology, addressing resource, energy, and carbon dioxide issues.

Formed Coke:

Another important fuel to substitute conventional coke is the recently developed formed coke. Several processes have been developed the worldover to produce moulded fuel from non coking lower rank coals. These are designated as formed coke. Following advantages are obtained from using formed coke :

- (i) Cheaper non coking coals are used to make the formed coke.
- (ii) Lower capital and running manufacturing cost.
- (iii) Consistent quality of product.
- (iv) Continuous operation of the plants.

Ferro Coke:

A carbonized lump produced from a mixture of iron bearing fines and non metallurgical coal is known as ferro-coke. It is produced by carbonizing the mixture much in the same way as that for conventional coke making.

Following are the advantages claimed:

- (i) Non metallurgical coal is used indirectly as a fuel in the blast furnace.
- (ii) Total fuel requirement in the furnace is reduced
- (iii) Reduced coke consumption
- (iv) Increased productivity.

LECTURE- 22

Irregularities in blast furnace operation and their remedies

In spite of the many improvements in burden materials and operating procedures that have been made in recent years, the blast furnace does not always run as smoothly as the casual observer may be led to believe. Furnace upsets are not as frequent as they were in former years; however, irregularities still do occur that cause considerable concern and often require quick thinking and the use of good judgment and skill on the part of the operator to prevent serious trouble. The most common irregularities occurring are:

Hanging:

If uniform descent of the burden in the furnace is interrupted either by wedging or bridging of the stock or by scaffolding, it is known as hanging.

Causes:

1. Solidification of previously fused slag, particularly in sinter, into a large impervious mass. In some instances, slag that has been melted is blown upward in droplets; when it subsequently contacts colder burden material it resolidifies and plugs up the openings between the particles and tends to cement them together.
2. Bridging of ore particles in the vicinity of fine coke particles which instead of separating the ore particles flow in the interstitial positions of the ore particles.
3. The deposition of large amount of carbon from Naumann reversion reaction $2\text{CO} = \text{CO}_2 + \text{C}$ in the voids if pressure is at higher side and consequent decrease in permeability of the burden as the carbon is deposited as soot will plug up the openings between the particles and will hold the particles together.
4. In some instances, where the alkali content of the burden is high, the alkali compounds will be reduced to alkali vapor that ascends with the furnace gas and condenses in the cooler portion of the charge in the upper part of the stack thereby cementing the charge into a impervious mass.
5. Excessive gas pressure resulting in counteracting the downward movement of the stock. When there is a high percentage of fines in the burden and the velocity of the furnace gas

is relatively high, the fines will plug the openings between the other particles and cause hanging.

6. A similar type of hanging may also occur if the furnace is being run at too high a flame temperature for the quality (particularly the reducibility) of the burden material. When the high temperature isotherms expand far enough up the furnace they can begin to melt unreduced material as that material descends into a more reducing environment it reduces and, depending on the temperature, it may solidify (the melting point of FeO is roughly 1370°C and pure iron is roughly 1535°C), and plug the burden.

Types

- A. Top hanging: occurring in the slack and mainly arising due to the carbon deposition reaction and alkali vapour condensation.
- B. Bottom hanging: occurring anywhere above the tuyere due to low voidage in the stack.
- C. General hanging: occurring anywhere above the tuyere due to low voidage in the stack.

Remedies:

1. Use of large lump lime stone produces CO₂ which forces the solution loss reaction to take place and improves permeability of the bed.
2. Hanging f/c responds better to a cooler blast.
3. Reduction in blast pressure distributes the gas flow, though slowly, but more uniformly.
4. A persistent hanging can be cured by blowing down the furnace to bosh level and filling it with coke blank.

Scaffolding:

Scaffolding is the term used when accretions or scabs build up on the furnace walls and cause a decrease in the cross-sectional area of the furnace stack. Scaffolding can occur relatively high in the furnace stack or relatively low, near the top of the bosh. The formation of scaffolds near the top of the bosh often results because of excessive fines in the burden material and a higher than normal lime chemical composition of the slag. The solution of lime into the slags formed in the furnace stack increases the slag melting point. Because the slag often carries some of the fine particles from the burden in suspension, the increase in melting point can cause this mixture of fines and slag to adhere to the upper bosh walls; this buildup deflects the hot gases farther into

the center of the furnace. With less hot gas along the walls, the accretions tend to cool down and solidify completely, and they can then grow until they block a large percentage of the furnace cross-section and render it ineffective for smelting. This increases the fuel rate, decreases the hot metal production rate and promotes hanging and slipping. Scaffolds can also be caused by alkali or zinc compounds that are reduced to metallic vapor near the bottom of the furnace and rise with the furnace gases to the cooler top portion where they are reoxidized to very fine solid particles. These very fine particles adhere to the furnace inwalls and entrap other fine material there to start the formation of a scaffold. Such scaffolds decrease the production capacity and distort the gas flow so that poor fuel efficiency results. Also, scaffolds can dislodge from the walls and descend into the hearth, causing serious furnace upsets and poor quality metal, or if large enough they can cause the furnace to freeze. Erratic working of furnace or decrease in fuel rate or shut off of blast for any reason may lead to scaffolding.

Remedies:

1. Charging a few coke blanks
2. Reducing the gas pressure on the affected side with more siliceous burden
3. Scaffolds can be removed by jumping the furnace (sudden put off of blast) or by use of dynamite ultimately.

Slips:

Slips are caused initially by hanging or bridging of the burden material in the stack of the furnace. When this occurs, the material below the hang continues to move downward, forming a space that is void of solid material but filled with hot gas at very high pressure. This space continues to grow until the hang finally collapses. In severe cases, the sudden downward thrust of the hanging material forces the hot gas upward with the force of an explosion and also chilling of hearth. This sudden rush of gas opens the explosion bleeders and sometimes is so great that it causes severe damage to the furnace top gear. The hanging that precedes slipping is caused by any of a number of different conditions in which the permeability of the charge is decreased because some of the material plugs up the voids between the charged particles and bonds them loosely together. Bad bosh design may cause slipping.

When the burden is not moving properly through the furnace, the operator must take corrective steps immediately to prevent a disastrous slip and determine the cause of the hanging so that

changes can be made in the operating procedure to prevent the hang from recurring. The best remedy is to allow the furnace to slip on its own by adjustment of the blast temperature and pressure.

Channeling:

Preferential flow of ascending gases through certain areas of the burden is known as channeling since the passages appear as channels. For satisfactory blast furnace operation the burden must be properly distributed both radially and circumferentially. If a preponderance of fines is charged in a particular area of the furnace the ascending gasses will be diverted from that area and channel around the fines. Wide size range of the burden can also cause channeling. This channeling will upset the preheat and reduction processes due to improper contact between gas and solids and result in off chemistry hot metal, lower productivity or unscheduled bleeder openings or worse.

Proper size and size range and proper distribution of burden minimizes channeling.

Breakout:

A 'breakout' is the term used to denote the conditions and results of the escape of gas and coke, or slag, or iron, from the bosh, tuyere breast, or hearth of a blast furnace. Breakouts may occur at any point below the fusion zone in the furnace, but the most of the severe breakouts are of liquid slag and of liquid iron. Liquid iron breakout takes place at a level below the surface of iron lying in the hearth, and are either through the hearth walls and jacket or into the hearth bottom and out under the hearth jacket. Breakouts are caused by failures of the walls of the hearth, with the result that liquid iron or liquid slag or both may flow in an uncontrolled way out of the furnace and surrounding auxiliaries. Slag break outs are usually not as serious as iron break outs, because there are not as much danger from explosions as in the case where molten iron and water come into contact. With either type of breakout, it is necessary, if at all possible, to open the tap hole and drain out as much liquid material as possible, and to take the furnace off blast. In case of a slag breakout, the breakout may be chilled by stream of water, and the hole where the breakout occurred may be closed by replacing the bricks, or pumping fireclay grout in the opening or ramming a plastic cement or putting asbestos rope into it. In case of iron breakout, there is practically no control. The hot metal is to run out of the hole until the furnace is dry. After the

accumulated iron has been cleared away, a suitable refractory may be used for closing the hole. In iron break out is severe then a complete hearth repair will be needed. In case of non severe break outs it may be necessary to change the damaged hearth cooling staves. A properly designed hearth with minimum of refractory joints and the use of carbon brick as refractory along with modern construction techniques have resulted in a robust hearth design for minimum breakouts.

Chilled hearth:

This is a very serious disorder since it affects tapping adversely. It may result because of low fuel input, excessive moisture in the blast, water leaking from tuyeres, etc. if it is due to these reasons chilling is gradual and can be rectified before it becomes serious. Slip also can cause chilling but it is sudden and depending upon the extent of chilling thus caused, gas torches are used to open the tap holes. The metal may even be tapped through cinder notch, if required and if possible, under such circumstances.

Pillaring:

If the blast is unable to penetrate right up to the centre of the furnace it may lead to the formation of a cold central column of the stock with an annular hot zone around it. This is known as pillaring. A bar inserted right through the through the tuyere hole would show red hot portion at both ends and a cold middle portion if pillaring exists in the furnace. The extent of cold middle portion of this rod indicates the extent of pillar existing in the furnace.

Pillaring can be eliminated by increasing the blast pressure which will penetrate more and heat up the pillar.

Choking of gas uptakes:

Furnace operation has to be suspended if dust gets accumulated in the uptakes and down comer and it can be resumed only after the clean up. This happens because of faulty gas uptake design, particularly the inadequate cross section and improper joints.

Flooding and coke ejection through tap holes:

In bosh, liquid slag and metal trickle through the permeable coke bed against the upward thrust of the ascending gases. An increase in gas or liquid flow can prevent the liquid metal and slag

from flowing downwards, causing it to accumulate in the coke interstices until weight of the liquid overcomes the upward thrust of the gases and descends suddenly into the hearth. This phenomenon is known as flooding which can be minimized by having a high voidage i.e. by using higher mean size of coke. Better quality coke is also beneficial since the degradation inside the furnace would be reduced and consequently the permeability in the bosh region would be improved.

Anything that imposes a constraint on the tuyere raceway volume causes hold up and subsequent tendency to flooding. The interruption in uniform blowing rate causes the raceway to collapse and when it is resumed again the small particles of coke can't reenter the raceway and consequently descend in to the hearth instead of burning in the tuyere area, resulting in a choke hearth thereby causing the well known phenomenon of coke ejection from the slag and the iron notches during tapping. This often leads to unjustified criticism of coke quality. Uniform blowing of the furnace is the best remedy to avoid this.

Leaking tuyeres, tap holes and coolers:

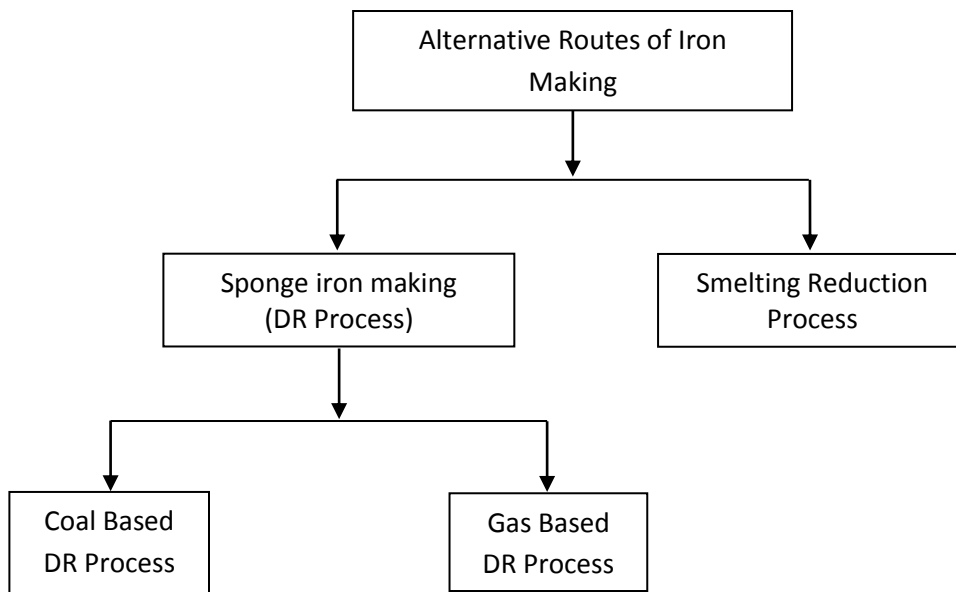
In spite of proper designs, the water cooled parts of the furnace may give way and these should be immediately replaced or rectified, as far as possible. The monkey is the troublesome part and often needs frequent replacements. If it is not possible to rectify or replace the faulty cooler it has got to be cut off from the water mains and put out of use.

Leaking tuyeres or coolers in the lower part of the furnace can have disastrous effects if these are not rectified in time. The leaking tap hole coolers lead to generation of steam which on coming in contact with the carbon hearth, erodes the hearth lining and the campaign has to be stopped for capital repairs. The maintenance instructions for tap holes must be scrupulously followed to minimize these troubles.

LECTURE-20 &21

Alternative routes of iron making: Sponge iron

The production of iron is only intermediate inevitable step in the ultimate production of steel from natural source like iron ore. The place where coking coal is not available but other gaseous (natural gas) or solid fuels like non-coking coals are readily and abundantly available an alternative method of iron production using other fuel is feasible. Traditionally, blast furnace hot metal and recycled steel scrap have been the main source of iron. However, blast furnace hot metal production has decreased over the years owing to problems associated with the availability of high grade metallurgical coke. With increased efficiencies of steel making as well as with the advent of continuous casting, scrap generation has also decreased. All these factors have contributed towards the development of alternative technologies to produce iron units.



Today processes are available to produce solid iron (Fe) in the form of sponge iron (also referred to as Direct Reduced Iron or DRI) as well as to produce hot metal without high-grade coke i.e. via Smelting Reduction (SR).

Direct-reduced iron (DRI), also called sponge iron, is produced from direct reduction of iron ore (in the form of lumps, pellets or fines) by a reducing gas produced from natural gas or coal. The reducing gas is a mixture, the majority of which is hydrogen (H₂) and carbon monoxide (CO) which act as reducing agents. This process of reducing the iron ore in solid form by reducing

gases is called direct reduction. DR processes convert iron ore (fines, pellets, sinter, etc.) into sponge iron at temperatures well below the melting point of iron. These processes distinctly differ from the conventional blast furnace process in two major respects: solid metalized product is produced, rather than molten iron; and a wide variety of reductants may be used in place of metallurgical-grade coke.

In this process, the ore is converted into metallized ore, by removing the oxygen from the iron oxide lattice to leave metallic iron. The iron left after the removal of oxygen has a honeycomb like construction and hence the name sponge iron. Originally, when direct reduction of iron or DRI plants were envisaged, the idea was to replace coke whose availability and cost became highly uncertain and prohibitive. The original process of DRI replacing coke and using coal was hailed as a precursor of a new regime of making iron and steel.

The conventional route for making steel consists of sintering or pelletization plants, coke ovens, blast furnaces, and basic oxygen furnaces. Such plants require high capital expenses and raw materials of stringent specifications. Coking coal is needed to make a coke strong enough to support the burden in the blast furnace. Integrated steel plants of less than one million tons annual capacity are generally not economically viable. The coke ovens and sintering plants in an integrated steel plant are polluting and expensive units.

Direct reduction, an alternative route of iron making, has been developed to overcome some of these difficulties of conventional blast furnaces. DRI is successfully manufactured in various parts of the world through either natural gas or coal-based technology. Iron ore is reduced in solid state at 800 to 1,050 °C (1,470 to 1,920 °F) either by reducing gas (H₂+CO) or coal. The specific investment and operating costs of direct reduction plants are low compared to integrated steel plants and are more suitable for many developing countries where supplies of coking coal are limited.

The direct reduction process is energy efficient, but most competitive with the Blast Furnace (BF) when it can be integrated with Electric Arc Furnace (EAF) to take advantage of the latent heat produced by the DRI

Typical Chemistry of DRI

Fe Total	90-94%
Fe Metallic	83-90%
Metallization	92-96%
C	1.0-2.5%
P ₂ O ₅ *	0.005-0.9%
S*	0.001-0.03%
Gangue*	2.8-6%

* Depends on the iron oxide source

* Gangue is the term for the oxide minerals (i.e. CaO, Al₂O₃, SiO₂, MgO) that remain in the DRI and are removed in the steel slag.

The amount of metallic iron produced from a given quantity of ore is termed as the degree of metallization and is defined by the ratio of the amount of the metallic iron produced to that of total iron in the ore.

Properties of sponge iron

1. Apparent density: 1.5-4.0 g/cm³. It is higher for Hot Briquetted Iron (HBI).
2. Specific Surface area: 0.5-4.0 m²/g (usually around 1.0m²/g).
3. Carbon content: 0.10-0.15 wt% in coal based and upto 2.5wt% in gas based DRI processes.
4. In situ rusting: the high porosity and large specific surface area of sponge iron make it prone to re-oxidation, which take place if DRI gets heated in air to its ignition temperature of 200°C. Reoxidation can also take place automatically if DRI comes in contact with moisture, even at normal atmospheric temperatures, since heat gets generated along with the evolution of hydrogen. This phenomenon is often referred to as in situ rusting.
5. Ideal replacement of scrap: Uniform size, uniform composition, absence of tramp elements like copper, zinc, tin, lead, arsenic, etc

Advantages of DR Process

- Elimination of dependence on expensive coking coal.
- Smaller module size.
- Lower total capital investment.
- Superior environmental friendliness.

- Carbon boil: DRI always contains FeO. The residual FeO in DRI, as long as it is not very high, leads to CO formation when the residual oxygen reacts with carbon present in the melt, resulting in stirring of the molten bath.
- Direct-reduced iron has about the same iron content as Pig Iron (MPI), typically 90–94% total iron (depending on the quality of the raw ore) as opposed to about 93% for molten pig iron, so it is an excellent feedstock for the electric furnaces used by mini mills, allowing them to use lower grades of scrap for the rest of the charge or to produce higher grades of steel.
- Hot-briquetted iron (HBI) is a compacted form of DRI designed for ease of shipping, handling, and storage.
- Hot Direct Reduced Iron (HDRI) is iron not cooled before discharge from the reduction furnace that is immediately transported to a waiting electric arc furnace and charged, thereby saving energy.

Disadvantages

- Final product is solid and requires melting.
- Low productivity in terms of unit volume.
- Low carbon content which may create difficulties in subsequent steel making
- Directly reduced iron is highly susceptible to oxidation and rusting if left unprotected, and is normally quickly processed further to steel.

Degree of Metallization:

The percentage of iron as part of the whole iron, existing as metallic iron is called degree of metallization.

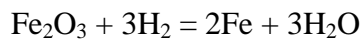
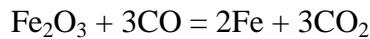
Percentage Reduction:

The percentage oxygen, associated with iron as oxides, removed from the ore particle is often referred to as percentage reduction.

Physico-chemical principles in DR processes:

The physico-chemical principles of reduction of iron oxide in solid state are complex because although the reduction occurs, the exact role played by the gaseous and solid reducing agents in the reduction of iron oxide is uncertain. Much of the mechanism and kinetics of iron oxide reduction in the blast furnace stack hold true for direct reduction processes to metalized ores without fusion.

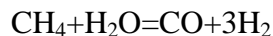
The reducing agents used in direct reduction are either gaseous and/or solid carbonaceous materials. The main reduction reactions are:



Since hydrogen is too costly to be used for such commercial applications, natural gas, where ever available and in reformed condition, is used as substitute for CO or H₂. Natural gas cannot be used directly because of very slow reduction as compared to CO and H₂ and carbon soot formation and consequent process problems like choking, unfavorable thermal balance, etc.

Reforming of natural gas

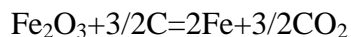
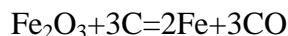
The natural gas is desulphurised and reformed by mixing with steam and passing it over nickel as catalyst.



Atypical composition of reformed natural gas contains 75% H₂, 14% CO, 3% CH₄ and 8% CO₂.

This mixture of CO and H₂ is used in some of the commercial direct reduction processes. Other hydrocarbons methane, naphtha, petroleum, etc. are used for reforming.

Solid reductants like non-metallurgical coals when used may carry the reaction like:



All the above reactions are endothermic and hence heat must be provided for these reactions to take place.

Direct reduction processes available can be broadly grouped under two categories based on the type of reductant used. These are:

- Coal based processes
- Gas based processes

Coal Based Processes

- **Rotary kiln based processes**

SL/RN, CODIR, ACCAR, DRC, TDR, OSIL, Jindal

- **Shaft furnace based processes**

Kinglor METOR

- **Rotary hearth furnace based processes**

INMETCO, FASTMET, COMET.

From amongst various solid based processes, only a few have attained commercial significance. Most of the processes such as SL/RN, KRUPP-CODIR, DRC, TDR, SIIL, JINDAL, OSIL, whereas Kinglor Metor process utilises an externally heated vertical retort.

Gas Based Processes

- **Retort processes**

HyL-1, Hoganas

- **Shaft furnace processes**

Midrex, HyL-III, plasmared, Armco, purofer.

- **Fluidized bed processes**

Finmet

Today gas based DR plants subscribe to more than 90% of installed DR capacity in the world, of which MIDREX and HYL together have about 85 % of the total capacity to their credit. In the gas based processes, the reduction of iron oxide is carried out by a mixture of CO & H₂ at a temperature of about 750-950°C. The reducing gas is produced by reformation of natural gas. The reformation is partial oxidation of hydro-carbons. To enhance the reformation process, normally a catalyst is used.

Coal based processes vs. Gas based processes

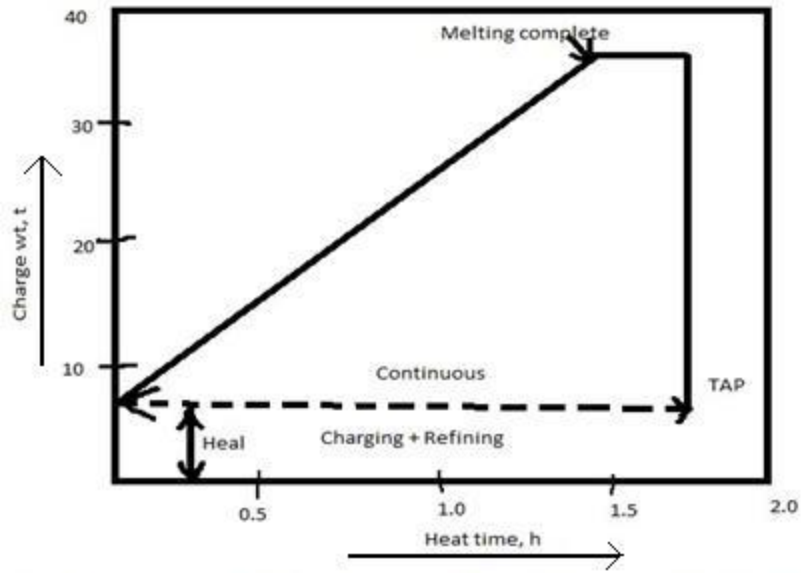
Coal based processes	Gas based processes
Uses Non-metallurgical coal	Uses reformed natural gas
Poor reduction kinetics	Good reduction kinetics
C%=0.15 to 0.2	Upto 2.5

Uses of Sponge Iron:

Though DRI is an intermediate product and does not have any direct use, it can readily be used in open hearth, LD and cupola for further processing to obtain final product. DRI-EAF has emerged as an alternative route for steel making and achieved much importance in Indian scenario.

For the production of plain carbon steel in electric arc furnace, steel scrap constitutes the principal metallic iron feed. The alternative iron sources (AIS) are:

- Hot briquetted iron
 - Solid pig iron
 - Hot metal (molten pig iron)
-
- Use of AIS is gaining popularity in EAF steelmaking. DRI/HBI is now the principal feed stock next to scrap. Solid pig iron and hot metal are also important AIS, constituting 5-8% of total feed .
 - it is recommended that hot metal charge is restricted to maximum of 40-45% of the total charge because excess hot metal usage can give rise to prolonged refining time and uncontrolled foaming.
 - DRI/HBI has a very low impurity content and hence does not require any additional refining time. However it is a porous material that tends to get severely oxidized in contact with moist air at high temperature.
 - Up to about 30% DRI (of the total charge) can be charged along with scrap in buckets. 1st a layer of scrap then DRI and then another layer of scrap are used in each bucket.
 - In all cases HBI is preferred because it is dense and does not get oxidized rapidly.
 - With the use of DRI/HBI, melting and refining can proceed simultaneously.
 - **Hot-heel practice:** 100% DRI is used, molten steel from a previous heat is not tapped out completely and allowed to remain in the EAF to provide a liquid metal bath for DRI charging right from the beginning of the next heat.
 - The figure below shows a typical change of weight of metal in the furnace as a function of time during an EAF heat for hot heel practice:



Arc furnace operation with high percentage of DRI in the charge using hot heel practice

LECTURE -24

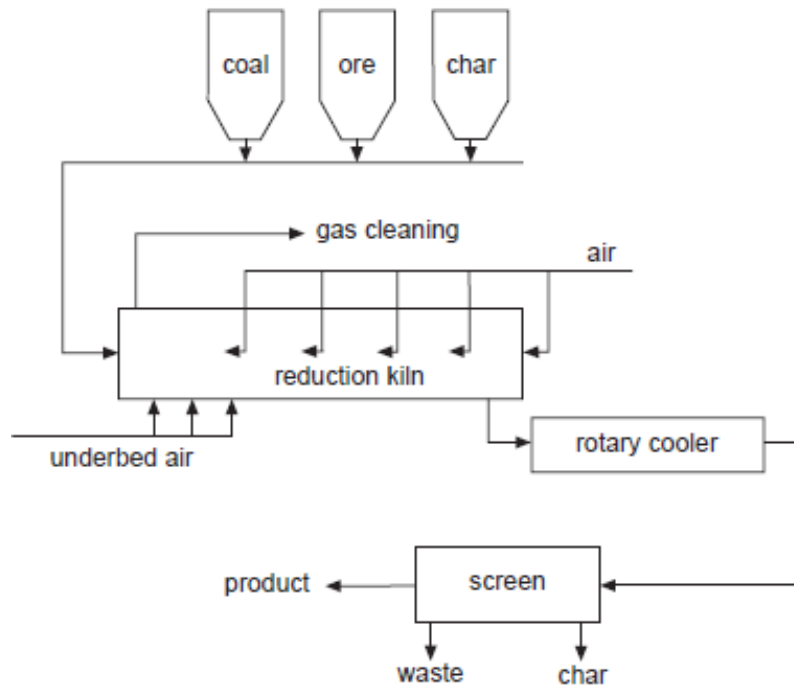
SL/RN Process

Introduction:

The forerunner to the SL/RN process, the R-N process (for Republic Steel Company and National Lead Corpora) was developed in Norway, primarily to recover TiO_2 from titaniferous ores for producing pig-ments. Further development showed that other iron bearing ores could also be treated to produce iron and a pilot plant was built in the United States. In 1964 Lurgi Chemie acquired the R-N patents and world rights and developed the technology further in cooperation with the Steel Company of Canada, Ltd. (Stelco) to form the SL/RN process. The reduction kiln is very similar to the system used in CODIR and the other coal based plants described later. The major differences between SL/RN and CODIR are that in SL/RN the product is cooled by indirect water sprays on the outside of the cooler, and the kiln off gas is cooled in a waste heat recovery boiler. Indirect product cooling requires a longer cooler than does direct cooling, however, there is no loss of metallization during indirect cooling. In the direct spray cooler, the loss of 0.5–1.0% metallization by reoxidation off-sets the economic advantage of a smaller cooler.

By 1998, 15 SL/RN kilns were in operation in Brazil, India and South Africa for the production of DRI for steelmaking. Their combined annual capacity is about 1.68 million tonnes.

Process description:



SL/RN direct reduction process flowsheet.

The SL/RN process is fed with indurate pellets and/or lump iron ore. Iron sands are used at New Zealand Steel, with design modifications to provide efficient operation. A wide range of fuels and reluctant including lignite, char, low temperature coke, coke breeze and anthracite coal have been used satisfactorily. Depending on the fuel used, the proportion of the reluctant fed through the inlet of the kiln with the oxide feed and the proportion fired through the burner at the kiln exit will be adjusted. With very low-volatile coal, a supplementary fuel such as natural gas or fuel oil is fed through the central burner or through the air tubes to maintain the proper temperature profile in the kiln. Smooth kiln operation is achieved when operating with a relatively high volatile coal charged together with the iron burden through the kiln feed end.

Coals with a high reactivity, low free-swelling index and high ash fusion temperature are preferred. The coal should also be non-coking. A low ash fusion temperature will promote formation of accretions in the kiln. The coal ash composition is also important because a siliceous ash might react with ferrous oxide to form the low melting compound, ferrous silicate, and interfere with the reduction to metallic iron.

The kiln exit gases pass through a settling chamber for coarse dust removal and a two-stage after-burner. Combustible gases are burned in the first stage, followed by fine carbon particles in the second stage. The afterburner chamber temperature is controlled by means of water sprays. In some plants, the gas emerging from the afterburner is passed through waste heat boilers, scrubbed and then discharged through the stack.

The DRI is discharged at about 1000°C. An enclosed chute at the discharge removes lumps and transfers the hot DRI to a rotary cooler. The cooler, a horizontal metal cylinder 3.6 m (12 ft) diameter and 40–60 m (131–197 ft) long is immersed in a water trough and has water sprayed on the upper surface. The cooler discharges the DRI into a hopper, which serves as an air lock to prevent air ingress. Trommel screens located at the hopper outlet separate the discharge into fractions which then pass through magnetic separators. Screens downstream separate the nonmagnetic portions.

Recent experiments show that air admitted through ports below the bed in the preheat zone will burn some of the gases that would otherwise leave the kiln unburned. Such a practice would also permit the use of lignite with up to 60% moisture without pre drying. This technique would greatly improve the efficiency of the process and arrangements are now being made to install it on several existing SL/RN kilns.

The table is given below:

Name of Plant	Capacity (10³ tonnes/yr)	No. of Modules	Start-up Year
Acos Finos Piratini, Charquedas, Brazil	60	1	1973
Sponge Iron India Ltd., Paloncha, India	60	2	1980 1986
Siderperu, Chimbote, Peru	90	3	1980
ISCOR Vanderbijlpark, South Africa	720	4	1984
Bihar Sponge Iron Ltd., Chandil, India			
– BSIL I	150	1	1989
– BSIL II	150	1	1998
Prakash Industries Ltd., Champa, India			
– PIL I	150	1	1993
– PIL II	150	1	1996
Nova Iron & Steel Company, Bilaspur, India	150	1	1994
Total	1680	15	

PROCESS ADVANTAGES:

- Use of any iron bearing material.
- Wide variety of reductants.
- Proven DRI technology.
- Economic production of DRI.

LECTURE – 25

MIDREX Process

Introduction:

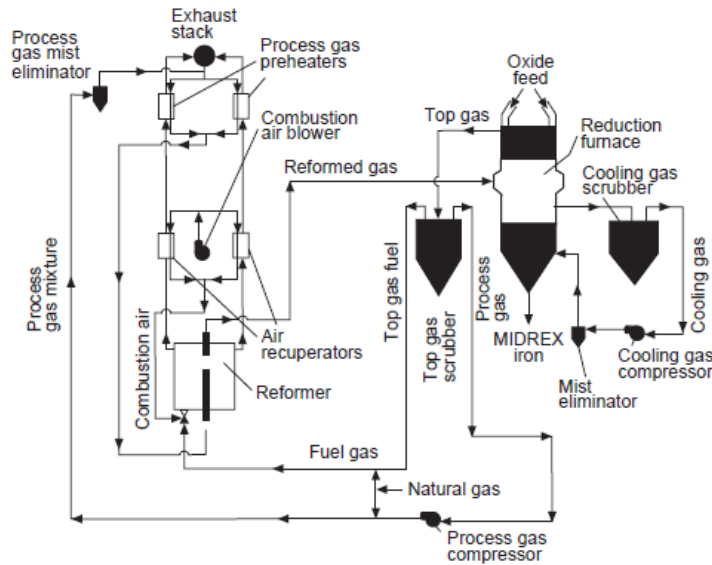
The Midrex process was developed by the Surface Combustion Division of Midland-Ross Corporation in the mid-1960s. The Midrex Division became a subsidiary of Korf Industries in 1974. Midrex was subsequently acquired by Kobe Steel, Ltd. in 1983.

The first commercial Midrex plant was installed near Portland, Oregon and started production in 1969. The plant included two shaft reduction furnaces of 3.4 m (11.2 ft) inside diameter and had a total capacity of 300,000 tonnes per year. The average energy consumption of this early plant was approximately 15 GJ/tonne (12.9 MBtu/ton) of DRI. Many difficult engineering and operating problems were solved during the first several years of operation of this plant that contributed significantly to the design, construction and operation of larger Midrex plants throughout the world.

The Midrex TM Direct Reduction process is based upon a low pressure, moving bed shaft furnace where the reducing gas moves counter-current to the lump iron oxide ore or iron oxide pellet solids in the bed. The reducing gas (from 10-20% CO and 80-90% H₂) is produced from natural gas using Midrex's CO₂ reforming process and their proprietary catalyst (instead of steam reforming).

Process description:

The iron oxide feed to a Midrex shaft furnace can be in the form of pellets, lump ore or a mixture of the two (in 0 to 100% proportions). The solid feed is discharged into a feed hopper on top of a proportioning hopper that evenly distributes the solids into the shaft furnace.



Midrex standard direct reduction process flowsheet.

A dynamic seal leg keeps the reducing gas inside the furnace. The shaft furnace operates at low pressure, under 1 bar gauge, which allows dynamic seals to be used on the furnace inlet and discharge. The iron ore burden in the shaft furnace is first heated, then reduced by the upward flowing, counter-current reducing gas that is injected through tuyer located in a bustle distributor at the bottom of the cylindrical section of the shaft. The ore is reduced to a metallization typically in the range of 93% to 94% by the time it reaches the bustle area.

Below the bustle area, it goes through a transition zone (with design to reduce agglomeration or lumping) and then reaches the lower conical section of the furnace. Lower carbon reduced iron (<1.5% C) is cooled using a circulating stream of cooled exhaust gas that is introduced in the conical section for cold DRI discharge. Higher carbon DRI (up to 4.0% C) can be produced by introduction of natural gas into this cooling gas. It readily reacts (and cracks) with the highly reactive metallic DRI.

For hot discharge of DRI to be used for hot charging of EAF's (i.e. Midrex's Hotlink™ Process) or for feed to hot briquetting presses (to produce HBI), the lower part of the furnace is modified to allow handling of hot burden. The Midrex gas generation system consists of a CO₂ reformer

using their own catalyst. The feed to the reformer is a mixture of process gas recycled from the furnace and makeup natural gas. The top gas leaving the shaft furnace at a temperature of 400 to 450C is cooled and dust is removed in a top gas scrubber. About two-thirds of the gas is recycled back to the process (process gas) and the rest is used as a fuel. The process gas is compressed, mixed with natural gas and is preheated in the reformer recuperates before entering the tubes of the reformer.

The reformed gas comprising of mostly CO and H₂ exits the reformer at about 850 °C and passes through collection headers to the reformed gas line. The ratio of H₂ to CO is controlled at about 1.5 to 1.8, and reducing quality at 11 to 12 for best operation.

PROCESS ADVANTAGES:

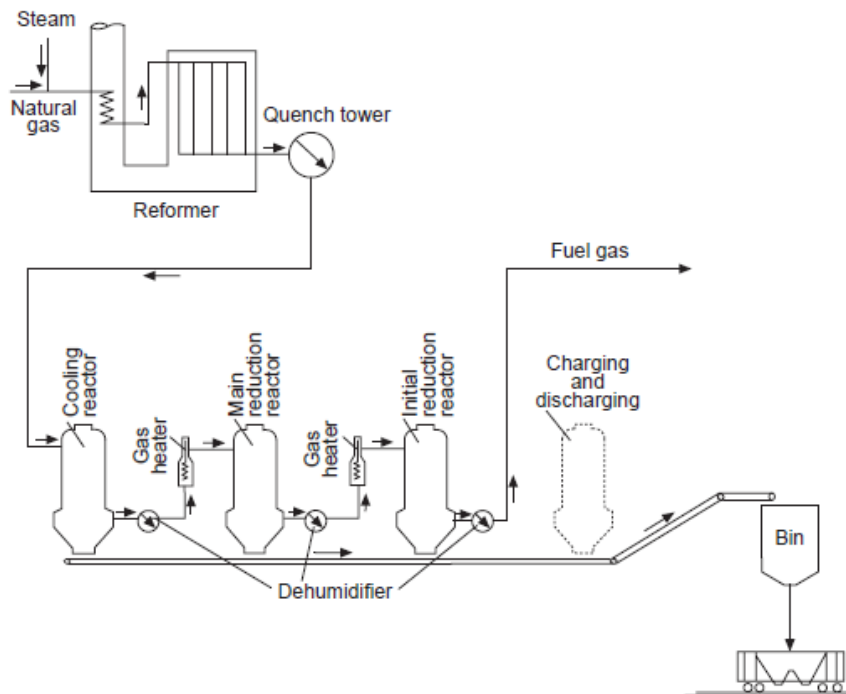
- World-wide commercial use proven. Performance “Relatively-forgiving” operation. Raw material flexibility.
- CO₂ reformer eliminates need for steam system, reformed gas quench, reducing gas heating and CO₂ removal system.

LECTURE -26

HyL Process

Introduction:

In the HYL process developed by Hojalata y Lamina S.A. (Hylsa) of Monterrey, Mexico, lump ore and fired pellets are reduced in fixed-bed retorts by reformed natural gas. The first commercial HYL plant was installed at Monterrey and started production late in 1957. This plant has a capacity of 200 tonnes per day of DRI and the reactors are approximately 2.5 m (8.2 ft) in diameter and hold approximately 15 tonnes of ore in a 1.5 m (4.9 ft) deep bed. The reactors in the more recent plants are 5.4 m (17.7 ft) in diameter and 15 m (49.2 ft) high. Design capacity is approximately 1900 tonnes per day of DRI having an average reduction of approximately 90%. The energy consumption in the most recent plants is 14.9 GJ/tonne (12.8 MBtu/ton) of 90% reduced DRI. In the more recent plants of the HYL II design, high temperature alloy tubes in the reducing gas reheating furnaces are used, which permits heating the gas to high temperatures, and the number of heating furnaces was reduced to two units from the original four units.



HYL process flowsheet.

In the HYL II process, reducing gas is generated by reforming natural gas with excess steam, to prevent carbon formation and to promote long catalyst life. The reducing section consists of a set of four reactors, three of which are in operation, while the fourth is engaged in discharging and charging operations. The HYL process is a cyclical batch operation, and the three on-line reactors operate in series. Reduction is performed in two stages, an initial reduction stage and a main reduction stage. Cooling, carburization (Fe_3C) and the final adjustment of metallization are performed in the third stage. Each stage takes roughly three hours. An intricate system of valves permits the reactors to be connected in any desired order so that any one reactor can be connected in its correct process stage.

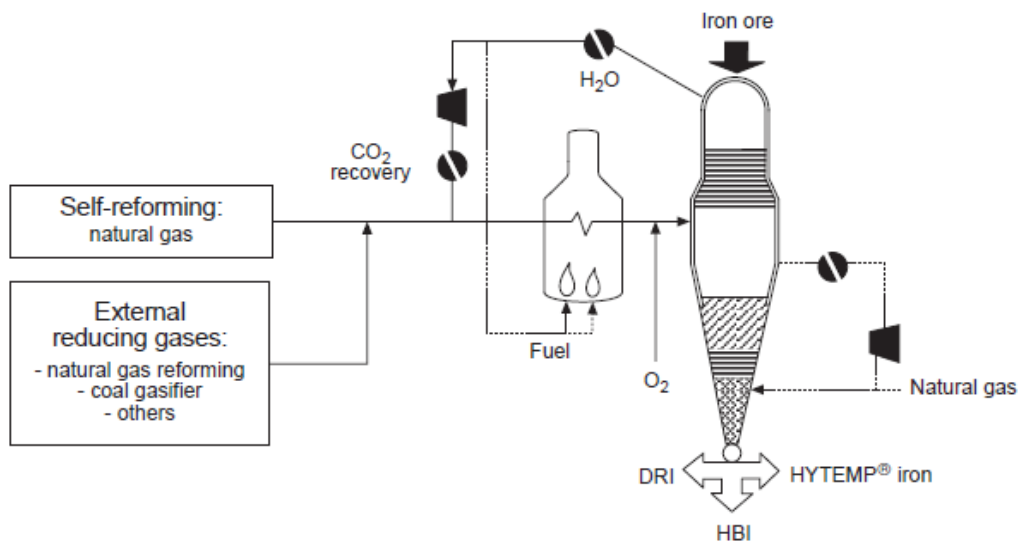
In 1979, a 9,000 tonnes/yr pilot plant investigated moving-bed technology as an alternative to fixed-bed technology. The first moving bed plant was a conversion of the second plant at Monterrey. The four fixed-bed reactors were replaced by the new shaft furnace and the process, then called HYL-III, was started up in Monterrey. The compatibility of the HYL and HYL-III processes made it possible to convert and modernize HYL plants to the new technology. Hylsa 3M in Monterrey and 2P in Puebla, Usiba in Brazil and P.T. Krakatau Steel in Indonesia were converted to HYL-III. Commercial DRI and HBI installations based on HYL technology currently in operation or under construction in Latin America, the Middle East. The flexibility of the process has allowed for numerous changes and upgrades, such that the generational label is no longer accurate. HYL has, since 1997, referred to its technology as simply the HYL process.

A general diagram of the HYL process, which has the flexibility to produce three different product forms, depending on the specific requirements of the user. Cold DRI is normally used in adjacent meltshops close to the direct reduction facilities. This product is also shipped and exported safely on a routine basis, following approved guidelines to avoid reoxidation. Hot briquetted iron can be produced from hot discharged DRI. HBI is generally exported. HBI production can also be used to process low quality, friable iron ores that cannot be used to produce cold DRI. Hot discharged DRI can be pneumatically transported directly to an EAF as HYTEMP[®] iron.

Reducing gases are generated by self-reforming in the reduction reactor, feeding natural gas as make-up to the reducing gas circuit and injecting oxygen at the inlet of the reactor. The partial oxidation of natural gas with oxygen in a combustion chamber generates reducing gases and

energy required for the reduction. Further cracking and reforming reactions are carried out in the shaft due to the catalytic effect of metallic iron. In addition to the self-reforming of reducing gases within the reduction reactor, the HYL process can use externally supplied sources of reducing gas. Some of the alternative gas sources include gases from natural gas steam reformers and from coal gasification units. Other reducing gas sources may be hydrogen, coke oven gas, gases from smelters gasifiers, and tail gases from other direct reduction processes.

Process description:



HYL process diagram.

The components in the reducing gas circuit are the reactor reduction zone, the top gas quenching/scrubbing system, the reducing gas recycle compressor, the CO₂ removal system and the process gas heater. The natural gas stream or reducing gas make-up is mixed with the reducing gas recycle stream from the CO₂ removal system. The reducing gas stream is heated to 933°C in the gas heater. Hot reducing gas is fed to the reduction zone at 5.5 kg/cm³ and flows counter to the solids.

Off gas from the reactor at about 400°C passes through a quenching/scrubbing system where water produced during reduction is condensed and removed along with most of the dust in the

gas. Scrubbed gas is fed to the CO₂ removal system by means of the reducing gas recycle compressor. Decarbonated reducing gas is mixed with the natural gas stream or reducing gas make-up to close the reducing gas circuit.

A belt conveyor transports iron ore pellets and/or lump ores to the top of the reduction tower where an automated system of bins and pressure locks receives the ore at atmospheric pressure in an open bin, pressurizes it in intermediate bins and charges it continuously to the reactor. Oxygen is removed from the iron ore in the upper section of the reactor by the reducing gases and the product is then carburized. A rotary valve at the outlet of the reactor regulates the continuous flow of the charge through the reactor. DRI is removed by automated pressurized bins and pressure locks.

In the production of cold DRI, the cooling gas circuit comprises the reactor cooling zone, the cooling gas quenching/scrubbing system and the cooling gas recycle compressor. The cooling gas is fed at the lower conical part of the reactor, at roughly 40°C, flowing counter to the descending DRI. The gas distribution is uniform throughout, with good contact between the gas and the solid material. The cooling gas leaves the reactor at the upper conical part at about 460°C and is then quenched/scrubbed by means of cooling water. Scrubbed cooling gas passes through the cooling gas recycle compressor for recycle to the reactor after being made up with natural gas injected to the cooling gas circuit for optimum efficiency and control of the cooling and carburization processes.

In the HYTEMP[®] iron process, hot DRI, at 700°C, is discharged continuously from the reactor to the pneumatic transport system. The product is transported by means of a reducing carrier gas to surge bins located above the meltshops for controlled feeding to the electric arc furnace.

For the production of HBI, hot DRI, at 700°C, is discharged continuously from the reactor to the hot briquetting machines arranged below. The HBI is cooled to about 80° in vibrating cooling conveyors using cooling water, and then discharged to the HBI transport conveyor.

The expected specific consumption figures in the HYL process for two typical production options are presented.

The self-reforming configuration has been in operation since April 1998 at Hylsa Flat Products

Division in Monterrey, Mexico. The plant has a nominal capacity of 700,000 tonnes/yr of DRI. Production during the first six months of operation has been at capacity, with metallization levels of 95% and carbon of 4% as iron carbide (cementite). Carbon levels up to 5.3% have been reached with no loss in metallization or productivity. Use of the product in a 150 tonne DC EAF has reduced overall cost/tonne of liquid steel by eliminating carbon additions, reducing electrical energy due to the faster melting reactions and improved slag foaming.

Table 11.4 HYL Process Consumption Figures

Item	Unit (/tonne DRI)	Self-Reforming	Natural Gas-Steam
		94% Metallization 4%C	Reformer 94% Metallization 2.2%C
Iron Ore	tonne	1.45	1.45
Natural Gas	Gcal	2.26	2.42
Oxygen	Nm ³	36.0	0
Electricity	kWh	100	60
Water Make-up	m ³	1.00	1.40
Labor	manhour	0.15	0.20
Supplies and Maintenance	US\$	3.15	3.50

PROCESS ADVANTAGES:

- Proven equipment performance (uses HYL II and HYL III reactor technology)
- Raw material flexibility
- Not sensitive to S in natural gas or ore
- No reformer – lower Capital costs
- High-energy efficiency (87% in comparison to 70% for most efficient other DRI plants).
- Hylsa claims lower overall operating costs (to be confirmed)

LECTURE 27 & 28

Smelting reduction process

Introduction

- Smelting reduction(SR), as the name suggests, involves both reduction and smelting, much like what happens in a blast furnace where both steps occur in a single reactor.
- The reduction step in SR does not proceed to the extent of 90-95% oxygen removal in the solid state, as it does in DRI production, but a certain fraction of oxygen ($\geq 70\%$) is removed in the solid state and the rest is removed in liquid stage.
- This is why the second practical step is smelting, i.e., melting involving chemical reaction.

Process classification:

The various SR processes developed so far can be classified into 3 Categories depending on the means by which the final reduction is effected

- Shaft furnace process
- Electric arc process
- Converter process
- **Shaft furnace process**

Shaft furnace units require metallurgical coke for a permeable bed since all the complicated and conflicting reactions occur within one large shaft. reduction by gas must be separated from the high temperature operations, so pre-reduced iron ore can either be charged from the top of the furnace or injected through the tuyeres at the bottom of the reactor, where it is melted and carburised to form hot metal. The gas generated during smelting leaves the shaft in the form of a (CO+H₂) mixture and is used for counter-current reduction of the ore in the low temperature reactor. The processes currently developed are:

COREX

Sumitomo's SC

Kawasaki Smelting Reduction Process

From the above only COREX has reached the commercial stage.

- **Electrical process**

These furnaces are often employed for melting pre-reduced metallic iron into hot metal. Heat is generated by electrical resistance in the slag. The electric power for melting can be generated in a gas-slag steam turbine system which utilises the off-gases from the process. The following are the advantages of these processes:

- High degree of metallization
- Null effect of fines on electrical resistance, and gas permeability

Various processes categorised under this group are:

- INRED
- ELRED
- COMBISMELT
- PLASMASMELT

- **Converter process:**

In these processes, energy is supplied by burning coal with oxygen in a converter. The large quantities of off-gases that are produced, can be used for the reduction of iron ore.

This system can achieve vigorous bath agitation and high production rates

Various drawbacks of these processes are:

- High refractory costs
- abnormal iron losses
- Reactive FeO rich slag

Advantages:

- High production rates
- Highly versatile
- Highly flexible
- Processes coming under this category are:
 - KRUPP-COIN
 - PCIG
 - REACTOR STEELMAKING

COREX process

The **Corex Process** is a smelting reduction process created by Siemens VAI as a more environmentally friendly alternative to the blast furnace. Presently, the majority of steel production is through the blast furnace which has to rely on ever decreasing amounts of coking coal.

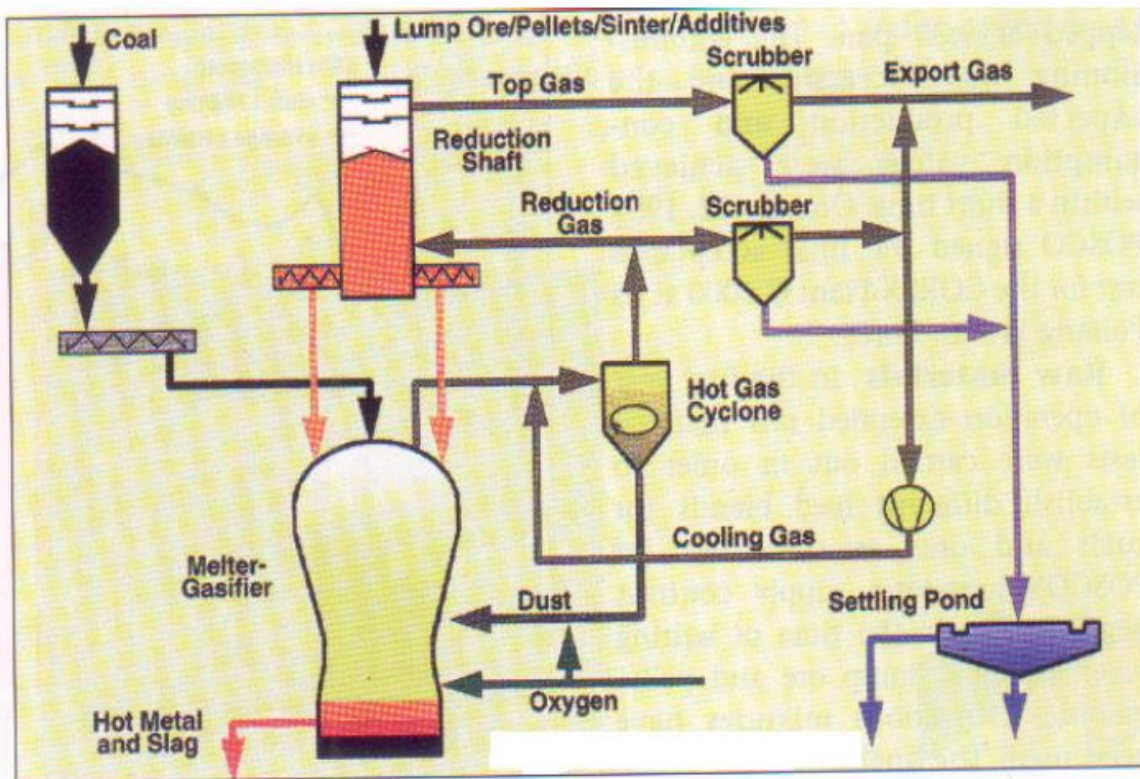


Figure: Schematic diagram of COREX process

That is coal which has been cooked in order to remove impurities so that it is superior to coal. In addition, the Blast furnace requires a sinter plant too in order to prepare the iron ore for reduction. Unlike the Blast Furnace, smelting reduction processes are typically smaller and use coal and oxygen directly to reduce iron ore into a usable product. Smelting reduction processes come in two basic varieties, two-stage or single-stage. In a single-stage system the iron ore is both reduced and melted in the same container. Meanwhile in a two-stage process, like Corex, the ore is reduced in one shaft and melted and purified in another. Plants using the Corex process have been put to use in areas such as South Africa, India, and China.

The Corex process consists of two main parts a *Reduction Shaft and Melter- Gasifier*. The main reagents for the Corex process are iron ore, noncoking coal, and oxygen. Unlike the Blast furnace the Corex process does not use a hot blast of nitrogen, thereby greatly reducing NOx gas emissions, but instead uses oxygen.

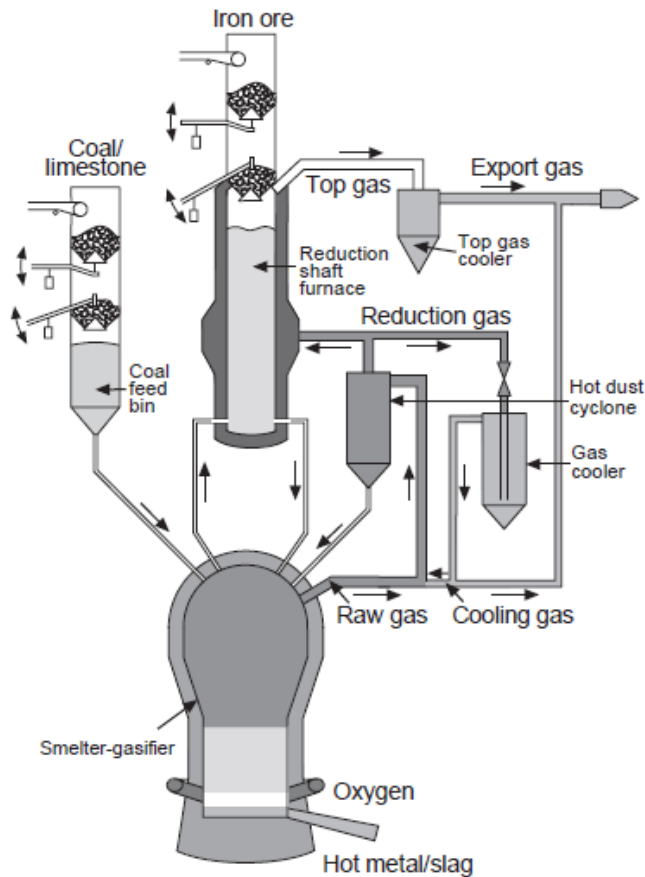


Figure: The flow sheet diagram of COREX

In addition, the Corex process can use iron oxides containing up to 80% lump ore and uses non coking coal directly as a reducing agent.

In the Reduction shaft the iron ore, along with limestone and dolomite additives, is added and then reduced by reduction gas into 95% direct reduced iron, DRI. The DRI is then redirected via six discharge screws into the melter-gasifier. The Melter gasifier has three main sections, the gaseous free board zone, the Char bed, and the hearth zone, and it has an effect on several stages in the process. First it serves to create the reduction gas by gasifying the coal with oxygen and then cooling it. After being reduced, the DRI is redirected to the char bed where the iron and slag

are melted and then directed to the hearth zone. The heat inside the metal gasifier keeps the amount of phenols small, keeping them out of the atmosphere. Meanwhile, carbon monoxide and hydrogen gas from the original gasification of the coal exit the gasifier while other byproducts are captured in the metallic slag. The rest of the hot gas is then cooled and sent into the reduction shaft resulting in the Corex export gas which is used to control pressure in the plant. Many of the gases resulting from this process can then be recycled or used to produce electricity.^[5] Dust particles also appear in these gases and the Melter gasifier recycles them with four dust burners.

The COREX process, shown schematically in Figure below, is a two-stage operation in which DRI from a shaft furnace like the Midrex and HYL process is charged into a final smelting gasifier. Reducing gas for the shaft furnace is produced by partial combustion of coal with oxygen in the fluidized bed of the smelter-gasifier. The energy needed to complete the reduction of the DRI and produce the hot metal and slag is provided by the partial combustion. The liquid products are tapped periodically and partially spent offgas from the shaft reducer is exported along with excess gas produced in the smelter-gasifier. The smelter-gasifier operates at 3–5 bars and comprises an upper fluidized bed zone at approximately 1500°C (2730°F) and a lower melting and liquid

collection zone at approximately 1550°C (2820°F). Coal and limestone

are injected into the freeboard above the fluidized bed zone where they are heated rapidly to 1000–1200°C (1830–2190°F). The volatile matter is driven off and shattered fixed carbon particles fall into the gasification zone where a gas with high oxygen content is injected through blast furnace-type tuyeres to burn the carbon to carbon monoxide. The exothermic combustion provides the energy to complete the reduction of the hot DRI and to melt the slag and hot metal. The gas leaving the

smelter-gasifier is cooled to 800–900°C (1470–1650°F) and cleaned in a hot cyclone to recycle entrained fines. A portion of the clean gas is then introduced into the shaft furnace as reducing gas containing more than 94% CO plus H₂. The remaining gas is mixed with the cleaned offgas from the shaft furnace and the mixture used as export fuel gas.

COREX uses approximately one tonne of coal per tonne of hot metal, with approximately 45% of the total energy input used in ironmaking and the rest going to export fuel gas. The hot metal produced

has carbon and silicon contents similar to blast furnace hot metal; however, the sulfur content is much higher because nearly all of the sulfur in the coal enters the slag and hot metal. In this connection, organic sulfur in the coal gasifies and is absorbed by the DRI and returned to the smelter-gasifier as iron sulfide.

The first commercial scale COREX plant was built for ISCOR in South Africa during 1985 through 1987 by Voest-Alpine Industrialanlagenbau. The plant, rated at 300,000 tonnes per year, was commissioned in November 1989 after testing, modification and operation following the startup in 1987. Since commissioning, the COREX plant at ISCOR has performed very well at the design specifications. A second COREX plant was constructed for POSCO at their Pohang Works

in Korea in 1998.

Experience of COREX at JSW

Total fuel rate requirement = 950 kg/thm. Out of which ~200 kg coke is required.

_ All the non-coking coal is imported. NCC of very high VM or very low FC cannot be used.

_ ~ 100% pellets are charged.

_ Oxygen requirement = 550 Nm³/thm (very high!)

_ Corex HM ~ BF HM:

_ Typical HM Composition: C~ 4%, Si=0.5-0.9%, S=0.025-

0.07%, P=0.13-0.19%, temp= 1480-1515°C.

Advantages

There are many advantages to the Corex Process:

- carbon dioxide emissions are up to 20% lower than with the conventional blast furnace,
- the Corex process produces far less SO₂ and dust than the blast furnace.

- In addition Corex plants do not release as many phenols or sulfides limiting water contamination.

Disadvantages

There are drawbacks:

- For example, at the JSW Steel plant in India it was found that to be viable the Corex process still needed about 15% coke.
- Furthermore, it has also been found that Corex plants require large amounts of oxygen which can be expensive.
- Also the export gas can make the process highly inefficient.
- However, this particular problem can be mitigated by using the export gas in electricity production

Hi –smelt:

The cooperative efforts of CRA and Midrex in the early 1990s led to the construction of a 150-tonne per day pilot plant in Australia. The original Hismelt reactor differed from DIOS and AISI in several major ways, in that it was a horizontal vessel into which coal and pre-reduced ore were injected directly into the metal bath. The final reduction reactions took place in the metal phase; in DIOS and AISI the medium for reduction and heat transfer is a deep slag bath.

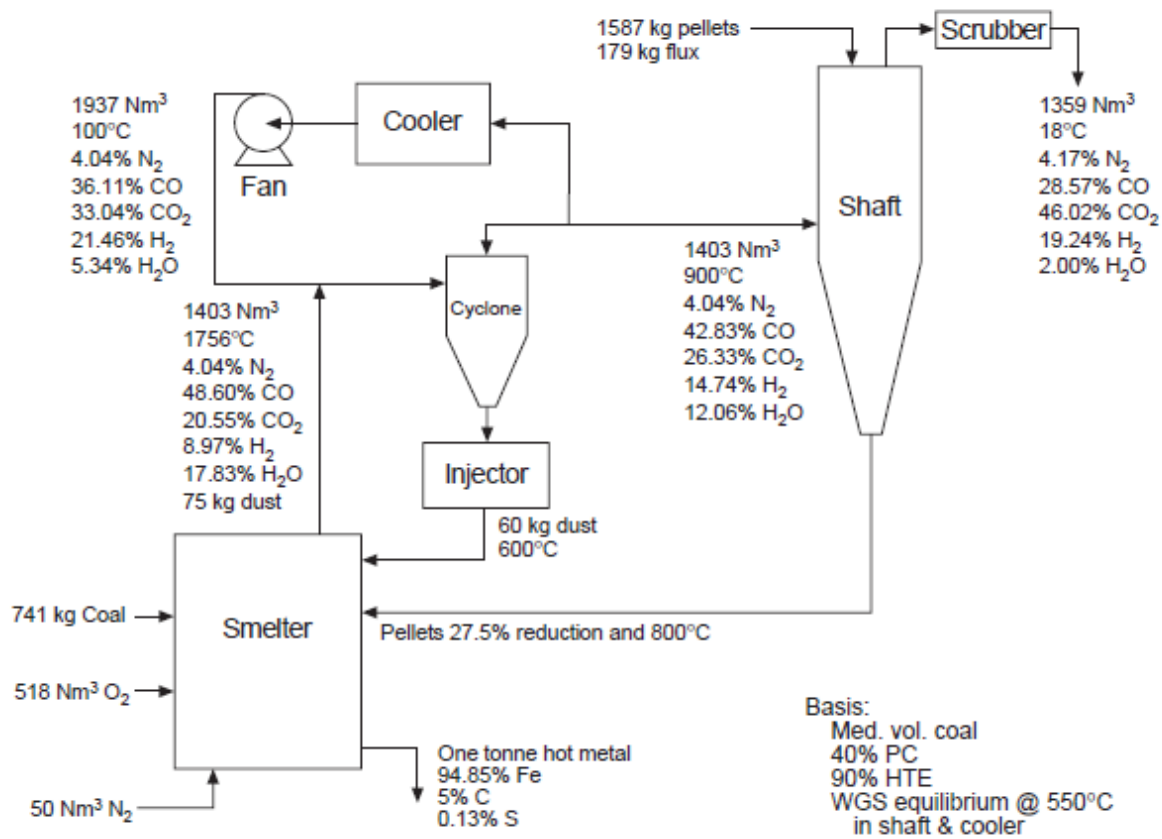
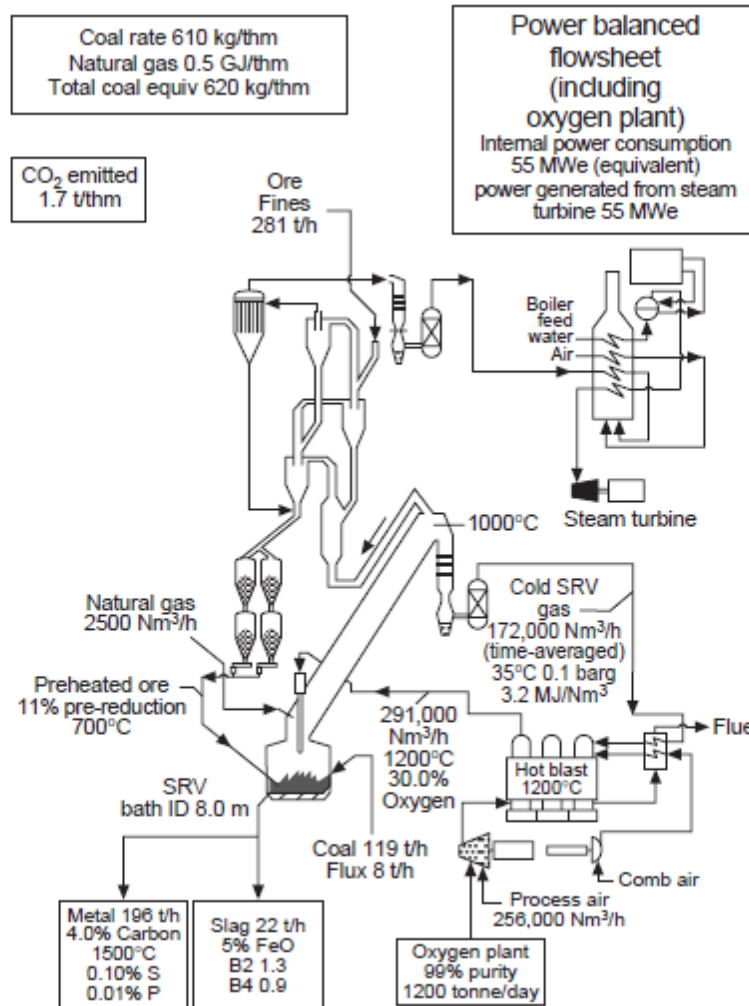


Figure: Schematic Diagram of the AISI process

Also, in the original HIs melt, devolatilization of the coal in the iron bath caused a relatively large injection of metal droplets into the freeboard where they picked up energy from post-combustion reactions for transfer back to the bath. In 1998 HIs melt adopted a more conventional vertical reactor design. Coal and pre-reduced iron ore are injected into the metal phase, where rapid dissolution and smelting occur. Offgassing generates a large liquid fountain with strong mixing of metal and slag. Oxygen enrichment of the hot blast is possible as back-reaction between the hot blast and dissolved carbon in the bath is minimized by the high slag volume in the liquid metal fountain. Post-combustion in the range of 50–75% is routinely achieved.



Schematic diagram of the Hismelt process.

A schematic diagram of the vertical steel refining vessel (SRV) Hismelt process with inputs and outputs is presented in Figure above.

ROMELT:

The ROMELT process was developed in Russia in the 1980s. A major feature is there is no prereduction process step as shown in Fig. 11.20. The smelter has a water-cooled roof and sidewalls in contact with slag and conventional refractories in contact with the metal. A mixture of air and oxygen is injected through two rows of tuyeres. Coal and ore are fed by gravity. The system, simple and robust, has produced hundreds of thousands of tonnes of metal and runs continuously for weeks. ROMELT consumes more energy than other smelting processes due to the lack of pre-reduction and extensive water cooling.

References :

1. A. K. Biswas, Principles of Blast Furnace Iron Making, SBA publication, Calcutta, 1999
2. Ahindra Ghosh and Amit Chatterjee: Ironmaking and Steelmaking Theory and Practice, Prentice-Hall of India Private Limited, 2008
3. V.R. Tupkary and R.H. Tupkary, Introduction to Modern Iron Making, Khanna Publishers

Questions for practice

Draw a neat sketch of B/F with labeling.

What is available base?

Give 3 functions of coke in a B/F.

What is the working principle of Dust Catcher?

What is “Dead man zone” in side B/F?

B/F is the best place for Sulphur removal- Justify.

In B/F hearth firebricks are used as facing lining.

Hematite is more reducible than magnetite-why?

CO is the best reducing agent in carbothermic reductions-justify.

Basicity of bosh slag is higher than final slag-why?

Compare and contrast Direct and Indirect reduction. State their importance in B/F iron making.

Describe the function of a B/F stove

What do you mean by Sintering? Describe the sintering process in Dwight Lloyd Sintering m/c.

What is “hanging”? Mention its causes and remedies

Why is DRI called Sponge Iron? Give the physico-chemical reactions of DR process. With a neat sketch describe the SL/RN process in brief..

What do you mean by Boudouard equilibrium? Draw the Fe-O-C equilibrium diagram overlapping the Boudouard curve and mention its role in iron ore reduction in side the blast furnace

Briefly describe the reactions taking place in the stack of a blast furnace.

Differentiate between Coal based and Gas based DR process.

Draw a neat sketch for simplified material flow in and out of a modern B/F. Give the composition of a typical Indian pig iron.

Briefly describe the three stage gas cleaning system in blast furnace plant with suitable sketches

Describe the Bell less charging in B/F

Write short notes on:

Slag granulation

PCI

Coke reactivity test

Ferro-coke