COMPUTAION OF DIFFERENT TECHNO-ECONOMIC PARAMETERS WITH VARYING INPUT MATERIALS TO PRODUCE HOT METAL THROUGH BLAST FURNACE ROUTE

A Thesis submitted in partial fulfilment of the requirements for the award of degree of

MASTER OF ENGINEERING IN METALLURGICAL ENGINEERING

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Declaration of Originality and Compliance of Academic Ethics

I hereby declare that this thesis contains literature review and original research work by the undersigned candidate, as part of his *"Computation of Different Techno-Economical Parameters with Varying Input Materials to Produce Hot Metal through Blast Furnace Route"* studies.

All information in this document have been obtained and presented in accordance with academic rules and ethical conduct.

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This is to certify that the entitled **'***Computation of Different Techno-Economical Parameters with Varying Input Materials to Produce Hot Metal through Blast Furnace Route***'** has been carried out by **Mr. Sudhanshu Kumar Singh (Examination Roll No: M4MET19002, Registration No: 140891 of 2017-18)** under my guidance and supervision and accepted in partial fulfilment for the degree of Master of engineering in Metallurgical Engineering from Jadavpur University. To the best of our knowledge the contents of this thesis or any part thereof have not been previously submitted for the award of any degree or diploma.

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The foregoing thesis is hereby approved as a creditable study of an engineering subject and presented in a manner satisfactory to warrant acceptance as pre-requisite to the degree for which it has been submitted. It is understood that by this approval the undersigned do not necessarily endorse or approve any statement made, opinion expressed or conclusion drawn there in but approve the thesis only for which it is submitted.

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<u>ACKNOWLEDGEMENT</u>

<u>ABSTRACT</u>

The demand of steel has increased over the days in global market and Indian steel market plays an important role in fulfilling the need of steel. To produce steel, use of iron ore fines is very important. So agglomeration plant has been introduced to use fines in the form of sinter or pellet. At present several Indian steel industries are using different combination of sinter, pellet and lump ore as iron ore burden in blast furnace. Several works have been done to study the effect on the furnace parameters due to charging of different burden percentage in blast furnace and several computation models have also been created to see the effect on the performance of the blast furnace. In this thesis an successful attempt has been made to create a simple excel sheet model to calculate the burden required for the fixed amount of hot metal production by proper mass balancing. Also calculation of weight of coke, coal and volume of blast air required for production of hot metal has been done. After that output material like slag rate, blast furnace off gas have been calculated. Also fluxes or additive required to maintain required slag basicity has been calculated and percentage heat lost or heat remained unused has been shown by proper heat balance. The effect of variation of burden material percentage on these calculated values has been discussed elaborately and also an attempt has been made for the selection of more economically acceptable burden percentage by cost calculation and by comparing net cost difference of different burden percentages.

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CHAPTER 1:

INTRODUCTION

1. <u>INTRODUCTION:-</u>

India is the 2nd largest producer of steel after Chania in the world and occupies sixth position in world's iron ore reserves. The national steel policy aims at 230 million tonnes (MMT) finished steel by 2030, for which approximately 250 MMT of quality iron ore (+63% Fe) is required. Besides that approx. 130 MMT of quality lump ore is required to meet the export commitments. All together 380 MMT of calibrated lump and fines are required which corresponds to iron ore mining of approx. 400 MMT of run-off-mine (ROM) thereby generating approx. 120 MMT of fines & blue dust every year [1].

Hematite and magnetite are the prominent iron ore forms found in India. Of these, hematite is considered to be the most important iron ore because of its high grade quality and lumpy nature, which is consumed by large number of pig and sponge industries in India. Magnetite deposits are not exploited so far for domestic use on account of their poor grade lumps (40% Fe). However, it can be utilized after beneficiation at a finer size followed by pelletization. Almost all of the present day production comes from hematite reserves. The overall hematite reserve in the country is the medium grade (+62%Fe) and accounts for around 28% of the total iron resource of the country. Domestic iron ore production is mainly in the form of lumps and fines in the ratio of around 2:3. Of course, domestic consumption in iron and steel making is only around 40-45% in the form of lumps and sinters, the remaining is exported. The bulk (around 90%) of the iron ore fines get exported , as they cannot by utilised in iron making without agglomeration [2].

As the demand of iron ore is increasing, it is necessary to recover the resources from the mines wastes or lean-grade iron ore through beneficiation. In that case the feed material for sinter or pellet making can be recovered from the wastes or low grade ores. Million tons of useful minerals are discarded in terms of fines and slimes every year, which leads to a rise in mines waste and environmental degradation [3].

The eastern region itself accounts for the 55% of the total Iron ore of our country. The share of lumps in total iron ore production has been about 40 % with the rest being accounted for fines and concentrates. Iron Ore Fines/blue dust cannot be charged in the blast furnace directly since they block the passage for ascending gas inside the feed. Moreover, the excessive fines generated from the iron ore crushing units are mostly stored as waste. It has been found that

the growth rate of steel industry was degraded from 12.7% in 2009-10 to 4.4% in 2011-12. Amongst the various reasons for this degradation, non-availability of quality iron ore feed was the major reason. Of late very small quantity of high grade fines have been utilized by few companies through sintering & pelletization process. Huge quantity of these fines have been exported. Also no effort have been made to utilize low grade deposits. Unless the generated fines wastes and low grade ore is effectively utilized, we cannot achieve our targets [4].

An advanced process synthesis for the utilization of iron ore within the framework of zero waste processing involves development of alternative mineral processing technologies for iron ore beneficiation such as magnetic carrier-technology, solid liquid fluidization, hybrid separation techniques, etc. and also development of agglomeration processes like Pelletization. Pellets are produced in the form of globules from very iron ore fines and are used for production of iron.

As of 2015-16 there exist pelletization capacity of about 85 MMT with a capacity utilization of 32.5% [1]. Impetus is given to pellet industry as it helps in mineral conservation by acting as direct feedstock in blast furnace in place of high grade iron ore.

The blast furnace process has remained up to date and competitive owing to several innovative developments that have taken place over the years. These innovations have led not only to more efficient process technology, but also to design and engineering of the equipment involved. Hot metal production rate of 8000-10000 TPD, fuel rates of around 500 to 600kg/THM, productivity levels of 2.5 to 3 t/m3/d(based on inner volume) and furnace availability ranging between 95% to 98% are the main results of improved process control. Lower silicon hot metal is now a days being produced consistently. Improvement in the raw materials used in terms of better quality coke with lower coke ash, increased used of agglomerated charge materials, use of high grade lime stones/dolomite with low alkali content etc. have lowered the slag rate, which has a directed bearing on the blast furnace productivity and fuel rates [5].

This is the reason, blast furnace has been in existence for the longest period and still remains dominant. Many of the alternative iron making process that have emerged can only complement the blast furnace in the years ahead. It is evident that even after these technologies are fully established in the coming years, the blast furnace will continue to be the principle method of iron making.

Heat energy analysis is a major field of study for the proper operation of a blast furnace. Heat balance is an account of the input and output of heat required in a process, which follows the

first law of thermodynamics. A proper heat balance not only helps to predict the efficiency of a furnace, but also to eliminate any excessive fuel wastages. Reduced fuel requirement not only reduces production costs, but more importantly saves a portion of rapidly depleting natural resources. To get a proper heat balance, a proper material balance is a necessity. A material balance is simply an account of input and output of mass, governed by the law of conservation of mass. A proper material balance will provide accurate quantitative values, thereby simplifying calculations in each step of the heat balance. Apart from all of this, a detailed study of each input and output components of the blast furnace has to be carried out, which includes: 1. Fuel Supplied; 2. Combustion Air or Hot Blast; 3. Blast Furnace Reactions; 4. Hot Metal; 5. Volatile Matter in Input Materials; 6. Moisture in Input Materials; 7. Blast Furnace Gas; 8. Dust Collected; 9. Blast Furnace Slag; 10. Cooling Water Supplied; 11. Other Heat Losses (Heat Loss through the tuyeres, Conduction, Convection, Radiation etc.) [6].

In this thesis, an attempt has been made to calculate the quantity of all input materials to produce a fixed amount of hot metal by proper material balance. The quantity of other output materials will also be computed after proper heat balance and blast furnace off gas generation calculation. A comparison of net cost difference between output and input material of blast furnace will also be carried out for different percentage of iron bearing input material.

CHAPTER 2:

SCOPE OF WORK

2. SCOPE OF WORK:-

Sinter, pellet and the lump ores are the three different forms of iron ore normally used in the blast furnace burden. Now a days the different forms of iron ore charge materials are used in blast furnace burden which can vary from zero to hundred percent with adjustment of the furnace parameters. There is no standard percentage for the choice of materials. The coke and coal required, slag rate, productivity and heat loss all are related to each other and depends upon the material charged into the blast furnace. The choice of materials depends on several factors varying from plant to plant. Some of the factors influencing the choice of materials are given below.

- Availability of the different forms of iron ore charge material of correct proportion depending upon the composition.
- Type of iron ore available in the mine or nearby area of the blast furnace.
- Availability of captive sinter plant.
- Metallurgical properties of forms of iron ore charge material such as the reducibility and softening properties.
- Chemical properties of the forms of iron ore charge material.
- Relative cost of the forms of iron ore charge material.
- Effect of the use of forms of iron ore charge material on the overall production cost of hot metal.
- Possibility of adjustment of various furnace parameters such as distribution pattern.
- Possibility of adjustment in process control parameters.
- Facilities available in furnace stock house.

So after considering these parameters the weight. of burden can be calculated by mass balancing to meet the hot metal output percentage and also coke and coal required can be calculated by stoichiometry. Now by varying the burden percentage, one can find out the optimised value of the burden for fixed hot metal composition and how much additives required to maintain the basicity of slag. Optimisation can be done to reduce the coke and coal consumption and slag rate with constant hot metal output and to increase the productivity because it directly reflects the cost of operation. Blast furnace off gas can be calculated and heat taken by the gases can also be calculated. Hence the main objective is to have trouble free blast furnace operation at the lowest possible cost of production of hot metal with fixed composition.

CHAPTER 3:

METHODOLOGY

3. METHODOLOGY:-

Calculation of coke and coal required, slag rate and heat balance will be done in the following steps and all the calculation will be validated by comparing with industrial output data.

3.1 CALCULATION OF COKE, COAL REQUIRED AND SLAG RATE:-

1. All input raw materials data (sinter, pellet, lump ore, coke, coal, quartz, limestone) and hot metal output component will be taken from industry.
2. Now the hot metal output weight will be fixed to1000kg (one tonne).
3. For 1 THM hot metal output sinter pellet and lump ore requirement (in kg) will be calculated by mass balance.
4. Now several assumptions will be considered for the calculation.
   a. 65% indirect reduction, 30% direct reduction and 5% reduction by H_2 inside the blast furnace. [7]
   b. Loss of Fe in slag is 0.2% and in flue dust is 0.5%, as taken from industry.
   c. According to the industrial standard, Slag basicity is fixed in between 1 to 1.15 [7].
5. Now the carbon required in the blast furnace will be calculated by stoichiometry and after that the quantity of coal and coke charged in the blast furnace will also be calculated.
6. Now weight (wt.) of different component of slag will be calculated.
7. Then by summing up all slag component, slag rate will be calculated.
8. Weight of additives or fluxes required to maintain slag basicity will be calculated by using solver.

3.2 CALCULATION OF BLAST AIR AND BLAST FURNACE OFF GAS:-

1. First all metalloid oxygen will be calculated.
2. According to industrial standard, CO/CO2 ratio will be taken as 1.08(range in between 1 to 1.3) [5].

3. Now according to the above assumption, O_2 required will be calculated. After that O_2 required from blast air will be calculated by subtracting the metalloid oxygen.

4. 21% O_2 and 79% N_2 is in the blast air. So total blast air required will be calculated.

5. Roughly 4% of total oxygen and 30% of total hydrogen will go into the BF gas (taken from industry).

6. Now after summing up the value of CO, CO_2, N_2, unreacted O_2, and H_2 total off gas and there percentage in the off gas will be calculated.

3.3 HEAT BALANCE CALCULATION:-

1. First all the calculated data will be converted into moles/THM.

2. Now the standard value of latent heat, sensible heat and reaction heat will be taken. By using these values, heat consumed during the process will be calculated.

3. Now the total heat generated will be compared to the total heat consumed.

After following all these steps, the proportion of the burden charged into the blast furnace, will be changed to optimise the working condition.

3.4 COST CALCULATION:-

1. After following all the steps mentioned above, summary sheet will be prepared.

2. Then cost of all the input raw materials and output material will be taken either from the industry or from the online sources.

3. Now the difference between input raw material cost and output cost will be calculated

4. After that comparison of cost difference will be done for the different burden percentages.

CHAPTER 4:

LITERATURE REVIEW

4. LITERATURE REVIEW:-

IRON making is the pyro-metallurgical process of turning solid iron ore materials into liquid hot metal called pig iron typically saturated with carbon (5% wt). The process uses injection of air and metallurgical coke in a counter-current flow reactor called a blast furnace (BF). This is probably one of the metallurgical processes that has captivated the most attention of scientists and engineers in the past. The major chemical reactions occurring in the BF are (1) the combustion of metallurgical coke by hot air (potentially enriched in O_2) at the tuyere level, resulting in the production of a CO-rich reducing gas; and (2) the direct and indirect reductions of the iron-rich minerals present in the ore by metallurgical coke and the reducing gas, respectively. The composition of the available iron ore include other oxides such as SiO_2, Al_2O_3, P_2O_5, MgO, CaO, Na_2O, K_2O, and S. Minor chemical reactions such as the reduction of these impurities can occur and affect drastically the quality of the final pig iron product, the chemical nature of emissions as well as the productivity of the BF.

4.1 RAW MATERIAL:-

The raw materials of the blast furnace are (i) solids (ore, coke, flux) which are charged into the top of the furnace; and (ii) air which is blasted through tuyeres near the bottom of the furnace. Hydrocarbon additives (gas, liquid or solid) and oxygen are also introduced through the tuyeres. The solid raw materials consist of:

4.1.1 Iron Oxides: Usually hematite, Fe_2O_3, occasionally magnetite, Fe_3O_4. In modern operations the iron oxides are added in the form of (i) 1-2-cm diameter pellets produced from finely ground, beneficiated ore (5-10% $SiO_2+Al_2O_3$, remainder Fe_2O_3); (ii) 1—3-cm chunks of sinter produced from ore fines; and (iii) sized (1—5 cm) direct lump ore. The charge of a particular blast furnace may contain two or all three of these iron oxide forms [8]

4.1.1.1 Lump Ore: All the lump ore either contain hematite or magnetite with iron contents ranging from 55% to as high as 66.5%. The higher the iron content in the feed, lower is the slag volume generated in the blast furnace, which automatically increases the productivity and reduces the coke rate. Therefore higher iron content in the feed are preferred. For the same reason iron ore contains less than 55% iron are often beneficiated to enrich the iron content to acceptable level.

4.1.1.2 Sinter:- Sinter by itself is sufficiently heat resistant, though it is produced at relatively low temperatures(1050^0C) without complete melting of iron ore partical.it is a product of recrystallization following partial melting of iron ore fines and slag formation. The quality of sinter is governed by the nature of the materials that are present after partial melting, fusion and subsequent recrystallization. A wide variety of minerals are found in iron ore sinter such as magnetite, wustite, ferrites, and in silicate like fayalite, olivine, glass etc. depending on the materials that are fed and the sintering condition.

Whenever the % of sinter used in the blast furnace is increased, the sinter chemistry has to be adjusted so that the total amount of fluxes (CaO/MgO) required in the blast furnace slag comes predominantly, if not wholly, through the sinter in the burden. The best situation exists when the entire flux is catered by the sinter. The process of making sinter is known as sintering.

4.1.1.3 Sintering:-

The sintering process is based on treating a mix (mineral fines, return fines, fluxes, etc.) layer in presence of coke dust to the action of a burner placed in the surface of the layer. In this way, heating takes place from the upper to the lower sections. The mix layer rests over a strand system and an exhausting system allows to the whole thickness to reach the suitable temperature for the partial melting of the mix, and the subsequent agglomeration. In the Dwight-Lloyd system, the sintering grate is a continuous chain of large length and width, formed by the union of a series of pallet cars that make the sintering strand (Figure 1). Each pallet car passes below a charging hopper where is charged firstly by material of coarse granulometry (10-20 mm) in a layer of 3-6 cm that forms the hearth layer composed mainly by sinter. The hearth layer protects steel grates from over-heating during the sintering process. Secondly, and over the first layer, a second layer of fine material (0-8 mm) is charged. This second layer is formed by fine mineral, return fines, fluxes and coke. Then, the pallet car passes bellow an initializing furnace, where the combustible ignition takes place in the surface of the mix. At the same time, the mix is subjected to downdraught suction through the load.
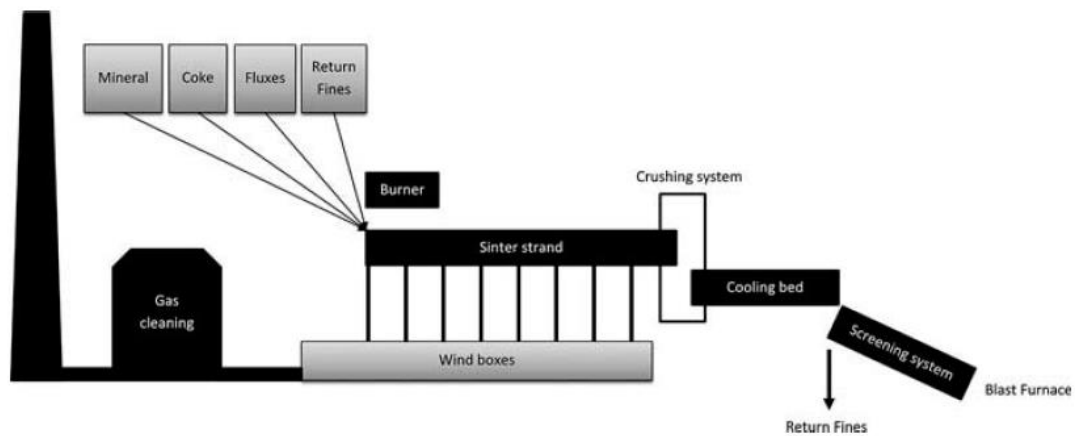
FIGURE.1 Sinter Making Process [9]

The pallet car continues the process and the combustion progresses in the direction of the gas flow. In this way, the sintering process takes place. The combustion process does not happen simultaneously in the whole thickness of the bed. On the contrary, combustion happens as a horizontal layer that moves vertically through the bed. The thickness of this layer is a small fraction of the bed. Permeability is a quality requirement for the load, and for that reason the granulation process is previously used (permeability is improved during granulation) [10]. In the region above the combustion zone, very hot sintered product heats the air that passes through this layer. In this way, pre-heated air arrives to the combustion area. The heat of the air/gases previously heated is absorbed in these cold sections, causing preheating of the load and evaporation of the water. In this context, high temperatures that cause partial melting are reached, and the sintering process takes place. This high thermal efficiency is caused by heat accumulation in a partial layer of the load called sintering zone or flame front.

Once the end of the strand is reached, the sintered material is discharged and subjected to cooling, crushing, and screening. Obtained product can be divided into three granulometric fractions:

- Between 0 and 5 mm (0 and 10 mm, according to Gupta 2010), called return fines, which are sent to the feeding Hoppers.

- Sinter with a granulometry within 5 and 20 mm is used as hearth layer in the sinter strand.

- More than 20 mm are directly sent to the blast furnace. Maximum grain sizes <50 mm [11].

Return fines are unavoidably formed during the sintering process, and are recycled back into the sintering process, making up 30 to 40% of the iron bearing materials.

4.1.1.4 Pellets:- Quality of the pellets is influenced by the nature of the ore or concentrate, associated gangue, type and amount of fluxes added. These factors in turn result in the variation of physicochemical properties of the coexisting phases and their distribution during the pellet induration. Hence properties of the pellets are largely governed by the form and degree of bonding achieved between the ore particles and the stability of these bonding phases during reduction of iron oxides in the blast furnace. Since the formation of phases and micro structure during induration depends on the type and amount of fluxes added, there is an effect of fluxing agents in terms of CaO/SiO_2 ratio and MgO content on the pellet quality. The swelling index of pellets is an Important of metallurgical property. Swelling indicates volume change of pellets during reduction. The volume expansion of pellets during the reduction results in lower compressive strength of pellets. High swelling inside the furnace causes increase in volume of the pallet which in turn decreases voids in charge. This Impedes gas flow in the furnace and results into pressure drop. This in turn causes burden hanging and slipping inside the blast furnace. The addition of dolomite is favourable for the improvement in swelling property of pellets. Maximum allowable swelling of pellets for the blast furnace ranges from 16 % to 18 %. Acid pellets (DRI pellets) and MgO free pellets exhibit higher swelling.

Fluxed pellets can be produced as equivalent to the best sinter in terms of reducibility and softening meltdown properties and are superior in terms of strength and low temperature breakdown. Fluxed pellets exhibit good strength, improved reducibility, swelling and softening melting characteristics. Because of these properties fluxed pellets give better performance in the blast furnace.

a. Acid Pellets – Basicity of acid pellets is usually less than 0.1. The fired pellet strength is, to a certain degree, due to hematite bridges of polycrystalline structure. These pellets normally have large volume of open pores. The reduction gas quickly penetrates through these pores into the pellet core and simultaneously attacks the structure in many places. This results into an early structural change which begins at low temperatures over the entire pellet volume.

b. Fluxed Pellets – These are also known as basic pellets. Basicity of fluxed pellets is greater than 0.1 and can vary. Basicity of normal basic pellets range from 0.1 to 0.6 and have low CaO percentage. During the firing of these pellets, a glassy slag phase consisting of SiO_2, CaO, and Fe_2O_3 of varying percentage is formed. Due to increased flux addition, there is formation of some slag and due to it, there is to a certain extent slag bonding with iron ore crystals. High

basicity pellets have a basicity level greater than 0.6. These pellets contain higher level of CaO. These pellets not only have glassy phase consisting mainly of SiO_2, CaO, and Fe_2O_3, but also calcium ferrites ($CaO.Fe_2O_3$). During firing of these pellets, the availability of CaO considerably favours the crystal growth of hematite. These pellets normally have a high mechanical strength after pellet firing.

4.1.1.6 Pelletization:-

Pellets are spherical balls formed by the agglomeration of natural or ground iron ore fines in the presence of moisture and binder. After subsequent induration at around 1250°C, they become suitable feed for iron making processes like the blast furnace and direct reduction furnace. A typical process flow sheet of pelletizing is shown in **Fig.2** [12]. Pelletizing process consists of three main stages;

a) Raw material preparation

b) Formation of green pellet

c) Pellet induration (hardening)
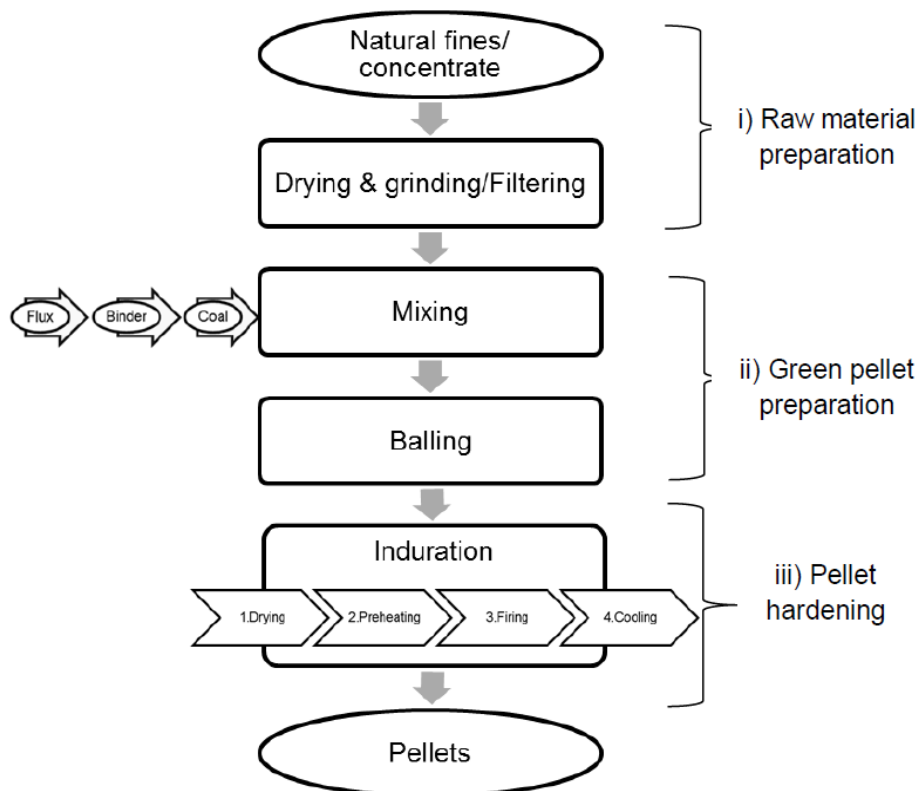
Figure2. Schematic Diagram of Pelletizing Process

Pellets are generally made by two processes:

a. Straight Travelling Grate Process:

It was developed by former Lurgi Metallurgie accounts for world's major installed capacities. In this process a double deck roller screen ensures right size of green pellet (8-16 mm) is evenly distributed across the width of the travelling grate. The grate carries the green pellets on a bed 300 mm to 550 mm thick through a furnace with updraft drying, downdraft drying, preheating, and firing, after firing heating zones. The flow chart of straight travelling grate process is given in the figure 3 [12].
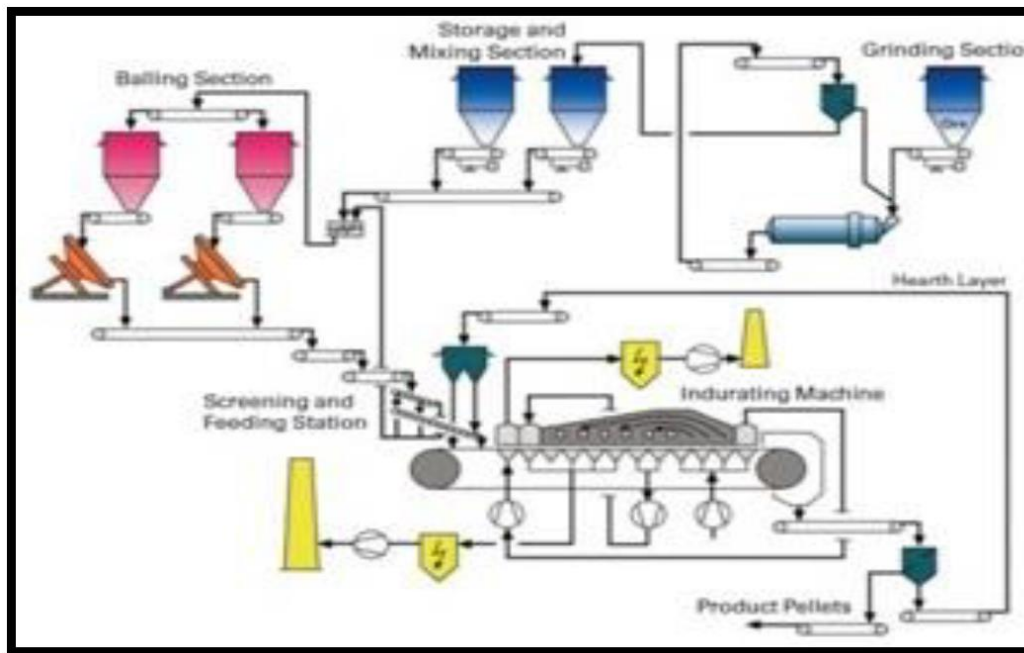
Fig.3: Straight Travelling Grate Process.

b. Grate Kiln Process:

It was developed by Allis Chalmer and the first plant on this technology was constructed in 1960. In the Grate-Kiln process (Fig 4) the traveling grate is used to dry and preheat the pellets. Material moves on straight travelling grate till it attains the temperature of 800°C to 1000°C. Thereafter it is transferred to refractory lined rotary kiln for induration where the temperature is further raised up-to 1250 – 1300°C. At 800°C, the Magnetite iron Ore gets converted into Fe_2O_3 in an exothermic reaction. The liberated heat hardens the green balls which is helpful to withstand the tumbling impact in the rotary kiln. A circular cooler is used for cooling of the fired pellets. The flow chart of Grate kiln process is given in the figure 4 [13].
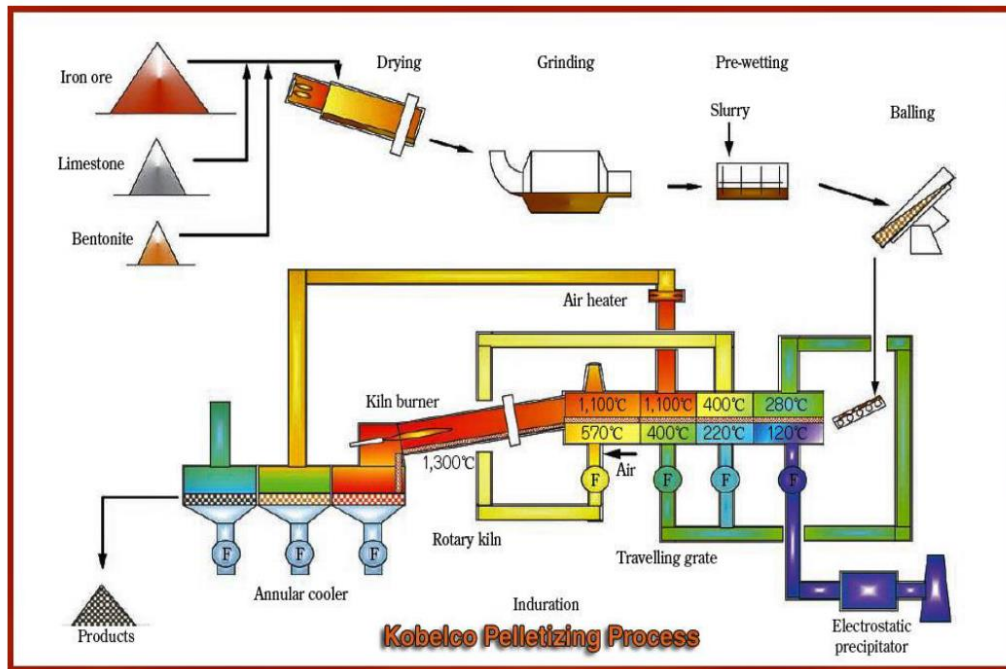
Fig.4: Grate Kiln Process

4.1.1.7 Advantages and Disadvantages of Pellets [14].

ADVANTAGES

1. Minimum closure of pores by fusion or slagging; open pore system; very good reducibility due to high micro porosity (porosity 2.5:-35 percent). Slag bonded are less reducible;

2. Spherical (egg) shape and small uniform size (10-20 mm) give very good bed permeability. The shape, size and low angle of repose· give, minimal segregation and charge distribution in the furnace, extending more towards the axis;

3. More accessible surface per unit weight and more iron per unit of furnace volume because of high bulk density, 3-3.5 tonnes/m3 • Larger Surface and increased time of residence per unit weight of iron give better and solid contact and improved heat exchange;

4. High iron content and more uniform chemical composition because of fine grinding and upgrading during ore dressing. Hence lower flux and fuel requirement in the furnace. Lower slag volume;

5. high softening temperature, 1200-1350°C with narrow range; dolomitized pellets have a softening temperature about 80°C higher than the corresponding limey pellets;

6. High strength average compression strength about 150-250 kg more for acid pellets;

7. Heat consumption much less than that for sintering;

8. Ease in handling, e.g., by pneumatic methods;

9. Ideal for direct reduction processes;
10. Maintenance of size during reduction in the blast furnace, much more than in the case of ores and sinters especially in the lower temperature regions;
11. Acid pellets are stronger than fluxed sinters or pellets. Reducibility of, fluxed pellets is higher than that of fluxed sinter or acid pellets;
12. Fluxed pellets are more economical than fluxed sinters.

DISADVANTAGES

1. High cost of production due to grinding and firing especially with oil burners;
2. Swelling and loss of strength inside the furnace;
3. Sticking during firing;
4. Resistance to flow of gas more than that sinter for the same size range due to lower voids;
5. Difficulty of producing fluxed pellets;
6. Fluxed pellets break down under reducing conditions much more than acid and basic sinters and acid pellets;
7. Strong highly fluxed sinters, especially containing MgO, are being increasingly preferred lo pellet.

4.1.2 Coke:-Coke is one of the most important raw materials for the hot metal production through the blast furnace route.it often account for almost 60% of the cost of producing hot metal in blast furnaces. Coke plays three important roles in a blast furnace

1. Supply energy for the chemical reaction to occur;
2. Supports the burden, particularly at the lower region when the charge melts, providing permeability for the gases to pass through a packed bed of solids;
3. It carburises the hot metal bath prior to the steel making.

Coke making involves carbonization of coal under control condition in coke oven. The quality of coke expressed in terms of its room temperature and high temperature strength, reactivity, ash content and chemistry has a major influence in the determining the coke rate and productivity of a blast furnace. The quality of coke depends on the quality of coal and the pre-carbonization technique adopted prior to charging coal into coke ovens [15].

Also PCI (pulverised coal) is charged from tuyeres.

4.1.3 <u>Limestone</u>: CaO and MgO which flux the silica and alumina impurities of ore and coke to produce a low melting point (1600 K), fluid slag. CaO has the beneficial secondary effect of causing part of the sulphur in the furnace charge, introduced mainly as an impurity in the coke, to be removed in the slag rather than in the product iron. The CaO and MgO are charged in pre-fluxed sinter or as 2-5-cm pieces of limestone, ($CaCO_3$) and dolomite ($CaCO_3:MgCO_3$); occasionally in steelmaking slag.

4.1.4 Raw Materials Introduced Through the Tuyeres:

(a) <u>**Hot-Blast Air**</u>*:* Preheated to between 1200 and 1600 K and in some cases enriched with oxygen to give blast containing up to 25 vol. % O_2. The hot blast burns incandescent (1800 K) coke in front of the tuyeres to provide heat for:
 (i) Reduction reactions and
(ii) Heating and melting of the charge and products. High blast temperatures ensure that the pig iron and slag products of the furnace are well above their melting points [8].

(b) <u>Gas, Liquid or Solid Hydrocarbons</u>*:* Which provide additional reducing gas (CO and H_2) for the reduction process. Fuel oil and tar are the most common additives. Natural gas and powdered coal are also used.

Injection of hydrocarbon liquids, gases or solids through the tuyeres has also led to improved furnace productivities by lowering the requirement for coke in the solid charge. This leaves more space in the stack for ore and it leads to a greater reduction capacity. Hand in hand with hydrocarbon injection has come the use of hotter blast. Hot-blast air offsets cooling due to the entry of cold hydrocarbons into the tuyere zone of the furnace and it also adds enthalpy to the system. Blast temperatures in modern installations are typically 1300 to 1600 K [8].

(C) <u>Supply of Pure Oxygen</u>: Enrichment of the air blast with pure oxygen has been useful in improving blast-furnace performance. The oxygen replaces a portion of the air requirement and it lowers the quantity of nitrogen passing through the furnace. This partial elimination of nitrogen:

(a) Increases the flame temperature in front of the tuyeres thus permitting increased injection of cold hydrocarbons;

(b) Permits an increased rate of CO production and consequently an increased rate of ore reduction without increasing the total rate of gas flow through the furnace [8].

The latter effect (b) can be used to raise the production rate of the furnace without causing channelling and flooding.

4.2 BLAST FURNACE PRODUCT:-

4.2.1 Pig Iron:-The main product of the blast furnace, molten pig iron, is tapped from the furnace at regular intervals (or continuously in the case of very large furnaces) through one of several holes near the bottom of the hearth.

The composition of the pig iron from a particular blast furnace is chosen to meet the requirements of the steelmaking plant to which the iron is being sent. It is controlled by adjusting;

(i) Slag composition and

(ii) Furnace temperature (particularly in the lower half of the furnace).

The pig iron is transported in the molten state to the steelmaking plant where the impurity elements are removed to low levels by oxygen refining. In some cases the molten pig iron is desulphurized by treatment with calcium carbide or magnesium-coke prior to oxygen refining. The exact stipulation of hot metal quality varies from country to country and even plant to plant in the same country, depending on the type of raw materials, grade of steel that have to be made and type of final product etc. table 1 shows the typical range of hot metal composition stipulated for blast furnace in several countries.

Country elements	INDIA	JAPAN	USA	EUROPE
Carbon	4 to 4.5	4 to 4.5	4 to 4.5	4 to 4.5
Silicon	.7 to 1.3	.3 to .45	.4 to .7	.3 to .45
Manganese	.04 to .1	.2 to .4	.5 to .75	.2 to .65
Phosphorus	.1 to .3	.095 to .105	.04 to .08	.06 to 1.5
Sulphur	.055 to .08	.02 to .025	.025 to 0.065	.02 to .065

Table 1. Typical Range of Hot Metal Composition. [5]

4.2.2. <u>Slag</u>:- When all the raw material and different types of fluxes are smelted in a blast furnace, hot metal is produced and the impurities in the feed is separated as slag. Thus slag is a product formed by the chemical combination of fluxes with the aluminates, silicates and other gangue constituents in iron oxide and coke and coal ash. A typical analysis of BF slag is [5];

CaO-40 to 45%, SiO_2-34 to 40 %, MgO-5-12%, Al_2O_3-12-20%, FeO-1-2%;

It is generally accepted that for smooth blast furnace operation, the basicity ratio, CaO/SiO_2, should be maintained in between 1.05 to 1.2 respectively and alumina content should not exceed 18% because many practical problem arises during the blast furnace operation.

<u>Usages of BF Slag</u>:- Depending on its chemical composition and physical characteristics after solidification, blast furnace slag can be put to various uses. Air cooled slag is allowed to flow into a slag is allowed to flow into a slag bank where it is cooled by air. After cooling, the slag is crushed, screened and then used as a filler between rail road sleepers, as a constituents of Portland cement, mineral wool, glass sand, ceramic etc.

If it is quenched in a pool water or cooled rapidly by using powerful water jets, it gets pulverised. A fine, granular, and almost fully non-crystalline powdery solid is obtained, which is used to produce blast furnace slag-cement, which exhibits excellent cementitious property.it can be also used as a soil conditioner, for making building blocks.

4.2.3 <u>Blast Furnace Gas</u>:- Gas which leaves through the gas-collection system at the top of the furnace. A typical modern furnace top gas composition is roughly [5];

21- 23% CO, 20-22% C02, 1-2% H2, 49-58% N2;

Almost 1500 to 1700 Nm3 of gas per ton hot metal is produced during blast furnace operation. It has a net combustion energy of about 3500-4000 kJ per Nm3 (about one-tenth of total energy input into any blast furnace). After removal of dust, this gas is burnt in auxiliary stoves to heat the air blast for the furnace. The dust is agglomerated by sintering or briquetting and recharged to the furnace or stockpiled for future use.

4.3 BLAST FURNACE:-

4.3.1 General Construction:- It is circular in cross section, and around 30m in height. The outer shell is made of steel. Inside the shell there is refractory lining. Nowadays the steel shell is welded construction rather than the earlier form of rivetted construction. The tall structure has recently been made free standing i.e. the only support is provided by the foundation. The furnace interior is broadly divided into;

- Stack whose wall slopes outwards going downwards
- Belly , the cylindrical portion below the stack
- Bosh, below the belly and sloping inwards going downwards
- Hearth, below the bosh and tuyere region and the cylindrical portion.

Pre heated air enters the tuyeres through the refractory lined bustle pipe, which is like a horizontal circular ring around the furnace. Iron notch is the tap hole for molten iron. It is kept sealed by refractory clay. For tapping, the clay seal is opened by a remote controlled mechanical device. Cinder notch is the hole for tapping molten slag. It is a above the molten iron in the hearth. The furnace is built on a massive reinforced concrete found.
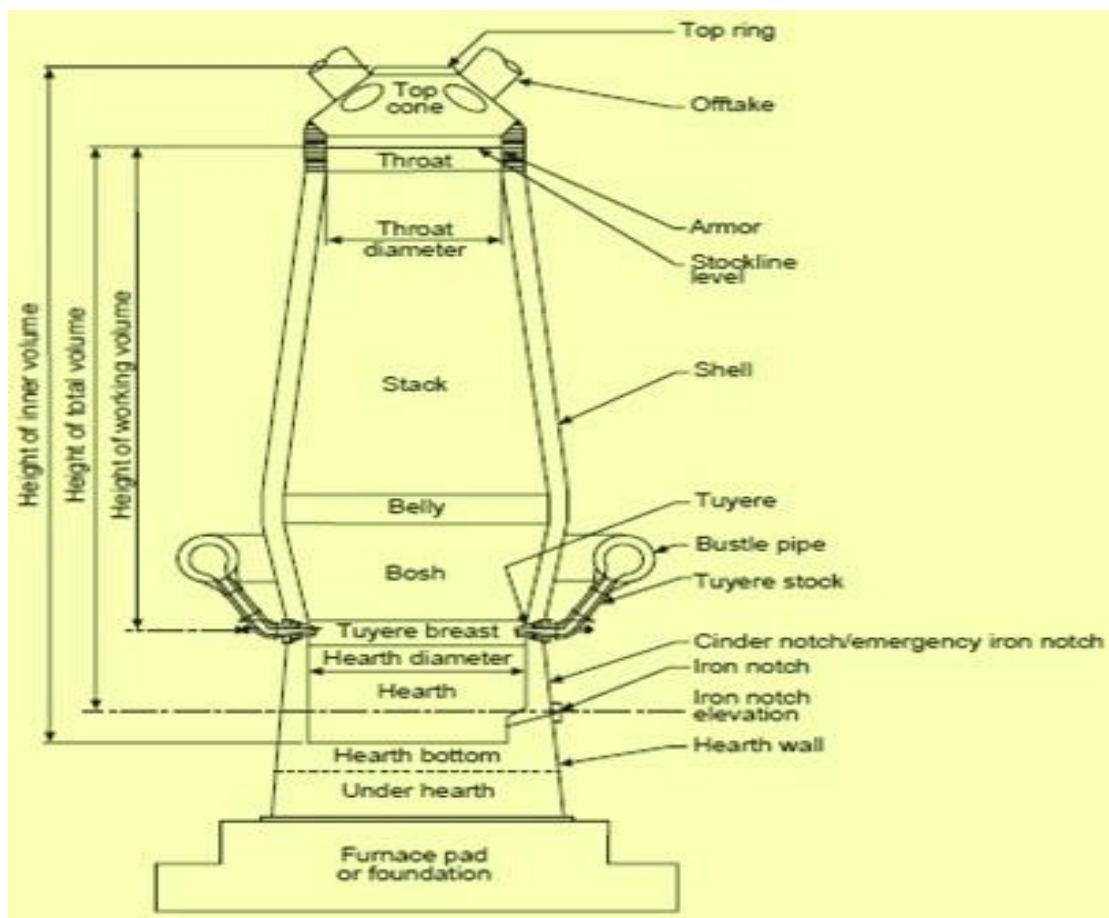
Figure.5: Blast Furnace Schematic View. [13]

4.3.2 Blast Furnace Operation:-

Solid raw material at room temperature at room temperature are charged from the top. Pre heated air above 1000-1100^{0}C is blown through its wall. Nowadays it is enriched with some pure oxygen and moisture. Most of the modern furnace are practising injection of pulverised coal as well. Exothermic combustion of coke and coal gasifies carbon into CO and also provides heat. The highest temperature zone of the furnace is at the level of tuyeres. As the reducing gas travels upward it heats up the solid charges as well as participates in various reactions at different zone of the furnace.

An ideal blast furnace is characterised by a well-developed thermal reserve zone (TRZ) and a chemical reserve zone (CRZ). Several investigators have experimentally determined the temperature and condition profile of gas phase in the stack by inserting thermocouples and sampling probes at various location. For solid data are more difficult to collect and hence are limited. Figure 6 shows an ideal sketch of the temperature distribution of gas and solid along the height of a blast furnace and also shows chemical reactions occurring in the three main temperature zone. The CRZ consist of the chemically inactive zone as well as indirect reduction zone and is a part of the TRZ. Some main features of blast furnace are as follows

- It consists of three zone, preheated zone, TRZ and CRZ, direct reduction and melting zone.
- The gas temperature is always higher than the solid temperature, thus allowing gas to solid heat transfer.
- The temperature of the solid remains constant at the TRZ zone.

Reactions in different zone [6].

Upper zone: solid raw materials are only preheated.

Middle zone (approx. temperature is 1000°C): mainly indirect reduction of iron ore and calcination of carbonates occur.

$3Fe_2O_3(s) + CO\ (g) \rightarrow 2Fe_3O_4 + CO_2\ (g)$ +12.47 kcal/mole;   (Exothermic)

$Fe_3O_4\ (s) + CO\ (g) \rightarrow 3FeO(s) + CO_2\ (g)$ -4.585 kcal/mole;   (Endothermic)

$FeO\ (s) + CO\ (g) \rightarrow Fe(s) + CO_2\ (g)$ +3.85 kcal/mole;

$3Fe_2O_3(s) + H_2(g) \rightarrow 2Fe_3O_4(s + H_2O(g) +46$ kcal/mole;

$Fe_3O_4(s) + H_2(g) \rightarrow 3FeO(s) + H_2O(g)$ -12.45 kcal/mole;
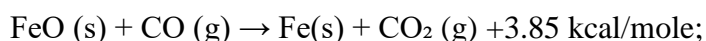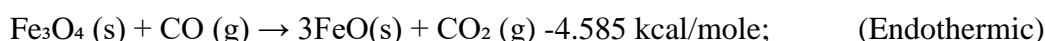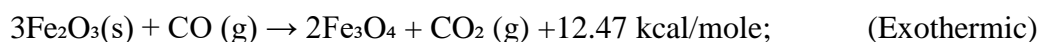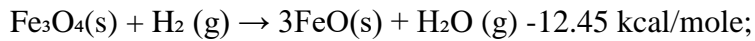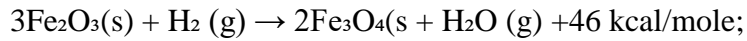
Lower zone (approx. temperature is $1400^{\circ}C$): here carbon is converted into reducing gases and also provided heat. Directed reduction of ores and also oxides of different elements are partially reduced.

$3Fe_2O_3(s) + C(s) \rightarrow 2Fe_3O_4(s) + CO(g)$ -26.95kcal/mole;

$Fe_3O_4(s) + C(s) \rightarrow 3FeO(s) + CO(g)$ -44.43 kcal/mole;

$FeO(s) + C(s) \rightarrow Fe(s) + CO(g)$ -27.48 kcal/mole;

$P_2O_5(s) + 5C(s) \rightarrow 2P(s) + 5CO(g)$ -206 kcal/mole;

$TiO_2(s) + 2C(s) \rightarrow Ti(s) + 2CO(g)$ -129.36 kcal/mole;

$MnO(s) + C(s) \rightarrow Mn(s) + CO(g)$ -69.33 kcal/mole;

$SiO_2(s) + 2C(s) \rightarrow Si(s) + 2CO(g)$ -158.89 kcal/mole;

$C(s) + H_2O(g) \rightarrow CO(g) + H_2(g)$ -32.1 kcal/mole;

$2C(s) + O2(g) \rightarrow 2CO(g)$ -28kcal/mole.
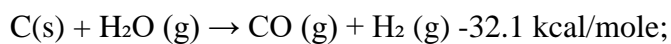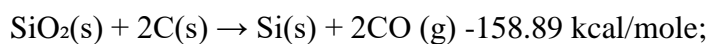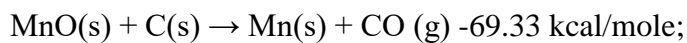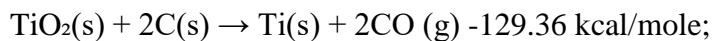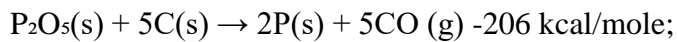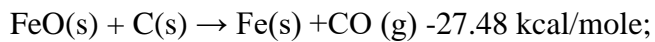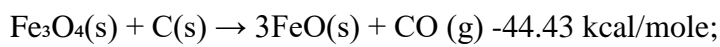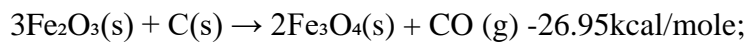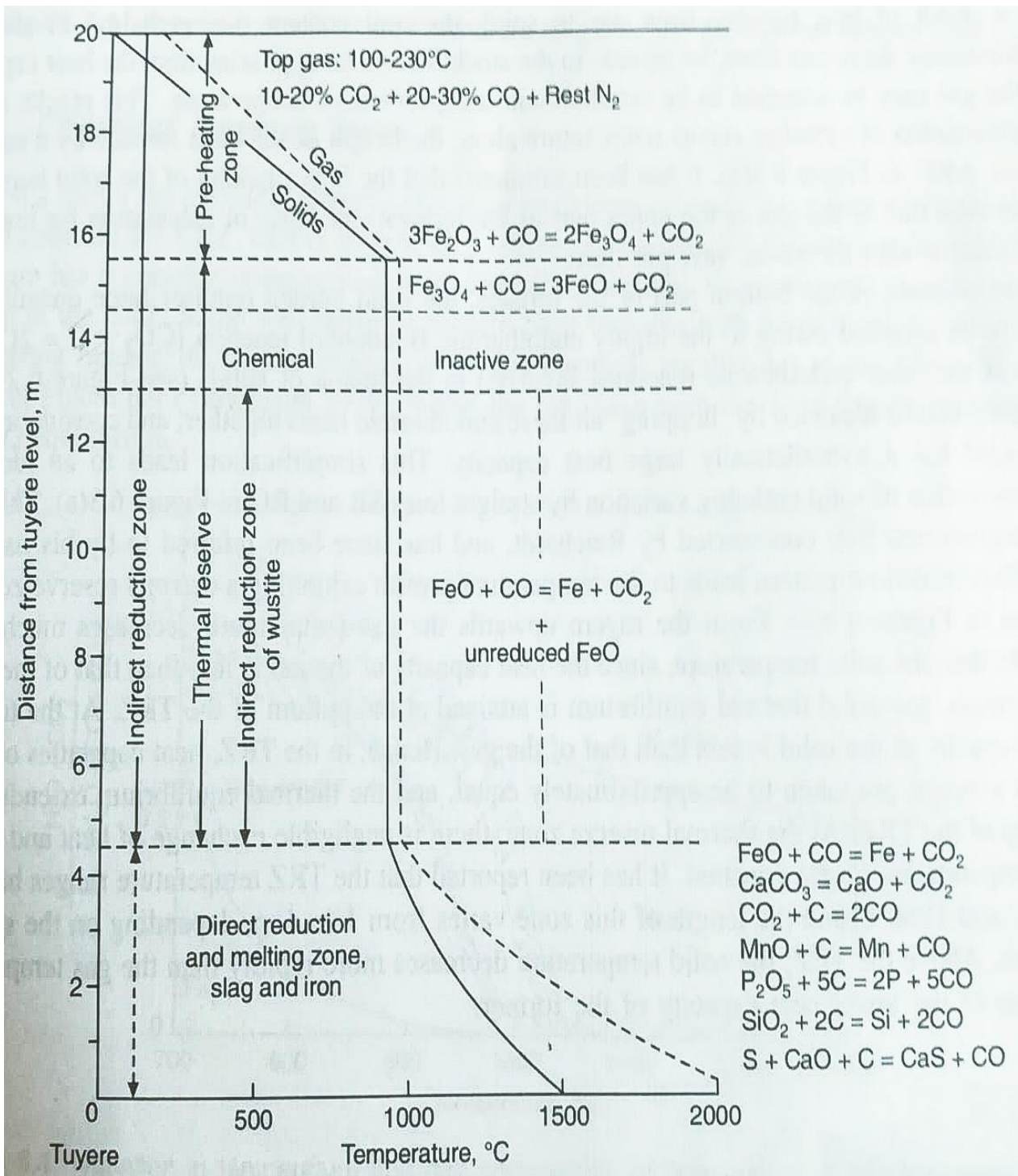
Figure 6: an ideal blast furnace temperature distribution of gas and solid along the height of blast furnace with chemical reactions. [14]

4.4 BLAST FURNACE OPERATION MODELLING

Various phenomena that take place inside a blast furnace are still not clearly explained. Therefore, blast furnace operation models have been developed to predict operational deviations and perfect the process to guarantee the stable blast furnace and to reduce fuel consumption [16–18].The blast furnace operation is usually investigated with two objectives: predicting the process indicators and under-standing the internal blast furnace phenomena. These two objectives can be studied by different methods, including industrial-scale investigations, pilot experiments (in laboratory), and mathematical models. Industrial-scale investigations contributed to the understanding of the internal aspects of the blast furnace by dissecting it. These investigations could also be conducted by sampling the material of the tuyeres; however, such techniques are difficult to use since they require stopping the blast furnace and are costly. Pilot experiments are also conducted to understand the internal phenomena of blast furnaces, but they are not able to completely represent the phenomena of a real blast furnace and are also associated with high costs. Hence, mathematical models have played an important role in predicting process indicators and investigating the internal phenomena of blast furnaces [19].

To improve the stability of the blast furnace, a thermo-chemical model was developed for process control. The purpose of the thermo-chemical model is to assist in operator decision-making regarding fuel consumption in blast furnaces. This model calculates the amount of carbon consumed at a given moment and establishes that the loaded quantity should equate to the consumption value in order to maintain a constant thermal level in the blast furnace [20]. Thermal control of the process is critical for the hot metal production in blast furnaces, since greater thermal control improves product (hot metal) quality and increases process efficiency, such as the reduction of the total energy (fuel) Consumption [21–22].

In addition to thermodynamics, the thermo-chemical model applies the concept of dividing the blast furnace into the preparation and elaboration zones, and considers the incoming gases in the preparation zone to be at thermodynamic equilibrium with iron and wustite. The thermo-chemical model is based on the degree of reduction in metallic burden within the preparation zone [23].

The gas yield (usage of CO gas) or CO/CO2ratio, cited by Spence and Pritchard [24] and also discussed by Harvey and Gheribi [25], has a large influence on the fuel consumption. It can be

observed that smaller CO/CO2ratios or better use of carbon monoxide results in the lower consumption of coke. Top gas analysis is the main sensor of the thermo-chemical model.

Hence, the control provided by the thermo-chemical model allows for corrections in the process as soon as the deviation occurs, decreasing the risk of abrupt cooling, improving the quality of hot metal and reducing fuel consumption [26].

The following are some of the advantages of applying a thermo-chemical model:- operational safety is very high; operational stability is also high with lower production costs;

According to the article ***Thermo-chemical model for blast furnace process control with the prediction of carbon consumption.***

The author simulated real data and deter-mining the calculated carbon consumption. The real and calculated carbon data were compared. In this step, the sensitivity of the model was verified in relation to the main operational parameters. With the validation from the analyses performed, the model was used by the operator for the process control with the objective of achieving greater assertiveness in the definition of the loaded carbon in relation to the consumed carbon (calculated by the thermo-chemical model).

During the validation of the thermo-chemical model, the calculated carbon consumption was analysed. **Figs. 7 and 8 [27]** show the sequence and boxplot graphs, respectively, comparing the results of the calculated and real carbon consumption by Blast Furnace A for each selected time period. The sequence graph indicates that the data sets follow the same trend and the boxplot graph shows that the means and dispersion of results are quite similar.
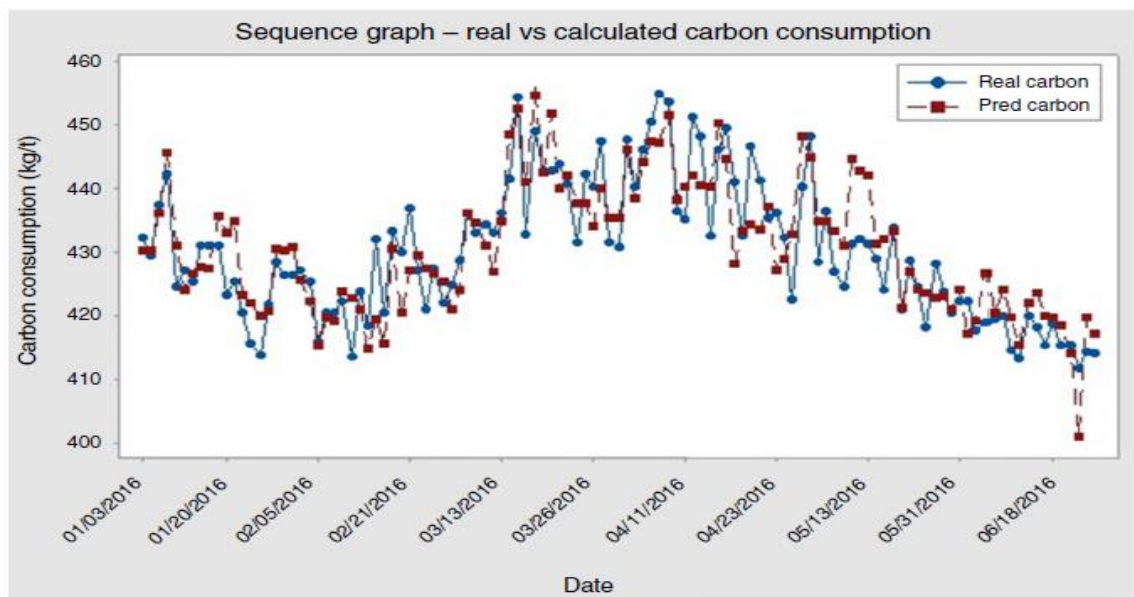
Figure 7: Sequence Graph for the Calculated Vs. Actual Carbon Consumption.
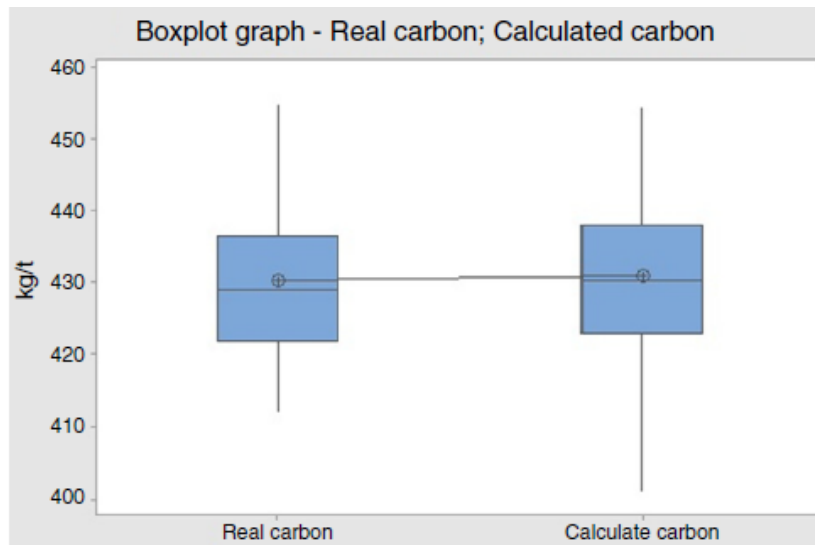
Figure 8: Boxplot Graph for the Calculated Vs. Actual Carbon Consumption.

Hence, it can be said that the thermo-chemical model is sufficient for calculation of the carbon consumption by Blast Furnace A, and can be used for process adjustment with the objective of guiding operator action toward judicious decision-making regarding fuel consumption. The specific carbon consumption (kg/t hot metal) is very much affected by the value of the omega factor. For an operation with low specific carbon consumption, the omega factor should be smaller, i.e., the process should develop a good efficiency of iron oxide reduction in the preparation zone [21].

After validating the results from the thermo-chemical model with the operational data of Blast Furnace A, the online model was analyzed. The online analysis of the thermo-chemical model is the last step for optimizing application of the model to assist the operator. The online evaluation was performed using the means of parameters every 2 h. Hence, every 2 h the operator will have a new result regarding fuel consumption to assist in the decision-making constituting blast furnace thermal control. **Fig**. **9 [27]** shows the set carbon consumption desired by the operator and the fuel consumption calculated by the thermo-chemical model.
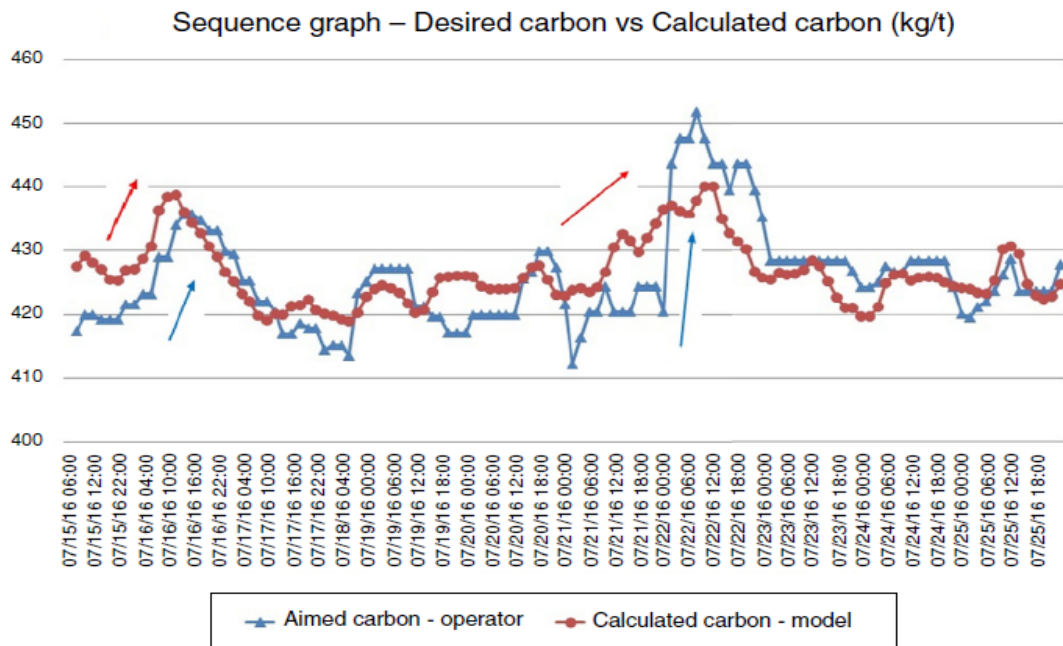
Figure 9: Sequence Graph for Desired Carbon vs. Calculated Carbon.

The model trends allow the operator to anticipate the actions and reduce variation in the thermal level of the blast furnace. Despite the preventive evaluation of the process by the thermo-chemical model, the corrective action should continue because the model does not evaluate all parameters that affect the thermal level of the blast furnace. Some examples of situations in which the use of the model is not recommended are the entrance of water in the blast furnace (leaks in the staves or tuyeres) and conditions with high process instabilities.

4.4.1 Intelligent Simulation Methodology for Blast Furnace Cooling Stave:

The intelligent simulation method for blast furnace cooling stave is to combine the traditional heat transfer simulation with the modern artificial intelligent technology. The main reason to research on the intelligent simulation method of blast furnace stave is that the heat transfer simulation does not have the functions such as self-study, self-adapting and self-organization. The math tool of intelligent simulation is quite different from that of the classical heat transfer simulation method. Heat transfer simulation theory is based on numerical heat transfer analyses, while artificial intelligence uses the math tools such as symbolic inference and first order predicate logic. The studying tool of the intelligent simulation method of blast furnace stave is a combination of the above two aspects [28, 29].

The mid to late 1980s witnessed the rapid development of artificial neural network (ANN) as an international frontier research field. In recent years, the artificial neural network theory has been applied to nearly every field of research. However, many researchers apply artificial

intelligence technology to the study object directly, rather than explore the intrinsic essence of the object. Such applications may not be suitable for all problems. As we know, the neural network theory achieves the complex functional relation through many simple relationships. It is a non-linear dynamic system essentially and does not depend on the model. The formation of ANN requires constant learning and adjustment. It is difficult for the pure ANN to analyze the internal characteristics of the object, because it only reflects the external behaviour of the object.

Artificial neural network technology is still under development. Although it has overcome the weakness of the classical heat transfer simulation which is complex and difficult to use on-site, its performance cannot fully achieve the same effect of the existing classical simulation in a short period. Therefore, the combination of the intelligent simulation and classical heat transfer simulation is an ideal way of using the strengths of both methods [30].

The heat transfer status of cooling stave is very complex when the blast furnace is working. The temperature of the blast furnace gas changes constantly, the water scale is produced on the water pipe, the thickness of furnace lining and brick is difficult to describe quantitatively, and the caking and abscission thickness of the slag is difficult to measure as well. Therefore, it is very difficult to calculate the highest temperature of the stave hot surface and to on-line monitor the state of blast furnace cooling stave in an accurate and universal manner. So the author has combined the mathematical model of heat transfer with the artificial intelligence. The mathematical model of heat transfer is used to ensure the correctness and universality of the quantitative calculation, and some inenarrable segments are replaced with the neural network as shown in **Fig**. **10 [31]**.
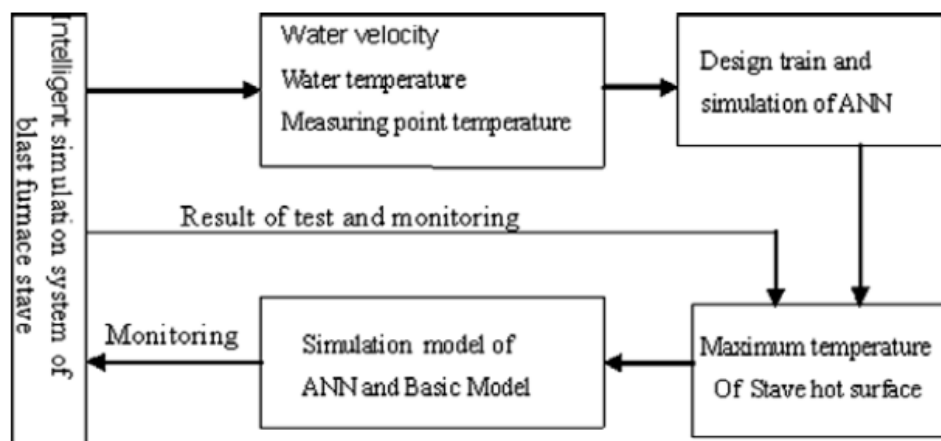
Figure 10: Schematic of Intelligent Monitoring.

This heat transfer model of blast furnace cooling stave which is combined with artificial neural network can be divided into two parts. The first part which is called basic model reflects the basic mechanism of heat transfer. Two comparisons will be made by the basic model: one is the relationship among the highest temperature of hot surface that was previously calculated by the 3-D model, the partial parameter of blast furnace cooling stave and the temperature of a measuring point. The other is the result of one-dimensional heat transfer calculation of blast furnace cooling stave. The second part is artificial neural network which is used to adaptively fill the gap between basic model and experimental data. The basic model will guarantee the universality, and an appropriate simplified regression relational expression can be used to make the calculation easier. The artificial neural network can improve the degree of agreement between the model and practical physical process by studying the experimental data [31].

4.4.2 Two-Dimensional Total Blast Furnace Model

The two-dimensional total blast furnace model has been developed to obtain two-dimensional information about the cohesive zone profile and gas/temperature distributions in the blast furnace. Aside from this total model, various other models for estimating cohesive zones have been developed [32, 33]. The two-dimensional total blast furnace model, which is based on the BRIGHT (Blast furnace Realization for Instruction Guide by Hybrid Theory) model developed by Sugiyama et al. [34, 35], integrates various models, including the burden distribution model [36]. In order to improve the accuracy of the conventional reduction model (greater accuracy was demanded by the diversification in raw materials and fuels used and blast furnace operation conditions), the total model has incorporated a multistage reducing reaction mode [37], and a modified high-temperature property model, etc. As the new BRIGHT model (N-BRIGHT model), it is used to analyze various phenomena that occur within the blast furnace. This is not to say that the N-BRIGHT model is always used off-line [38, 39]. When built into a process computer, the model permits the display of calculation results on-line in the operation room. Thus, it can be also effectively used to determine blast furnace operation policy

a. Bright Model

When it comes to changing certain operational conditions for an actual blast furnace, it is necessary to predict how the cohesive zone profile and phenomena in the furnace respond to such change. However, in the case of a blast furnace, which is a large and complicated system, using an actual blast furnace to determine the effect of a specific factor experimentally takes much time and money. Besides, since furnace phenomena is reflected to complicated interactions of gas/solid flows, as well as mass and heat transfers, in order to get the detail knowledge of the in-furnace phenomena from data obtained by sensors and basic experiments, etc. and apply such knowledge to actual blast furnace operations, it is necessary to employ a theoretical mathematical model that combines the above factors organically. Therefore, a two-dimensional mathematical model was developed that permits deduction of the estimated gas/solid flows, temperatures and reactions in the blast furnace from particular operational conditions. As shown in **Fig. 11 [40]**, this model is made up of several sub-models—namely the burden distribution, gas flow, solid flow, chemical reaction, liquid flow and heat transfer sub-models. The total blast furnace model which integrates all these sub-models is used to calculate the cohesive zone profile (**Fig. 12**) [40].

The model has these characteristics:

(a) It is applicable to all blast furnaces,

(b) It is a theoretical model allowing for simultaneous analysis of solid/gaseous flows, reactions and heat transfer,

(c) It places the major emphasis on two-dimensional burden distribution in furnace radial direction,

(d) It permits theoretical estimation of cohesive zone profiles,

(e) It does not always require input of data obtained by probes, though input of probe data is possible,

(f) It allows for theoretical experiments and drastic changes of operational conditions that cannot be tested in an actual blast furnace, and it readily permits reflection of the results of basic experiments.
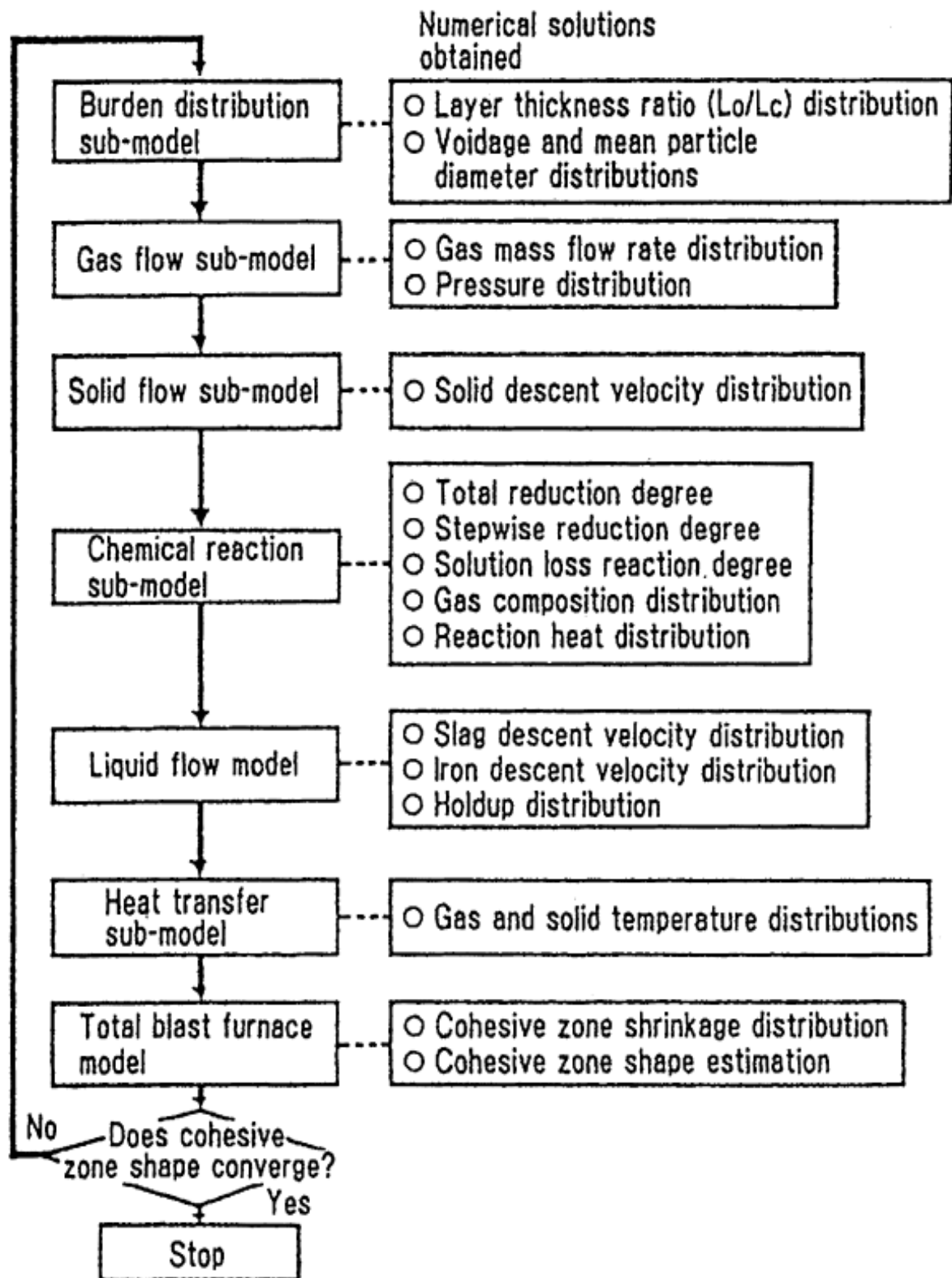
Figure 11: Basic Concept of Bright Model.
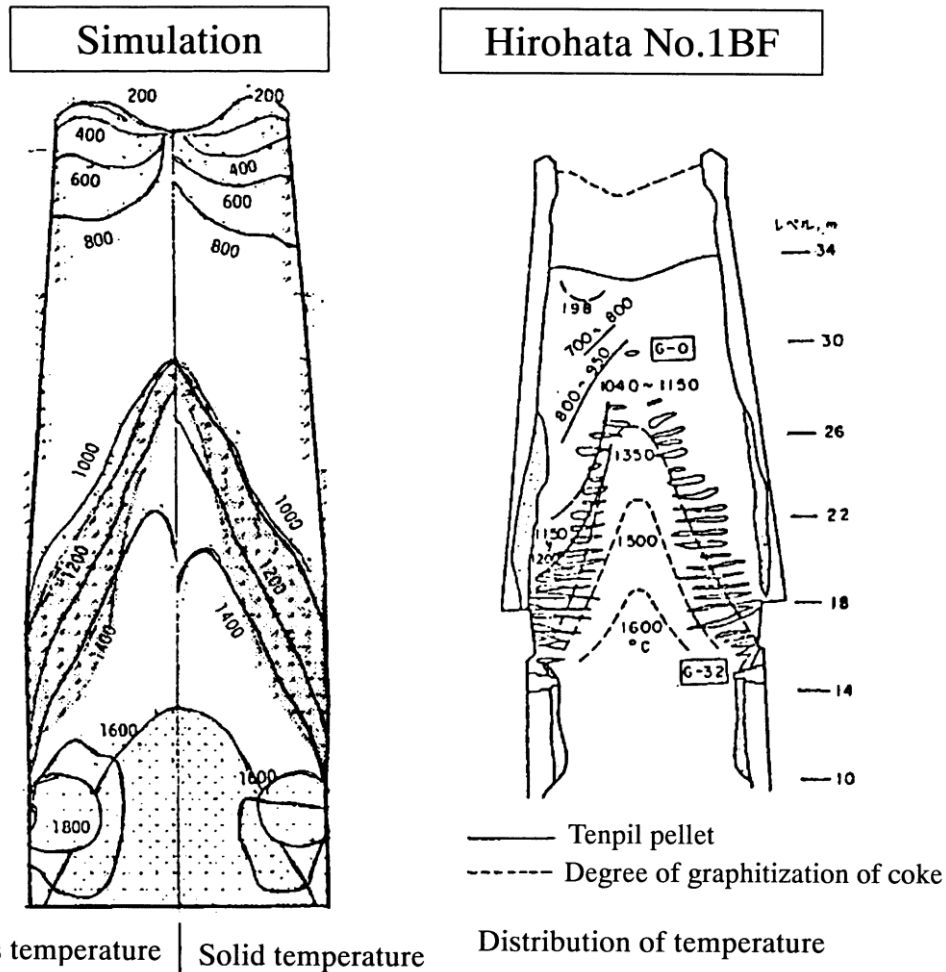
Figure 12: Comparison between Model Calculation Result and Dismantling
Blast Furnace Finding.

b) N-Bright

This model incorporate multistage reaction zone model. The N-BRIGHT model incorporates a multistage reaction zone model—a highly sophisticated version of the reduction model that is the most important constituent of the total blast furnace model. As a sinter reduction model, the one-interface model that allows for easy analysis or the multi-interface unreacted core shrinking model has been used. The BRIGHT model incorporated a three-interface unreacted core shrinking model. However, since it could not fully reflect the difference in sinter properties on the in-furnace phenomena, the multistage reaction zone model developed by Naito et al. has been incorporated in the N-BRIGHT model. The multistage reaction zone model assumes that hematite, magnetite, wustite and iron ore continue reacting hand in hand, rather than independently of one another with a definite boundary between them as assumed by the three-interface unreacted core shrinking model. With this model, it has become possible

to obtain calculation results that agree well with measurement results obtained in an actual blast furnace and experimental results on reductions obtained in a furnace, etc.[41-43] (**Fig.13**).
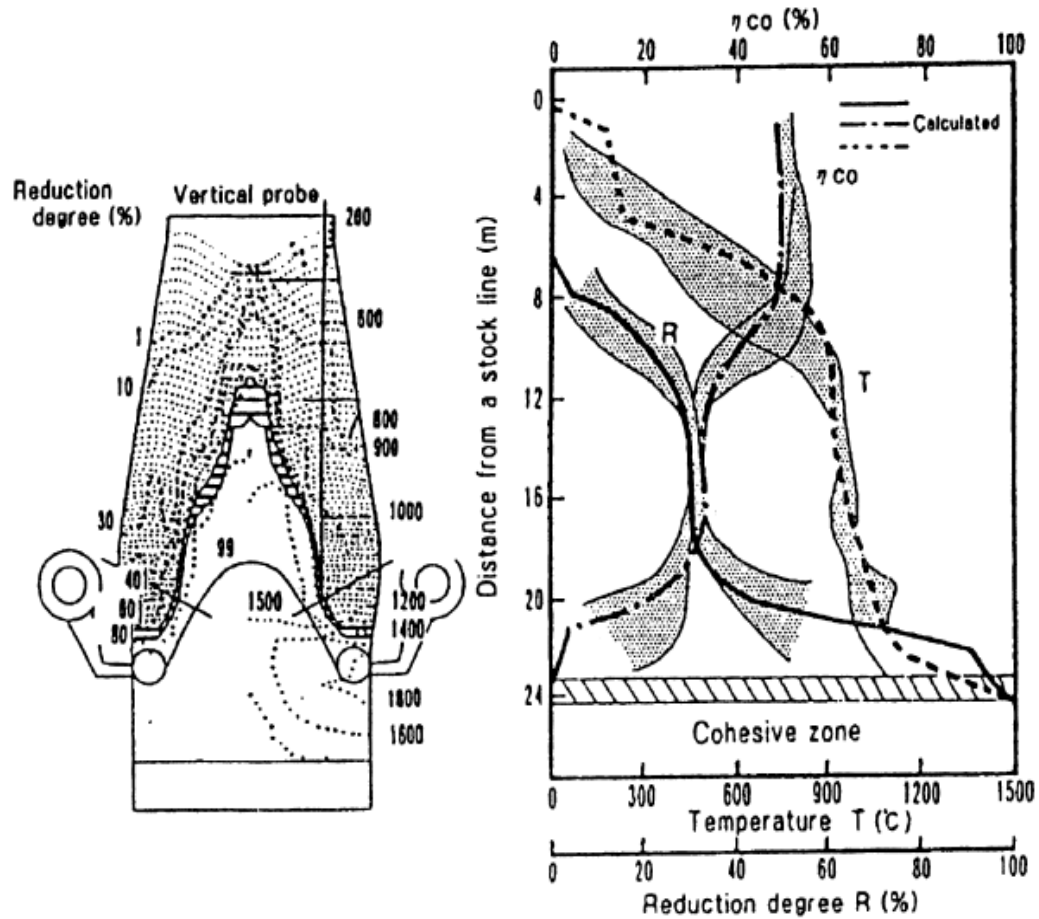
Figure 13: Comparison between Analyzed Values and Values Measured with Probes in Blast Furnace [43].

4.4.3 Blast Furnace Slag Viscosity Model:

The viscosity of blast furnace slag is a very important element in blast furnace operation. The softening/shrinkage and permeability of the cohesive zone and permeability in the raceway and deadman depend considerably on the slag viscosity. On the other hand, the slag viscosity itself is influenced by the change in composition and the temperature at the time of slag formation. It is already known that the slag viscosity depends on its composition and temperature and that the presence of coke powder or the crystallization of solid phase causes it to decrease markedly. Nevertheless, there are few models that take those influences into consideration. Therefore, an

attempt was made to develop a slag viscosity estimation model that takes into account the increase in slag viscosity due to precipitation of the solid phase. To estimate the viscosity of blast furnace slag, the experimental recurrence equation developed by Sugiyama has been applied, Nakagawa et al. [44]. As long as the slag temperature is somewhat higher than its melting point, the calculation result obtained by the equation agrees well with the experimental result. However, when the slag temperature drops below its melting point and the solid phase begins to precipitate, the difference between the calculated and measured results widens significantly. If the solid phase ratio can be obtained, it should become possible to accurately calculate the slag viscosity even when the slag temperature drops below its melting point. Therefore, it was calculated that the solid phase ratio of slag using SOLGASMIX—a model for theoretical calculation of a phase diagram [45, 46] and estimated that the viscosity of suspended slag at the formation of solid phase using the relational expression developed by Mori et al. [47]. On the basis of the results obtained, a model was developed for calculating the viscosity of slag at the time of precipitation of the solid phase [48, 49] (**Fig. 14**).
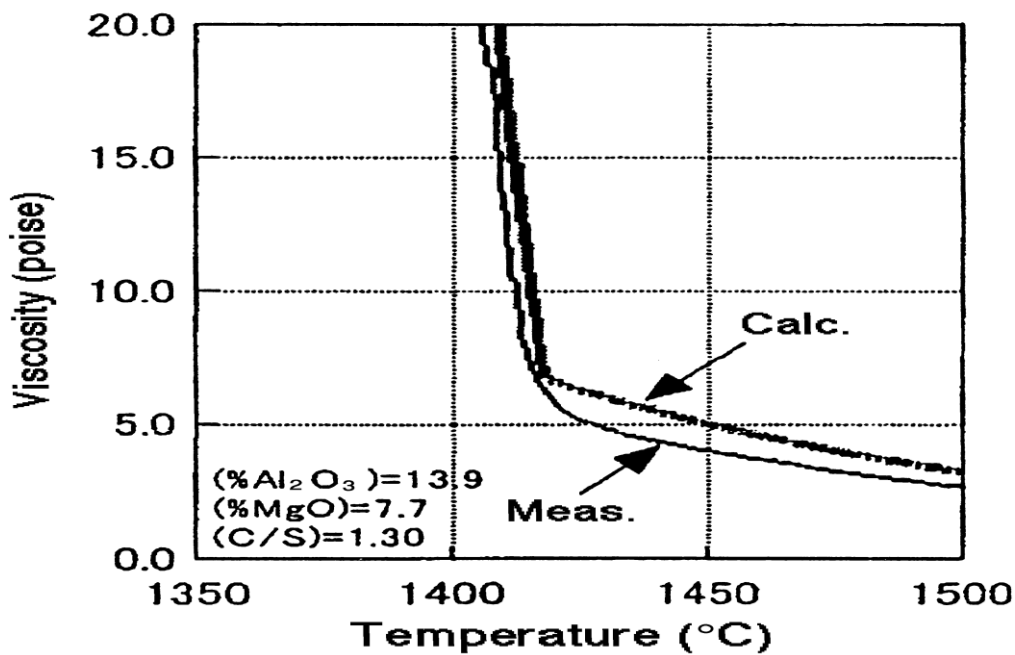
Figure 14: Comparison of Slag Viscosity between Measured and Calculated One [49].

4.4.4 RIST Modelling:

Burden is charged in blast furnace and RIST modelling is used to reduce CO_2 emission and increase the productivity of iron [50].

The simulation illustrates the effects of coke substitution by means of hydrogen injection. A necessary assumption for this scenario is constant thermodynamic efficiency which is expressed by a constant horizontal approach of the Rist operation line to the wustite equilibrium. Depending on operating conditions, the blast furnace gas temperature can range from 100 °C to 230 °C (Lüngen and Yagi, 2012) In this particular case, it is specified with 150 °C. Based on this, the injection of H2 is increased from 0 to 200 m3 (STP) / t (HM).

The overall simulation results of this variation are given in Figures 15-17 [50, 51]. It can be observed in Figure 5 that the specific amount of coke required for the process drops from 455 to 401 kg/t hot metal. Moreover, the specific amount of hot blast is also decreased from 952 to 889 m3 (STP) / t hot metal. This effect is explained by a shift in the reduction mechanisms in the blast furnace which is caused by hydrogen injection. Since solid carbon is exchanged by gaseous H_2, highly endothermic direct reduction with solid carbon is partially replaced by less endothermic indirect reduction with H_2. Therefore, the input of energy-rich hot blast has to be reduced in order to ensure constant blast furnace gas temperature. Figure 17 [51] illustrates calculated process outputs. A minor change can be observed for the specific slag production rate with a decrease from 222 to 217 kg/t hot metal. In addition to this, the CO_2 equivalent emissions decrease from 1,428 to 1,259 kg/t hot metal which is equal to a reduction of approximately 12 %. Both effects are linked to the decreasing coke mass flow which supplies both carbon and slag forming components to the blast furnace process. A drop from 2,300 to 2,022°C is visible for the RAFT which is explained by the decreasing supply of hot blast and the increasing hydrogen injection rate [50].
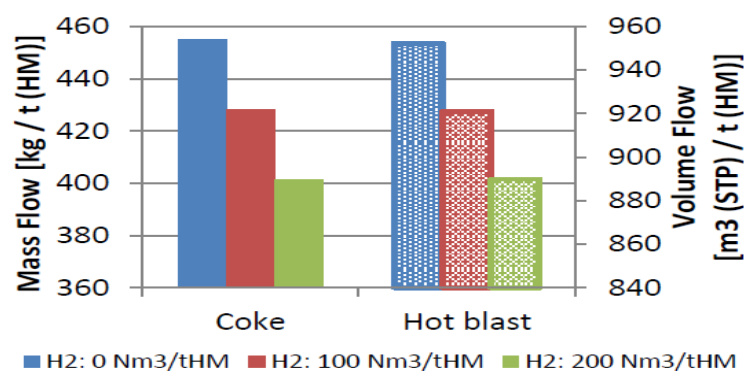
Figure15: **Calculated Input Variations Output**
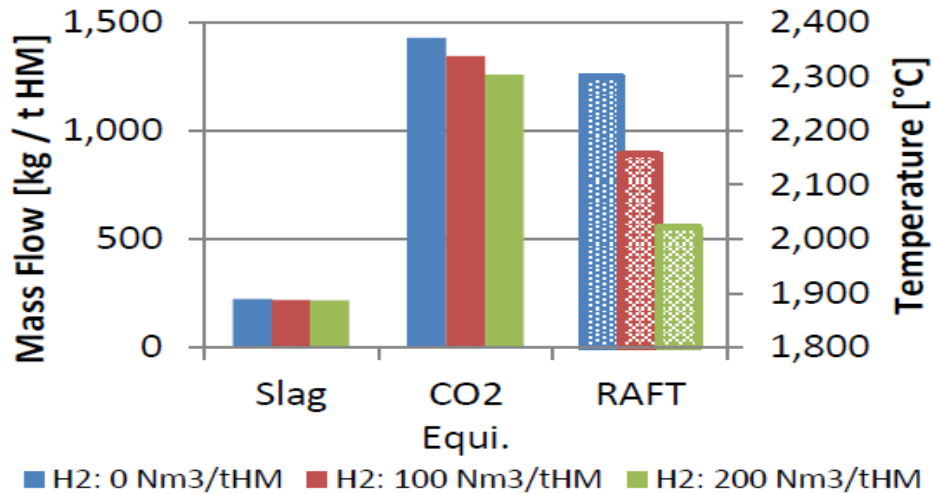
Figure16: Slag and CO_2 Equivalent RAFT
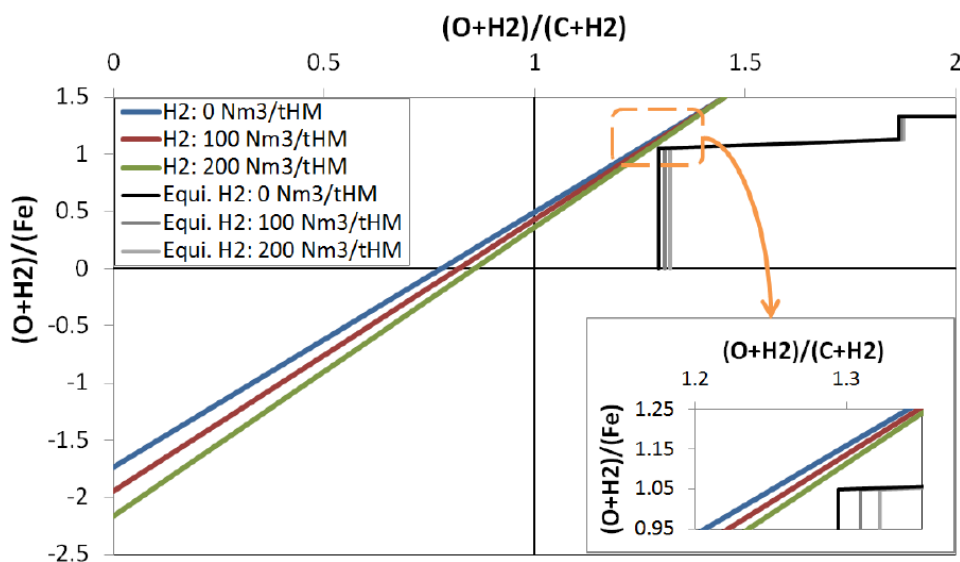
Figure17: RIST Operating Diagram

Figure 17 evaluates the corresponding Rist operating diagram of this simulation. The gradient of the operation line is decreasing with increasing hydrogen injection. This can be concluded that a decreasing reducing agent demand of the process which is in line with the observed decrease of CO_2 equivalent emissions. Furthermore, a drop of direct reduction percentage can be observed which corresponds to the shift in the reduction mechanisms from carbon to hydrogen [50].

4.5 BF BURDEN SCENARIO IN INDIA

Plant Name	Burden proportion			Fuel Rate				Flux		Productivity	Slag Rate
	Sinter	Pellet	Lump	PCI	Nut coke	Metallurgical coke	Total Fuel	Lime stone	Dolomite		
SAIL – BOKKARO	70	0	30	505 (Individual data not available)			505	Data not available		1.6	380
SAIL- BHILAI	70	0	30	42.6	25.86	507.142	575.602	Data not available		1.65	411.57
SAIL – ROURKELA	75	0	25	130	30	417	577	Data not available		2.11	342
SAIL – DURGAPUR	70	0	30	47.33	18	492.67	558	Data not available		1.697	325.67
ISP	75	0	25	16	23	497.5	536.5	Data not available		1.08	345
TATA	45	40	15	179.25	378.75		558	Data not available			291
RINL – VIZAG	75	0	25	13.2	21.75	513	547.95	Data not available		1.77	305.5
JSW - Bellary	70	10	20	117.25	39.75	395.5	552.5	Data not available		2.59	407.5
JSW- Dolvi	75	25	0	160	35	315	510	Data not available		2.77	340-360
JSPL				120.15	20.02	414	554.17			3.42	402.08
BHUSHAN	80	20	0	121.5	36.5	367	525			2.5	322

Table 2: BF Burden Distribution Scenario in Indian Steel Companies,

[60th OCM Presentation].

In India all the public steel player (SAIL & RINL) are using mainly sinter and lump as burden in iron making units , whereas private players are utilizing iron ore pellet in the burden as shown in table 2. TATA and ESSAR steel is using around 40% or more than 40% pellet in the blast furnace and they are trying to enhance the percentage without affecting the blast furnace smooth operation.

4.6 HEAT BALANCE OF BLAST FURNACE

A heat balance analysis for a continuous production blast furnace was presented by [52]. The authors gave a simplified model to calculate heat distributions for various components of the furnace. However, the authors did not provide any insight to the reactions occurring in the furnace. Also, no sub-divisions to the calculations were shown. A layout was shown by the author which is very useful for the further calculation.
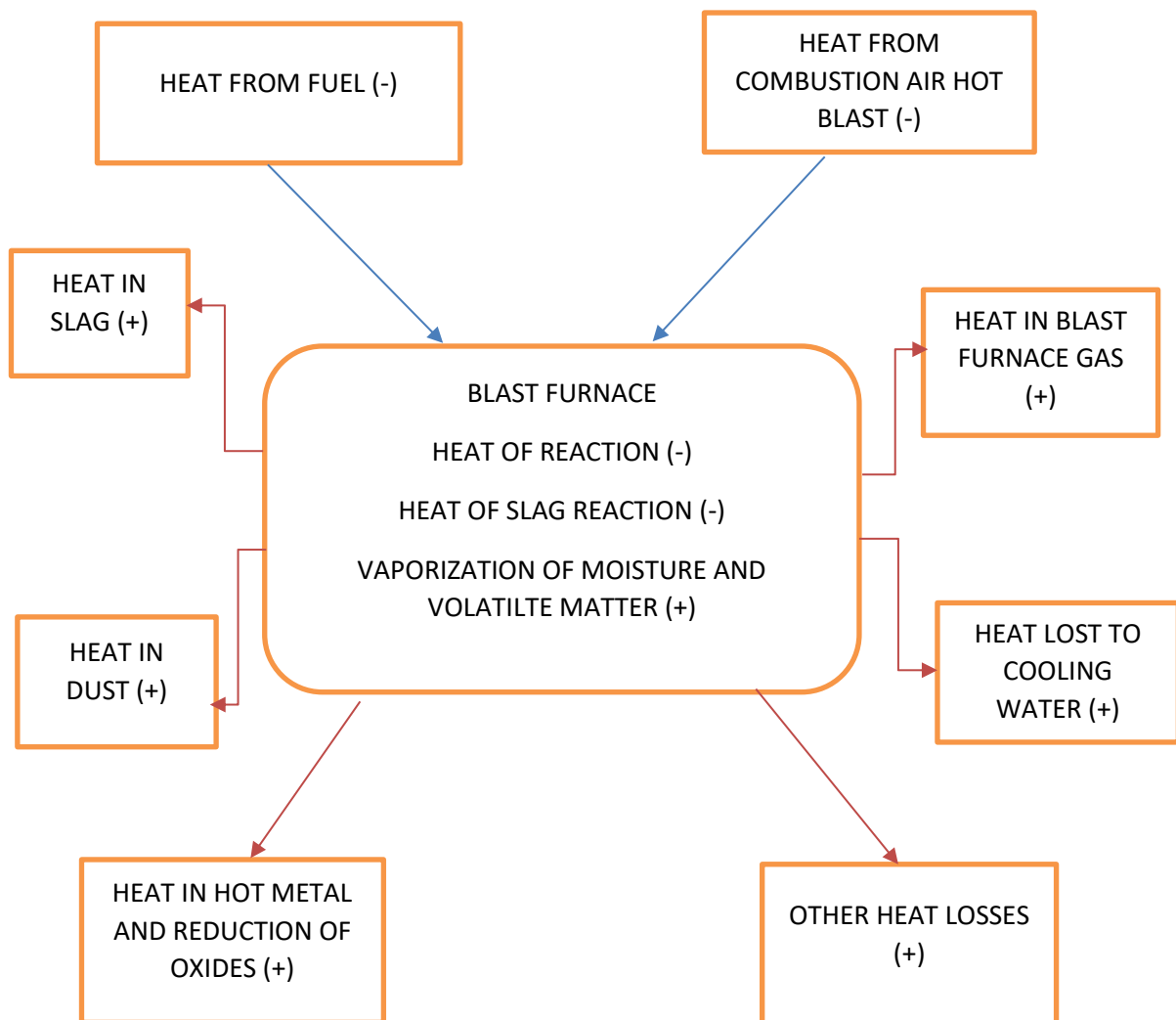
So for heat calculation enthalpy of all compounds and elements are needed. Various standard textbooks, and papers have been referred to get an in-depth insight into the factors which should be taken into consideration and to get enthalpy of all compounds and element.

The enthalpy data of elements and compounds at high temperatures has been taken from [53]. These are as follow;

Latent heat of H_2O = 540 kcal/kg;

Sensible heat of H_2O (298K to 1473K) =11.14 kcal/moles;

Sensible heat of O_2 (298K to 1473K) =9.37 kcal/moles;

Sensible heat of N_2 (298K to 1473K) =8.93 kcal/moles;

Sensible heat of hot metal elements at 1500°C has been shown in table 3;

Element	Sensible heat (kcal/moles)
Silicon	21
Manganese	17.89
Phosphorus	46.26
Titanium	11.38
Carbon	8.84
Iron	17.171

Table 3: Standard Value of Sensible Heat of Elements.

Sensible heat of blast furnace gas at 150°C has been shown in table 4;

Gas	Sensible heat(kcal/moles)
CO	1.07
CO_2	1.51
N_2	1.11
O_2	1.09
H_2	1.06

Table 4: Standard Value of Sensible Heat of Gases

Calorific value of blast furnace gas is 870 kcal/kg;

Heat capacity of volatile matter (phenol) is 122 kcal/kg;

The dust formation temperature is taken as 1150^0C as the reactants with the dust, undergo reaction only after reaching the desired temperature. Also, dust is produced after a reaction, hence the calculation for sensible heat of dust will take into consideration the temperature in which a reaction takes place. Standard value of enthalpy of compounds at 1150^oC has been shown in table 5.

Compound	Sensible heat(kcal/moles)
C	4.99
Fe_2O_3	37.65
SiO_2	17.64
Al_2O_3	30.8
CaO	13.43
MgO	12.57

Table 5: Standard Value of Sensible Heat of BF Dust.

The heat of solution data has been taken from [54]. And it has been shown in table 6.

Element	Heat of solution (kcal/mole)
Silicon	-28.5
Manganese	1.2
Phosphorus	-29.2
Titanium	-13.35
carbon	7.645

Table 6: Value of Heat of Solution.

Heat of slag formation is very important. The standard heat value is taken from [54].

Heat produced during slag formation = -140 kcal/kg;

Specific heat of slag (at 1500°C temperature) = .2741 kcal/kg^{0}C;

Specific heat of slag (at ambient temperature) = 0.185 kcal/kg^{0}C.

CHAPTER 5:
ASSUMPTION MADE FOR CALCULATION

5. ASSUMPTION MADE FOR CALCULATION:

Raw material composition and output hot metal composition data which is taken from industry, is assumed to be fixed for all calculation. And it is given below;

SINTER

Fe_2O_3	FeO	SiO_2	CaO	MgO	P_2O_5	MnO	TiO_2	Al_2O_3
71.04	9.23	5.47	10.1	1.59	0.063	0.18	0.14	2.1

PELLET

Fe_2O_3	SiO_2	Al_2O_3	CaO	MgO	FeO
93.1	2.51	2.64	0.98	0.12	0.64

LUMP ORE

Fe_2O_3	SiO_2	P_2O_5	MnO	TiO_2	Al_2O_3
90.23	4.6	0.09		0.14	4.13

COKE

Fixed Carbon	85.51
Volatile Matter	2.16
Ash	12.34

COKE ASH

SiO_2	Fe_2O_3	CaO	MgO	MnO	TiO_2	P_2O_5	Al_2O_3
57.2	6.67	1.81	0.66		1.79	1.03	28.1

COAL

Ash	10.35
Moisture	1.7
V.M	25.5
F.C	62.45

COAL ASH

SiO_2	Fe_2O_3	CaO	MgO	MnO	TiO_2	P_2O_5	Al_2O_3
45.1	7.17	6.76	0.69	0.03	1.62	1.83	34.2

LIME STONE (LOI free)

CaO	MgO	SiO2	Al2O3
83.5	1.51	2.21	1.41

QUARTIZE (LOI free)

$CaCO_3$	$MgCO_3$	SiO_2	Al_2O_3
		96.9	3

DUST ANALYSIS

Fe_2O_3	SiO_2	Al_2O_3	CaO	MgO	C
55.87	5.72	3.32	5.81	1.07	28.21

HOT METAL

Fe	Si	Mn	P	Ti	C
94.853	0.62	0.09	0.081	0.056	4.3

For cost comparison, unit price of different material is taken from another industry as well as from online website steelmint.com and it is also assumed to be fixed for entire calculation.

Unit in Ton	Rs./ton
Unit cost of sinter(in house production)	4651
Unit cost of pellet(in house production)	5600
Unit cost of lump	3310
Unit cost of coke	25801
Unit cost of coal	11853
Unit cost of quartz	2100
Unit cost of lime stone	2350

Table 7: Unit Cost of Input Materials

Unit cost of slag	500	Rs./ton
Unit cost of off gas	1250	Rs./Gcal
Unit cost of pig iron	27500	Rs/ton

Table 8: Unit Cost of Output Material

CHAPTER 6:

RESULTS AND

DISCUSSION

6. <u>RESULTS AND DISCUSSION:</u>

In this chapter, all the theoretical work and calculations have been shown for one set of data (50% sinter, 30% pellet and 20% lump ore) and also due to variation of burden (*20% lump is fixed and sinter and pellet % have been changed from 10% to 80% with the gap of 10, also burden % of Tata steel, JSW Bellary, SAIL Durgapur and RINL Vizag have been taken from section 4.5*) charged, what changes appears (*calculation has been done for each variation of burden percentage in Microsoft excel*), have been shown. All the weight percentage data is taken from chapter 5.

6.1 CALCULATION OF HOT METAL:

As discussed in methodology, hot metal weight was fixed to 1 ton or 1000 kg. Calculation of weight of hot metal compositions and moles are shown in table 9. And also calculated weight of hot metal composition is same for all burden percentage variation.

HOT METAL COMP		wt.(kg)	moles
%		(%*1000)/100	Wt.*1000/atomic weight
Fe	94.853	948.53	16938.04
Si	0.62	6.2	221.42
Mn	0.09	0.9	16.36
P	0.081	0.81	26.12
Ti	0.056	0.56	11.66
C	4.3	43	3583.33

Table 9: Hot Metal Composition's Weight and Moles.

Total weight of Fe in 1 THM = 948.53 kg;

Considering 1% loss, wt of Fe = (948.53*.01+948.53) = 958.0153 kg;

6.2 CALCULATION OF WT. OF BURDEN CHARGED IN BLAST FURNACE:

Fe from burden in hot metal in %		Fe wt in kg/THM
Sinter	50	479.007
Pellet	30	287.404
Lump	20	191.603

Table 10: Wt. of Fe from Burden

Fe in Fe_2O_3 = (116/160) = 0.7;

Fe in FeO = (56/72) = 0.77;

Total weight of ore in burden = (total Fe from ore in hot metal/ % Fe in ore)

Fe in pellet = (93.1*0.7) + (.64*0.77) = 65.6628 %;

Wt. of pellet = 437.697 kg;

Same as weight of sinter = 842.802 kg;

Weight of iron ore lump = 303.356 kg;

Weight of composition of pellet has been calculated and shown in table 11.

	Fe_2O_3	SiO_2	Al_2O_3	CaO	MgO	FeO
%	93.1	2.51	2.64	0.98	0.12	0.64
WT. IN KG/THM	407.496	10.986	11.555	4.289	0.525	2.801
MOLES	2546.854	183.103	113.286	76.597	13.131	233.432

Table 11: Calculation of Wt. of Composition of Pellet

Weight of composition of sinter has been calculated and shown in table 12.

	Fe$_2$O$_3$	FeO	SiO$_2$	CaO	MgO	P$_2$O$_5$	MnO	TiO$_2$	AL$_2$O$_3$
%	71.04	9.23	5.47	10.1	1.59	0.063	0.18	0.14	2.1
WT. IN KG/THM	598.72	77.790	46.101	85.12	13.421	0.530	1.517	1.179	17.698
MOLES	3742.04	1080.42	768.35	1520.05	335.014	3.739	21.366	14.749	173.518

Table 12: Calculation of Wt. and Moles of Composition of Sinter

Similarly weight. of composition of iron ore lump has been calculated and shown in table 13.

	Fe$_2$O$_3$	SiO$_2$	P$_2$O$_5$	MnO	TiO$_2$	Al$_2$O$_3$
%	90.23	4.6	0.09	0	0.14	4.13
WT. IN KG/THM	273.71	13.954	0.273	0	0.424	12.528
AT. WT. IN GM	160	60	142	71	80	102
MOLES	1710.742	232.573	1.922	0	5.308	122.821

Table 13: Calculation Wt. and Moles of Composition of Lump

6.3 CALCULATION OF WT. OF COKE AND COAL REQUIRED:

6.3.1. Calculation of Fixed Carbon Required for Operation:

Indirect reduction   65%

Direct reduction   30%

Reduction by H2   5%

Total Fe$_2$O$_3$ = 1279.942 kg;

$$3Fe_2O_3 + \quad CO \quad \rightarrow \quad 2Fe_3O_4 + \quad CO_2$$
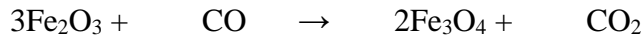
Fe_2O_3 reduced by CO = 831.962   (indirect reduction)

To reduce 160 kg Fe_2O_3 we need = 84 kg CO

To reduce Fe_2O_3 we need = 436.783 kg CO

$$3Fe_2O_3 + \quad C \quad \rightarrow \quad 2Fe_3O_4 + \quad CO$$
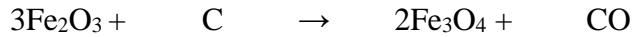
Fe_2O_3 reduced by C = 383.984 kg   (direct reduction)

To reduce 160 kg Fe_2O_3 we need = 36 kg C

To reduce Fe_2O_3 we need = 86.394 kg C;

$$3Fe_2O3 + \quad H_2 \quad \rightarrow \quad 2Fe_3O_4 + \quad H_2O$$
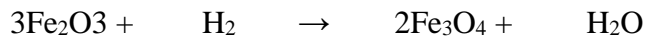
Fe_2O_3 reduced by H_2 = 63.997   (indirect reduction)

For 160 kg Fe_2O_3 we need =6 kg H_2;

For 64.15118kg Fe_2O_3 we need   = 2.399 kg H_2;

$$C \quad +H_2O \quad \rightarrow \quad CO \quad + \quad H_2$$

To convert H_2O into H_2, C required = 14.399 kg;

Total FeO = 80.591 kg;

$$FeO \quad + \quad CO \quad \rightarrow \quad Fe \quad + \quad CO_2$$
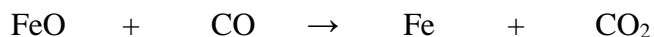
FeO reduced by CO = 52.384;

For 72 kg FeO we need = 28 kg CO;

For 54.45347 kg FeO we need = 20.371 kg CO;

$$FeO \quad + \quad C \quad \rightarrow \quad Fe \quad + \quad CO$$
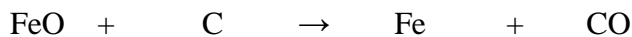
FeO reduced by C   = 15.715;

For 72 kg FeO we need = 12 kg C;

For 13.61337 kg FeO we need = 2.619 kg C;

$$FeO \quad + \quad H2 \quad \rightarrow \quad Fe \quad + \quad H2O$$
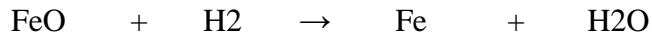
FeO reduced by H_2 = 4.029 kg;

For 72 kg FeO we need = 2 kg H_2

For 3.889 kg FeO we need = 0.112 kg H_2;

To convert H_2O into H_2, C required = 0.671 kg;

Si   6.2 kg

$$SiO_2 \quad + \quad 2C \quad \rightarrow \quad Si \quad + \quad 2CO$$
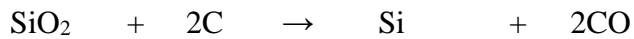
To produce 28 kg Si from SiO_2 we need = 24 kg C;

To produce 6.2 kg Si from SiO_2 we need = 5.314   kg C;

Mn   0.9   kg

$$MnO \quad + \quad C \quad \rightarrow \quad Mn \quad + \quad CO$$
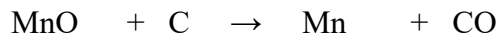
To produce 55 kg Mn from MnO we need = 12 kg C;

To produce .9 kg Mn from MnO we need = 0.196   kg C;

P   0.81   kg

$$P_2O_5 \quad + \quad 5C \quad \rightarrow \quad 2P \quad + \quad 5CO$$
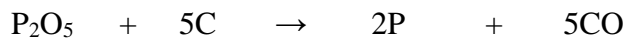
To produce 62 kg P from P_2O_5 we need = 60 kg C;

To produce .81 kg P from P_2O_5 we need = 0.78   kg C;

Ti   0.56   kg

$$TiO_2 \quad + \quad 2C \quad \rightarrow \quad Ti \quad + \quad 2CO$$
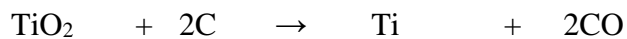
To produce 48 kg Ti from TiO_2 we need = 24 kg C;

To produce .56 kg Ti from TiO_2 we need = 0.28 kg C;

Total CO = 457.152   kg;

Carbon in CO = 42.85 %

Total Carbon in CO = 195.922 kg;

So total theoretical carbon required for reduction = sum of all carbon calculated from stoichiometry and carbon in hot metal =349.581kg;

But in blast furnace, there are also other endothermic reactions occur so carbon requirement is more than theoretically calculated. And here for calculation it is taken to 22% of theoretical requirement (after analysing the carbon requirement of different industries).

So total carbon required = 426.494 kg;

6.3.2. Calculation of Wt. of Coke and Coal

Now 77% of carbon is contributed by coke charge and 23% of carbon is contributed by PCI charge (as it is fixed that PCI charge could vary between approx. 150 kg to 160 kg).

So weight of coke charged = (total carbon*77) / carbon % in coke;

Weight of coal or PCI charged = (total carbon*23) / carbon % in coal;

Weight of coke = 384.0468 kg;   weight of coal = 157.0745 kg;

Calculated weight of coke and coal compositions have been shown in table 14 and 15.

Coke Chemical composition :-		
	%	wt. in kg
Fixed Carbon	85.51	328.3984
Volatile Matter	2.16	8.29541

Coke ash:

	SiO_2	Fe_2O_3	CaO	MgO	MnO	TiO_2	P_2O_5	Al_2O_3
%	57.2	6.67	1.81	0.66	0	1.79	1.03	28.1
WT. IN KG/THM	27.107	3.161	0.857	0.312	0	0.848	0.488	13.311
AT. WT. IN GM	60	160	56	40	71	80	140	102
MOLES	451.797	19.756	15.317	7.819	0	10.603	3.486	130.558

Table 14: Calculation of Wt. and Moles of Coke and Coke Ash Compositions.

coal chemical composition:-		
	%	wt. in kg
Ash	10.35	16.257
Moisture	1.7	2.670
V.M	25.5	40.054
F.C	62.45	98.093

Coal ash:

	SiO_2	Fe_2O_3	CaO	MgO	MnO	TiO_2	P_2O_5	Al_2O_3
%	45.1	7.17	6.76	0.69	0.03	1.62	1.83	34.2
WT. IN KG/THM	7.332	1.165	1.098	0.112	0.005	0.263	0.297	5.559
AT. WT. IN GM	60	160	56	40	71	80	140	102
MOLES	122.2	7.282	19.624	2.804	0.068	3.292	2.125	54.509

Table 15: Calculation of Wt. and Moles of Coal and Coal Ash Composition

6.4 CALCULATION OF SLAG RATE AND WT. OF FLUX ADDED:

Calculation of weight of Burden charged already has been shown in the above section. Now weight of BF dust as well as weight. of BF dust composition has been calculated. After that slag rate has been calculated.

Loss of Fe in BF dust = 0.5% of total Fe

Total weight (wt.) of BF dust = (0.5 % of total Fe / Fe % in BF dust)

Fe percentage has been taken from the industry and practically it is not fixed but for calculation and observation it has been taken as fixed quantity.

Total wt. of dust = 12.2480158 kg;

	Fe_2O_3	SiO_2	Al_2O_3	CaO	MgO	C
%	55.87	5.72	3.32	5.81	1.07	28.21
WT.	6.842	0.701	0.406	0.711	0.131	3.455
MOLES	42.768	11.676	3.986	12.704	3.276	287.931

Table 16: Calculation of Wt. and Moles of Dust Compositions

SiO2 converted into Si = 13.285   kg

MnO converted into Mn = 1.161   kg

TiO2 converted into Ti = 0.933   kg

Now all the impurities present in burden, coal ash and coke ash in the form of different oxides are converted into slag and fraction of that oxides are gone into BF dust and taken part in the reactions. Weight of oxides, converted into slag are calculated below;

SiO_2 converted into slag   = 96.626 kg;

MnO converted into slag   = 0.360   kg;

TiO_2 converted into slag   = 1.782   kg;

CaO converted into slag = 101.457 kg;

MgO converted into slag   = 14.415 kg;

Al_2O_3 converted into slag = 60.586 kg;

FeO converted into slag = 2.463 kg;

So total weight of oxides = sum of all oxides = 277.691 kg;

Now to maintain the taping basicity 1.05, fluxes (lime stone and quartz) are added during the blast furnace operation. And the minimum weight of fluxes have been calculated by using solver technique in Microsoft excel.

So weight of lime stone (LOI free) added = 12.932 kg;

Weight of quartz (LOI free) added = 5 kg;

6.5 CALCULATION OF BLAST AIR REQUIRED AND OFF GAS VOLUME:

Basically blast air used as oxygen supply. Firstly all the metalloid oxygen has been calculated then calculation of extra oxygen required from blast air has been done. To calculate their volume, several assumption has been made which has been already discussed in methodology section.

6.5.1. Calculation of Blast Air:

Total carbon converted into CO/CO_2 = 380 kg (except carbon goes into hot metal and dust)

CO/CO_2 ratio is taken as 1.08

So 52 % carbon is converted into CO = 197.618 kg;

And 48 % carbon is converted into CO_2 = 182.417 kg;

O2 required to convert C into CO = 263.491 kg;

O2 required to convert C into CO2 = 486.441 kg;

  Total O2 required = 749.938 kg;

Now calculation of weight (wt.) of metalloid oxygen from burden has been shown in table 17.

Oxides	Wt. of oxides (kg/THM)	Wt of metalloid Oxygen
Fe_2O_3	1279.942	383.92 kg
FeO	80.591	18.53 kg
SiO_2	13.281	7.085 kg
MnO	1.161	0.261 kg
TiO_2	0.933	0.373 kg
P_2O_5	1.589	0.908 kg

Table 17: Calculation of Metalloid Oxygen from Burden.

Total metalloid Oxygen = 411.148 kg

So total O_2 required from blast air = (total O2 required to convert carbon into CO/CO2 – total metalloid oxygen – oxygen from decomposition of steam);

Steam is added to dry air = 32.294 kg (depending upon the H_2 required for indirect reduction and loss in blast furnace off gas)

O2 from decomposition of steam = 28.706 kg;

So Wt. of oxygen from blast air = 310.083 kg;

After considering 4% of total O2 loss i.e. equal to 12.403, total wt of oxygen required from blast air = 322.486 kg = 225.741 Nm^3;

So total blast air = total oxygen required from blast air / 0.21 (because 21 % O2 and 79% N2 is present in blast air);

Volume of total blast air = 1074.957 Nm^3;

2% of total blast air volume, O2 enrichment provided = 21.499 Nm^3

	O_2	N_2	H_2O
m3/THM	247.24	849.215	
WT. IN KG/THM	351.081	1061.52	32.294
MOLES/THM	10971.28	37911.42	1794.161

Table 18: Calculation of Volume, Wt. and Moles of Blast Air Compositions.

6.5.2. Calculation of Blast Furnace off Gas:

Volume of CO into blast furnace gas = (wt. of CO*22.4/12) = 368.8885 Nm^3;

Volume of CO2 in blast furnace gas = (wt. of CO_2*22.4/12) = 340.5125 Nm^3;

Volume of N2 in blast furnace gas = (total blast air*.79) = 849.2157 Nm^3;

Volume of O2 in blast furnace gas = (O2 lost*22.4/32) = 8.682342 Nm^3;

Volume of H2 in blast furnace gas = (H_2 from H_2O*22.4/2) = 12.05675897 Nm^3;

So total volume of blast furnace gas / THM = 1579.356 Nm^3;

Now percentage of gases and moles of the gases in blast furnace off gas have been shown in table 19.

	CO	CO_2	O_2	N_2	H_2
m3/THM	368.889	340.512	8.682	849.215	12.056
%	23.356	21.560	0.549	53.769	0.763
DENSITY	1.14	1.98	1.42	1.25	0.082
WT.IN KG/THM	420.532	674.214	12.328	1061.52	0.988
MOLES/THM	15019.03	15323.06	385.278	37911.42	494.327

Table 19: Calculation of Percentage, Weight and Moles of Composition of Off Gases.

6.6 HEAT BALANCE CALCULATION OF BLAST FURNACE:

Weight of every constituent is already converted into moles and standard heat value is taken from different books which have been already mentioned in literature review. Now here total heat generated and used heat has been calculated and also unused heat (heat loss due to convection, conduction, radiation, and heat taken by water supply to cool the blast furnace) percentage has been calculated. In heat balance calculation (-) means exothermic and (+) means endothermic.

6.6.1 Heat Generated in Blast Furnace:

Calorific value of coke =7000 Kcal/kg;

Calorific value of coal = 5000 Kcal/kg;

Now weight of coke generated heat = total carbon burn*0.77 = 292.627 kg;

Weight. of coal generated heat = total carbon burn *0.23 = 80.404 kg;

Total heat generated due to burning of carbon of coke and coal = 2450416.05 Kcal/THM (-);

Heat given by blast air is shown in table 20.

	O_2	N_2	H_2O
MOLES	10971.28	37911.42	1794.161
LH(KCAL/KG)	0	0	540
SH(KCAL/MOLES)	11.142	9.371	8.931
TOTAL HEAT	122220	355230	33461.09

Table 20: Heat Given by Blast Air Gas.

So total heat given by blast air = sum of all total heat of gases present in blast air = 510911.0763 Kcal/THM (-);

So total heat input in blast furnace = 2450416.05 + 510911.0763 =2961327 Kcal/THM (-);

6.6.2 Heat Used in Blast Furnace:

1. *Heat taken by hot metal:*

Heat is required to maintain the hot metal solution. So heat of solution of different element in hot metal has been calculated and shown in the table 21. Also latent heat and sensible heat taken by hot metal element is shown in table 21.

	MOLES	HEAT OF SOLUTION(KCAL/MOLE)	LH (KCAL/MOLE	SH (KCAL/MOLE)	TOTAL HEAT
Fe	16938.04	0	0	17.172	290843
Si	221.4286	-28.5	0	21.1	-1660.71
Mn	16.36364	1.2	0	17.88	312.3818
P	26.12903	-29.2	0	46.267	445.9181
Ti	11.66667	-13.35	0	11.38	-22.9833
C	3583.333	7.645	94.14	8.85	396406.3

Table 21: Heat Taken by Hot Metal Elements.

So Total heat taken by hot metal = sum of all elements total heat = 686323.8635 Kcal/THM (+);

2. *Heat taken by volatile matter*:

 Volatile matter (phenol) present in coke and coal = 48.34940893 kg;

Heat capacity of phenol = 122 Kcal/kg;

Total heat taken by volatile matter = wt. of volatile matter*heat capacity of phenol= 5898.62789 kcal/THM (+);

3. *Heat taken by moisture present in burden*:

Total moisture present in the burden = 2.670266572 kg = 148.3481429 moles

Sensible Heat of moisture = 1.353 Kcal/mole;

Latent heat of moisture =   540 Kcal/kg   ;

Total heat taken by moisture = (moisture moles*SH + LH* moisture weight) = 1642.658986 Kcal/THM (+);

4. *Heat taken by slag formation:*

Total weight of slag formed = 277.6908841 kg;

Heat produced during slag formation = 140 Kcal/kg-slag

Total Heat produced = 38876.72378 kcal (-);

Sensible heat of slag = 426 Kcal/kg-slag

Total sensible heat = 118296.3166   Kcal (+);

So total heat taken by slag = (-38876.72378 + 118296.3166) = 79419.59287 Kcal/THM (+);

5. *Heat taken by blast furnace off gas*:

Total blast furnace gas = 1579.35584 Nm^3/THM

Latent Heat of BF gas = 870 Kcal/Nm^3

Total LH heat of BF gas = 1374039.58 Kcal/THM (+);

Now calculation sensible heat taken by blast furnace off gas have been shown in table 22. Blast furnace off gases temperature is changes from approx. 1500^0C to 150^0C. So sensible heat value is calculated at 150^0C.

	CO	CO_2	O_2	N_2	H_2
MOLES	15019.03	15323.06	385.2789	37911.42	494.3271
SH (KCAL/MOLE)	1.072	1.52	1.14	1.07	1.09
TOTAL HEAT	16070.37	23137.82	427.659	41323.44	523.986

Table 22: Sensible Heat of BF off Gas Components.

So total sensible heat of BF off gas = sum of total SH of each gas in off gas

 = 81483.279 Kcal/THM (+);

So total heat taken by blast furnace off gas = (1374039.58 + 81483.27883) = 1455522.86 Kcal/THM (+);

6. *Heat taken by blast furnace dust*:

During blast furnace operation approximately at 1150^0C dust are generated and it come out with BF gas. So it takes sensible heat from fuel supplied. Calculation of sensible heat taken by dust has been shown in table 23.

	Fe_2O_3	SiO_2	Al_2O_3	CaO	MgO	C
MOLES	42.768	11.676	3.986	12.707	3.276	287.930
SH (KCAL/MOLE)	34.64	17.65	30.85	13.44	12.58	4.97
TOTAL HEAT	1481.93	205.9724	122.787	170.659	41.184	1436.773

Table 23: Sensible Heat of BF Dust Compounds.

So total sensible heat taken by BF dust = 3459.305701 Kcal/THM (+);

7. *Reaction heat*:

Elements took part in the reaction

	Fe_2O_3	FeO	C	SiO_2	MnO	P_2O_5	TiO_2	H_2O	O_2
Moles /THM	7983.91	1279.65	31669.69	221.42	16.36	11.271	11.67	1794.161	10586

Combustion of fuel at 1700 K;

$2C$ (21171.99391) + O_2 (10585.99696) → $2CO$ (21171.99391);
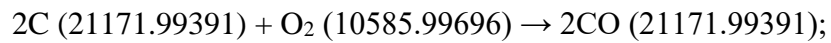
Now different reactions occur in blast furnace at different temperature. These all reaction and calculation of heat taken during the operation is shown as a figure, taken from Microsoft excel calculation sheet.

	reaction								TEMP.(K)	reaction heat(Kcal/mole)	total heat
	C +	H2O	→	CO +	H2						
moles	1794.161	1794.161		1794.161	1794.161				1700	32.1	57592.55
	3Fe2O3 +	CO	→	2Fe3O4 +	CO2						
moles	5189.542	1729.847		3459.695	1729.847				1000	-12.47	-21571.2
	Fe3O4 +	CO	→	3FeO +	CO2						
moles	3459.695	3459.695		10379.08	3459.695				1200	4.585	15862.7
	FeO +	CO	→	Fe +	CO2						
moles	11210.86	11210.86		11210.86	11210.86				1200	-3.85	-43161.8
	3Fe2O3 +	C	→	2Fe3O4 +	CO						
moles	2395.173	798.3911		1596.782	798.3911				1400	26.95	21516.64
	Fe3O4 +	C	→	3FeO +	CO						
moles	1596.782	1596.782		4790.347	1596.782				1400	44.43	70945.04
	FeO +	C	→	Fe +	CO						
	5174.242	5174.242		5174.242	5174.242				1700	27.48	142188.2
	3Fe2O3 +	H2	→	2Fe3O4 +	H2O						
	399.1956	133.0652		266.1304	133.0652				1200	-46	-6121
	Fe3O4 +	H2	→	3FeO +	H2O						
	266.1304	266.1304		798.3911	266.1304				1200	12.45	3313.323
	FeO +	H2	→	Fe +	H2O						
	862.3736	862.3736		862.3736	862.3736				1300	3.73	3216.654
	P2O5 +	5C	→	2P +	5CO						
	11.27358	56.36789		22.54716	56.36789				1700	206	2322.357
	TiO2 +	2C	→	Ti +	2CO						
	11.66667	23.33333		11.66667	23.33333				1700	69.33	808.85
	SiO2 +	2C	→	Si +	2CO					1700	158.89	35182.79
	221.4286	442.8571		221.4286	442.8571						
	C +	CO2	→	2CO					1300	40.05	23837.62
	595.1966	595.1966		1190.393							
		Fe(s)	→	Fe(L)					1800	3.56	61401
		17247.47		17247.47							
	MnO +	C	→	Mn +	CO				1700	69.33	1134.491
	16.36364	16.36364		16.36364	16.36364						
	H2O +	CO	→	CO2 +	H2						
	1261.569	1261.569		1261.569	1261.569				1000	-8.33	-10508.9

Figure 18: Calculation of Reaction Heat.

So total reaction heat = sum of all different reaction heat shown in figure 18.

= **357959.3** Kcal/THM (+)

6.6.3 Unused Heat Percentage Calculation:

Total heat supplied into the blast furnace = 2961327 Kcal/THM (-);

Total heat used in blast furnace operation = 2590226 Kcal/THM (+);

So total unused heat or heat lost by different process = heat supplied- heat used

$$= 371100.9 \text{ Kcal/THM}$$

So % heat loss in the process = (371100.9/2961327)*100 = 12.53157%;

The above calculation is for one particular burden percentage (50% sinter, 30% pellet and 20% iron ore lump). But when these burden percentage are change then the calculated value of theoretical carbon, coke and coal required, slag rate, blast air requirement, BF off gas and unused heat percentage are also changed. These changes are discussed below.

6.7 VARIATION IN THEORETICAL CARBON WITH VARIATION OF BURDEN PERCENTAGE:-

Change in theoretical carbon required due to change in burden charged percentage have been shown in table 24 and figure 19. From fig.19 it is clear that theoretical carbon required variation is linear with change in burden percentage. And as the percentage of sinter charge increased and pellet charge decreased, the theoretical calculated carbon is linearly decreased. Here in graph only sinter variation is taken because lump charged percentage is fixed and only sinter and pellet percentage is varied accordingly.

	Burden %		Theoretical Carbon(kg/THM)
Sinter	Pellet	Lump	
0	80	20	356.7735
10	70	20	355.3354
20	60	20	353.8973
30	50	20	352.4593
40	40	20	351.0212
50	30	20	349.5831
60	20	20	348.1451
70 (JSW BELLARI)	10	20	346.707
80	0	20	345.2689
45 (TATA STEEL)	40	15	350.2563
70 (SAIL DURGAPUR)	0	30	346.7988
75 (RINL VIZAG)	0	25	346.0339

Table 24: Change in Calculated Theoretical Carbon Due to Change in Burden%.
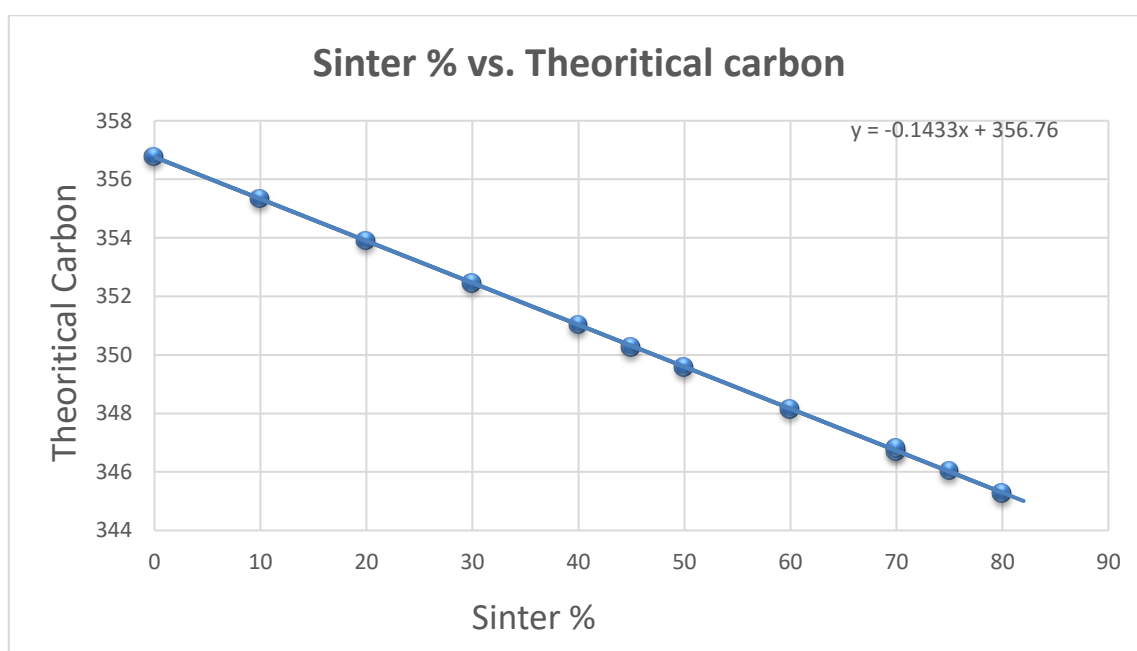
Figure 19: Linear Graph of Change in Burden% vs. Calculated Theoretical Carbon.

6.8 VARIATION IN SLAG RATE AND WT. OF FLUXES REQUIRED TO MAINTAIN SLAG BASICITY WITH VARIATION OF BURDEN PRECENTAGE:

Now change in slag rate and minimum flux required to maintain the basicity for different burden percentage have been shown in table 25.

	Burden %		Slag Rate(kg/THM)	Wt. of Lime stone (LOI free) (kg/THM)	Wt. of Quartz (LOI free) (kg/THM)
Sinter	Pellet	Lump			
0	80	20	203.9778	68.26646	0.5
10	70	20	216.8215	56.07189	0.5
20	60	20	231.0978	44.7285	1.178942
30	50	20	242.5091	31.68301	0.5
40	40	20	273.5938	30.32451	9.145704
50	30	20	277.6909	12.93412	5
60	20	20	290.9729	1	5.207743
70 (JSW)	10	20	324.3446	1	14.93736
80	0	20	357.7162	1	24.66701
45 (TATA)	40	15	265.4478	15.43056	4.544773
70 (SAIL)	0	30	323.8642	1	10.10205
75 (RINL)	0	25	340.7902	1	17.38454

Table 25: Change in Slag Rate and Wt. of Fluxes to Maintain the Slag Basicity for Different Burden %.
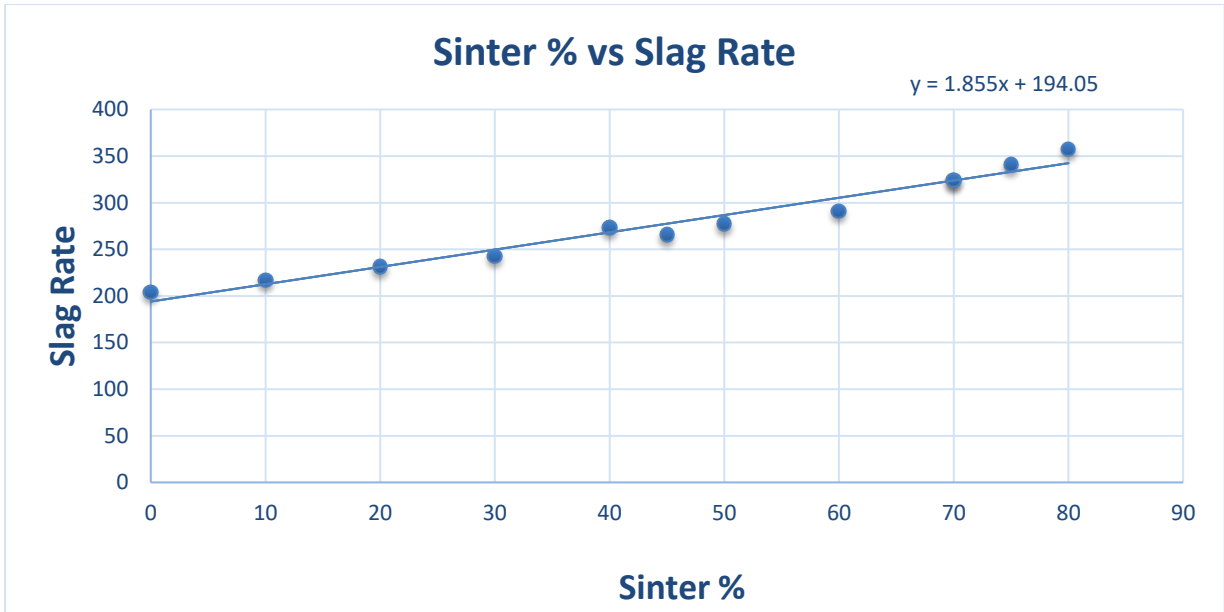
Figure 20: Graph of Sinter % vs. Slag Rate

From fig.20, it is clear that as the sinter percentage increases slag rate is also increased and it is maximum when 80 % sinter, 20% lump ore and 0% pellet is charged and it is minimum when 80% pellet, 20% lump ore and 0% sinter is charged. Here in graph only sinter variation is taken because lump charged percentage is fixed and only sinter and pellet percentage is varied accordingly.
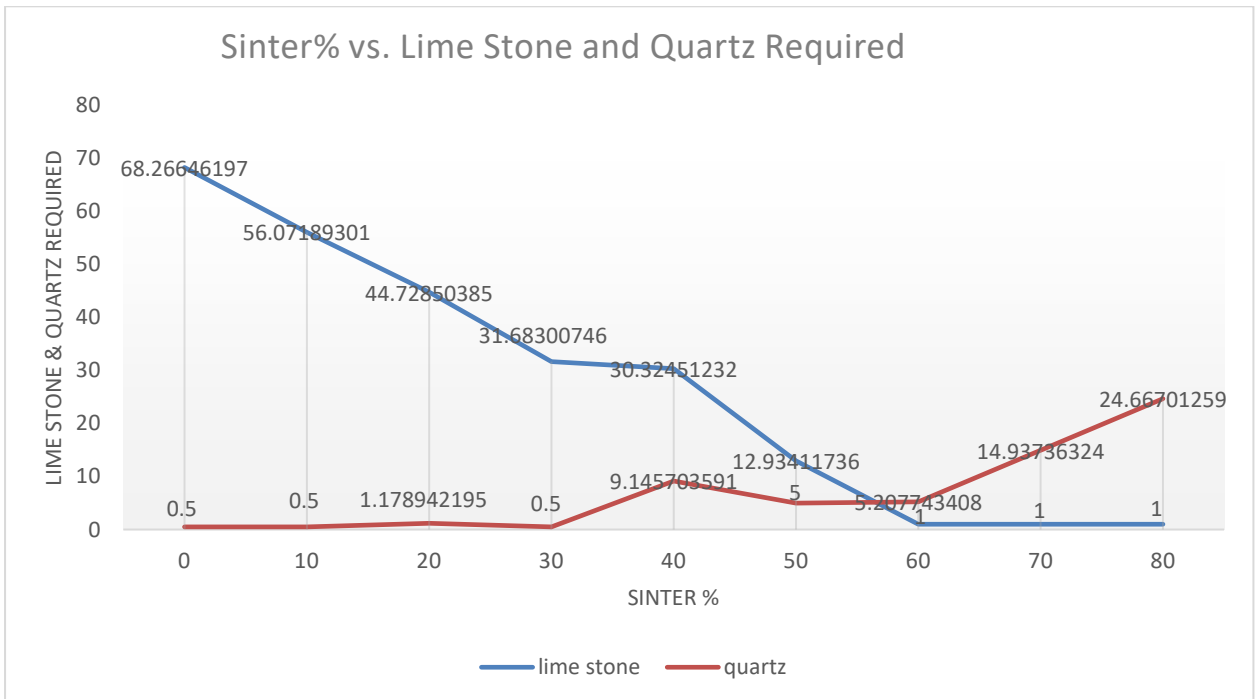
Figure 21: Trending Graph of Sinter % Vs. Fluxes Required.

In fig.21, graph has not followed linear trend. But it has been observed that for low sinter percentage requirement of lime stone is much more as compare to the quartz. This is because pellet has the basicity in between .5 to .8 and basicity of sinter is high and value is around 2 to 2.5. It means sinter contains more CaO so when sinter content in burden has been increased then requirement of quartz has been increased to maintain the basicity 1.05. And also when pellet percentage is high (as in pellet the SiO_2 % is more than CaO) then to maintain basicity the requirement of lime stone is more.

6.9 VARIATION IN BLAST AIR AND BLAST FURNACE OFF GAS WITH VARIATION OF BURDEN PERCENTAGE:

Now the variation of blast air and blast furnace gas with respect to change in burden percentage has been shown in table 26 and also trending graph of sinter percentage vs. blast air required and BF off gas generated has been shown in figure 21.

	Burden %		Blast air	Off Gas
Sinter	Pellet	Lump		
10	70	20	1101.94	1614.179
20	60	20	1095.194	1605.474
30	50	20	1088.448	1596.768
40	40	20	1081.702	1588.062
50	30	20	1074.957	1579.356
60	20	20	1068.211	1570.65
70 (JSW)	10	20	1061.465	1561.944
80	0	20	1054.719	1553.238
0	80	20	1108.686	1622.885
45 (TATA)	40	15	1078.256	1583.544
70 (SAIL)	0	30	1061.612	1562.273
75 (RINL)	0	25	1058.166	1557.756

Table 26: Change in Blast Air Required and BF Gas Generated due to Variation in Burden%.
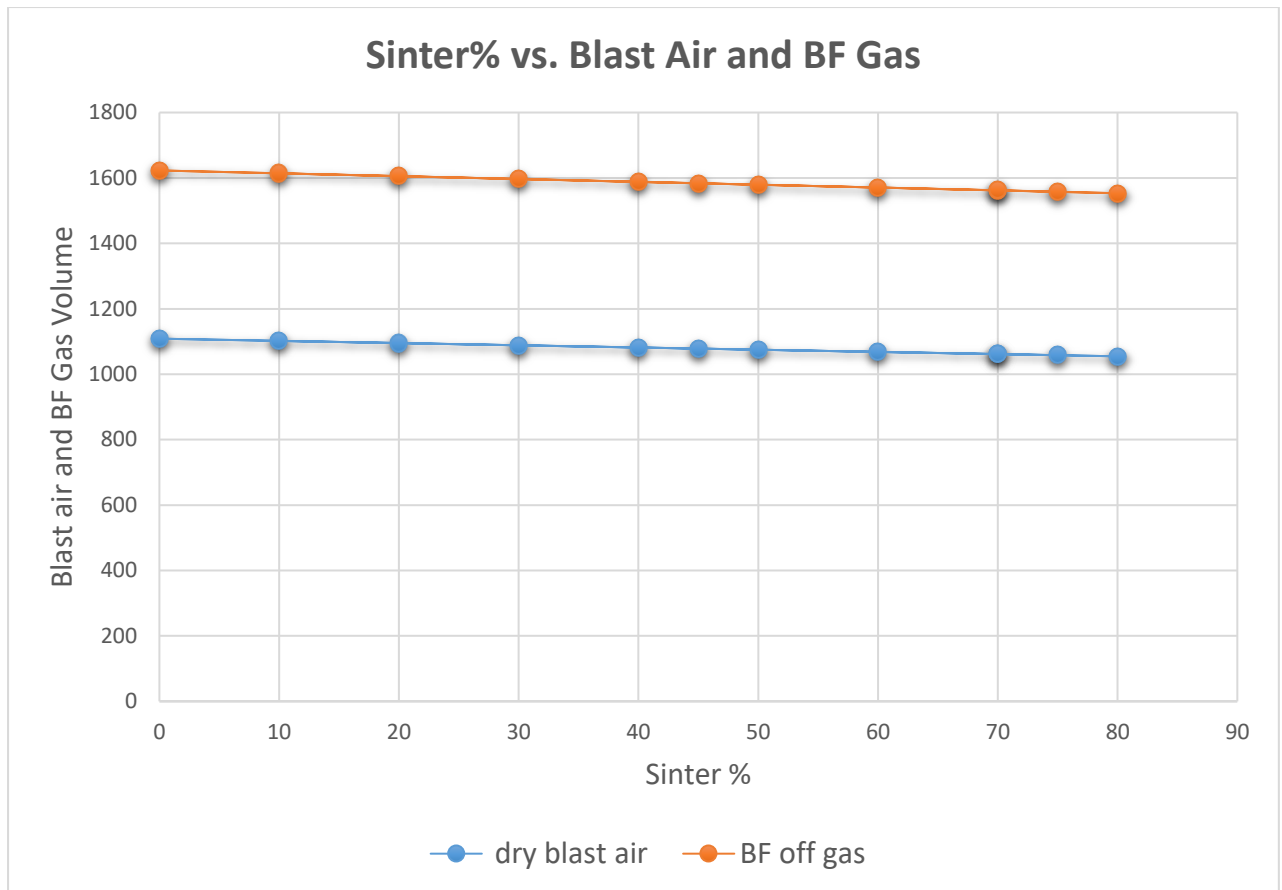
Figure 22: Trending Graph of Sinter% vs. Blast Air and BF Gas Volume.

From fig.22 (graph) it is observed that with increase in sinter percentage the requirement of blast air has decreased and BF gas generation has also decreased, because blast air and off gas is totally dependent on total coke and coal required and here extra carbon required has been fixed to 22% for all burden distribution, that's why it is following the same trend as fig. 19 (sinter % vs. theoretical carbon required). But if the extra carbon charged varies due to some further changes then the requirement of blast air and BF gas generation is also varied accordingly.

6.10 VARIATION IN UNUSED HEAT PERCENTAGE WITH VARIATION OF BURDEN PERCENTAGE:

When the burden percentage is changed then this unused heat value is also changed and it has been shown in table 27. And also trending graph of variation of unused heat percentage with sinter percentage has been shown in fig.23.

	Burden %		% Unused Heat or Heat loss
Sinter	Pellet	Lump	
10	70	20	13.679
20	60	20	13.405
30	50	20	13.156
40	40	20	12.715
50	30	20	12.531
60	20	20	12.256
70 (JSW)	10	20	11.782
80	0	20	11.304
0	80	20	13.937
45 (TATA)	40	15	12.723
70 (SAIL)	0	30	11.785
75 (RINL)	0	25	11.545

Table 27: Heat Loss % due to Change in Burden%.
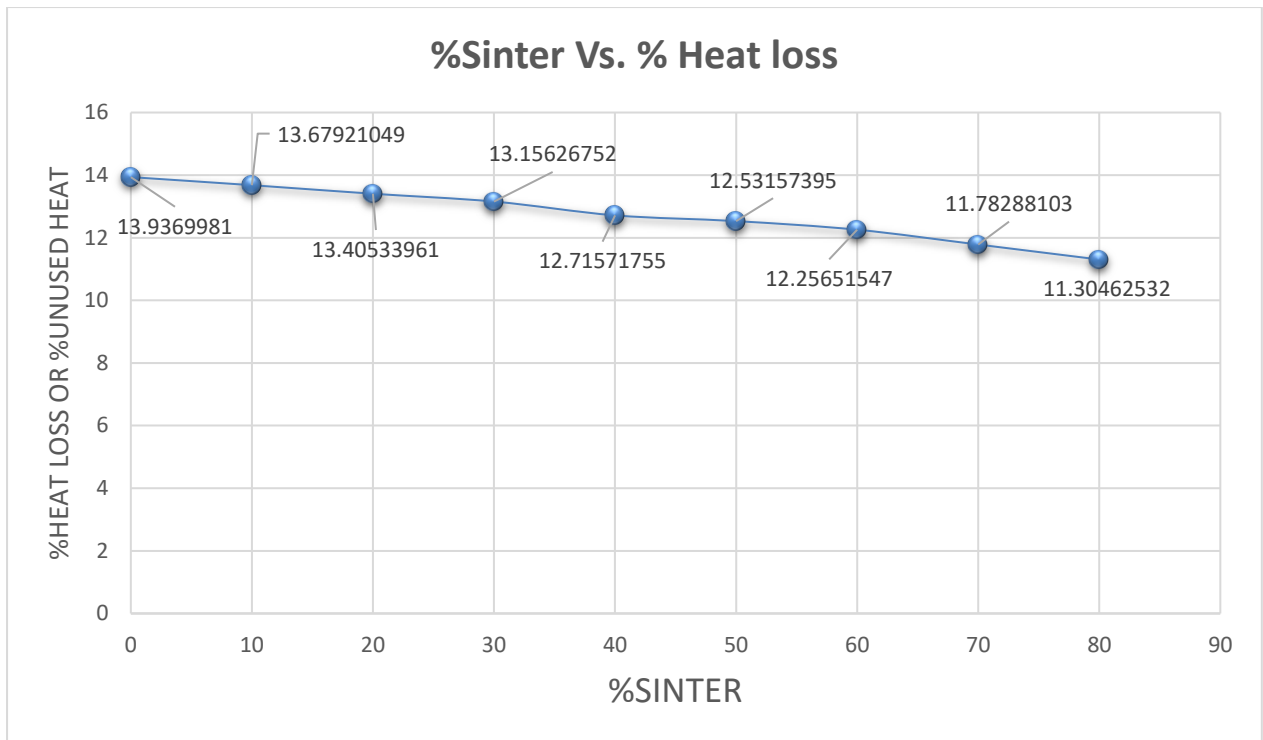
Figure 23: Graph of %Sinter vs. % Heat Loss or % Unused Heat.

From figure 23, it has been observed that as sinter percentage increased the unused heat percentage is decreased. It means more heaty is consumed for higher sinter percentage. But when pellet percentage is higher side then the unused heat percentage is more. So it means that the fuel charged is more than required and it can be decreased by reducing the fuel charged.

6.11 <u>VALIDATION OF CALCULATION</u>:

For the validation of calculation, a set of input and out data have been taken from an industry. Where the burden charge trend is 35.52% sinter, 56% pellet and 6.48% iron ore lump. Now this data have been put into the Microsoft excel modelling sheet and the calculated output data have been compared with that industry output data. A comparison table is shown below.

Material	Calculated data	Industry data	error in %
Pellet	830.2412	840	-1.161
Sinter	607.3866	600	1.231
Lump ore	95.57873	100	-4.421
Coke +coal	536.9138	550.82	-2.524
Slag rate	209.8307	219.6183	-4.456
Total BF dust	64.36346	70	-8.052

Table 28: Comparison Table of Calculated Data with Industry Data.

From error calculation it is clear that range of variation of calculated data with actual data are in between -5% to +5% and this is a considerable range. Hence calculation process is accurate and this modelling can be used for the further calculation.

6.12 COST CALCULATION AND COMPARISON FOR DIFFERENT BURDEN PERCENTAGE:

Unit cost of raw material and unit cost of output material have been already mentioned in assumption section. Unit cost of materials are flexible in nature it varies for different places as well as different plant. But for cost comparison unit cost of all materials are taken as a fixed amount. Now here calculation of total cost and difference between output cost and input cost have been shown for one set of data whose burden percentage is as follow, 50% sinter, 30% pellet and 20% lump.

Unit cost of sinter (in house production) = 4651 Rs. /Ton;

Unit cost of pellet (in house production) = 5600 Rs. /Ton;

Unit cost of lump = 3310 Rs. /Ton;

Unit cost of coke = 25801 Rs. /Ton;

Unit cost of coal = 11853 Rs. /Ton;

Unit cost of quartz = 2100 Rs. /Ton;

Unit cost of lime stone = 2350 Rs. /Ton;

Calculation of input material is shown in table 29.

INPUT MATERIAL	WEIGHT IN KG	COST CALCULATION	TOTAL COST(RS)
Sinter	842.802	Wt.*unit cost of sinter/1000	3919.874
Pellet	437.697	Wt.*unit cost of pellet/1000	2451.107
Lump ore	303.356	Wt.*unit cost of lump/1000	1004.11
Coke	384.046	Wt.*unit cost of coke/1000	9908.791

Coal	157.074	Wt*unit cost of coal/1000	1861.804
Quartz	5	Wt.*unit cost of quartz/1000	10.5
Limestone	12.9341	Wt.*unit cost of limestone/1000	30.395

Table 29: Calculation of Input Raw Material Cost.

So total input raw material cost = sum of all cost shown in table 29 =Rs. 19186.58

Unit cost of slag = 500   Rs. /Ton;

Unit cost of off gas calorific value = 1250 Rs. /Gcal;

Unit cost of pig iron = 27500 Rs. /Ton;

Cost calculation of output material is shown in table 30.

Output material	Wt. of material output (kg)	Cost calculation	total cost (Rs.)
Slag	277.690	(Wt. of slag)*(unit cost of slag)/1000	138.845
Off gas calorific value	1374040	(Calorific value) *(unit cost of calorific value)/1000000	1717.549
Hot metal	1000	Wt. of hot metal*unit cost of hot metal/1000	27500

Table 30: Cost Calculation of Output Material.

Total cost of output material = sum of all cost in table 30 = Rs. 29356.39

So cost difference between output material cost and input material cost is equal to = Rs.10169.81;

Now as the burden percentage changed, the input material cost, output material cost and cost difference are also changed and it has been shown in table 31.

Lump ore	Pellet	Sinter	Input cost (Rs.)	Output cost (Rs.)	Difference (Rs.)
20	80	0	19714.57	29366.88	9652.308
20	40	40	19317.64	29363.81	10046.18
20	30	50	19186.58	29356.39	10169.81
20 (JSW)	10	70	19016.44	29360.79	10344.34
20	0	80	18955.39	29368	10412.61
15 (TATA)	40	45	19388.18	29354.83	9966.65
30 (SAIL)	0	70	18694.4	29360.9	10666.5
25 (RINL)	0	75	18824.9	29367.08	10542.18

Table 31: Variation in Cost with Variation in Burden%.

Also this variation is shown as graphical form in figure 24.
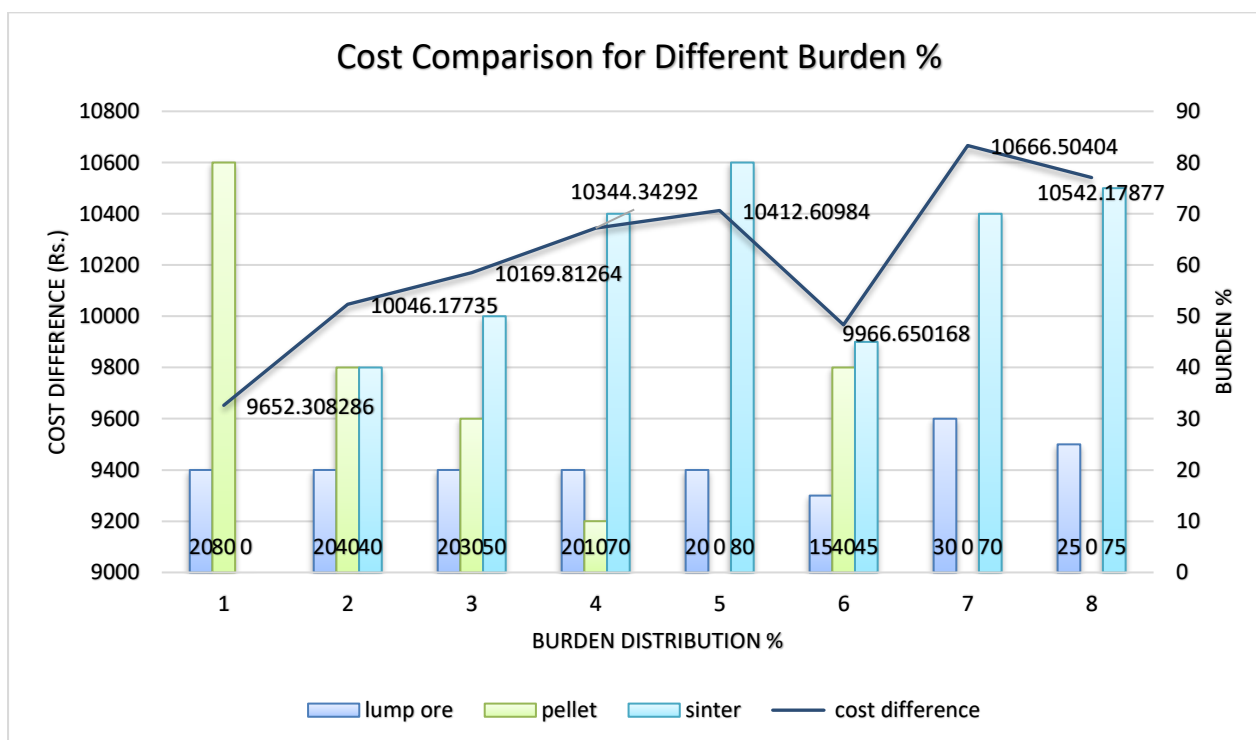
Figure 24: Graph of Cost Difference vs. % Burden Charge

From fig. 24 it is observed that when lump quantity is fixed to 20% of total burden and sinter percentage is increased then the cost difference is also increased and it is maximum for burden distribution of 30% lump and 70% sinter.

It is shown in the fig. 23 that as pellet percentage increased, the unused heat percentage has also increased. So to reduce the unused heat percentage, extra carbon charged (beyond the theoretical carbon requirement) can be decreased then the weight of input coke and coal will be decreased and hence the total price of input material will also decrease for same hot metal output. So the cost difference will increase.

CHAPTER 7:
CONCLUSION

7. <u>CONCLUSION</u>:

- Calculation has been carried out for all iron bearing burden percentage variation in Microsoft excel and data have been tabulated but detailed calculation of one set of data (50% sinter, 30% pellet and 20% lump ore) has been shown.

- Weight of burden required to produce one ton of hot metal has been calculated by proper material balance. The quantity of coke and coal required for reduction process has also been calculated by proper chemical stoichiometry. Also blast air required for process has been calculated.

- Other output material like weight of slag components, slag rate, weight of dust produce and volume of blast furnace off gas have been calculated and also weight of fluxes like limestone and quartz required to maintain slag basicity of 1.05 has been calculated by using solver technique in Microsoft excel.

- A proper heat balance is done to calculate how much heat remains unused or is lost by different heat transfer process.

- After that variation of required theoretical carbon, slag rate, weight of fluxes volume of blast air, blast furnace off gas and percentage unused heat with different burden percentage have been shown and it is observed that requirement of theoretical carbon is decreasing with increase in sinter percentage in burden. But slag rate is increasing with increasing with increase in sinter percentage in burden. As volume of blast air and blast furnace off gas is totally dependent on total carbon charged in blast furnace so these are also decreasing with increasing sinter percentage in burden.

- Weight of fluxes required is following a pattern i.e. when pellet percentage is high then requirement of lime stone is increased and requirement of quartz is negligible to maintain basicity but when sinter percentage increases the requirement of lime start to decrease and at higher percentage the requirement of lime stone is negligible and requirement of quartz is increased.

- Unused heat percentage or heat lost percentage is increased as the pellet percentage increases and sinter percentage decreases. It means that when the pellet quantity is high in the iron bearing burden then the requirement of heat for operation is less than the requirement of heat when quantity of sinter is high in burden.

- This Calculation model is also validated with industry data and the variation of calculation error is lying between ± 5%. So it can be concluded that this calculation model is quite precise with very low error.

- A comparison of net cost difference between output material and input raw material has also been carried out for different iron bearing burden percentages. It has been found that net cost difference is minimum for higher pellet percentage in burden (80% pellet and 20% lump) because price of pellet is higher than other iron bearing material and net cost difference is maximum for SAIL Durgapur burden percentage (70% sinter and 30 % lump). But it has been observed that for higher pellet percentage, unused heat percentage is also high so it can be decreased by decreasing the extra carbon (beyond the theoretical carbon) required for the operation. So the cost of coke and coal will decrease and net cost difference will increase marginally.

So from this thesis, one can find out comparative material balance, heat balance, weight of flux required and also cost of material and product in blast furnace iron making. From this analysis it has been found that burden consisting of 70% sinter and 30% lump ore is techno economically more acceptable because the cost difference is highest among all the variable combination of sinter, pellet and lump ore in burden. But it doesn't mean that it is best for blast furnace operation. Using of high pellet quantity in burden have their own advantage as it decreases the slag rate and also it has been discussed above that how the coke and coal quantity can be decreased for higher pellet percentage and their effects on the cost of production. Hence further calculation will be needed to carry out for achieving plant operating data in more elaborative manner.

CHAPTER 8:

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8. <u>REFERENCES</u>

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