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GRADUATE SCHOOL OF NATURAL AND
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STUDIES ON IMPROVING SINTER QUALITY

by
Ö. Saltuk BÖLÜKBAŞI

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İZMİR

STUDIES ON IMPROVING SINTER QUALITY

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by
Ö. Saltuk BÖLÜKBAŞI

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Ph. D. THESIS EXAMINATION RESULT FORM

We have read the thesis entitled **"STUDIES ON IMPROVING SINTER QUALITY"** completed by **ÖMER SALTUK BÖLÜKBAŞI** under supervision of **PROF. DR. TURAN BATAR** and we certify that in our opinion it is fully adequate, in scope and in quality, as a thesis for the degree of Doctor of Philosophy.

Prof. Dr. Turan BATAR
Supervisor

Prof. Dr. İ. Akın ALTUN
Thesis Committee Member

Prof. Dr. Erol KAYA
Thesis Committee Member

Prof. Dr. E. Ilker Ozer
Examining Committee Member

Assist. Prof Dr. İsmail BENZER
Examining Committee Member

Prof.Dr. Mustafa SABUNCU
Director
Graduate School of Natural and Applied Sciences

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STUDIES ON IMPROVING SINTER QUALITY

ABSTRACT

This thesis provides an overview of chemical and physical properties of different iron ores as well as their influences on sintering performance. The three different types of foreign iron ores and domestic iron ores usage in sinter making. The aim of this study is to present a new approach to the characterization of complex sinter mineralogy and production standard, especially those found in effect of sinter quality parameters and productivity.

A number of the commercial iron ores composition were used in an industrial sinter plant to study the effect of iron ores on the sinter quality were tested. For this purpose; preparation of sinter mix was done by considering the sintering conditions at Iskenderun Iron and Steel Works (ISDEMIR). Experiments were carried out in four stages. These are sinter applications at ISDEMIR, R & D laboratory test work was done operating agreement between China Metallurgical Equipment Corporation (MECC) and ISDEMIR, Statistic quality control of sinter samples and studies on sinter quality with mineralogical analysis. Sintering process was performed for each individual using iron ore as same on ratio on used of basicity, coke powder, flux and almost half of imported and domestic iron ore. For these tests; iron ores, coke breeze, flux material were made supplied by ISDEMİR. After the physical, chemical and mineralogical characterization of the sinter samples are investigated, they were fulfill to technological tests and the results obtained were compared with those obtained for other sinter samples. Characterization analyses were carried out by X-Ray Diffraction (XRD) and Scaning Electron Microscope (SEM-EDS). Statistical process control (IPC) techniques are measured for certain periods to variable physical, chemical and reducibility properties of sinter production. Sinter quality parameter determines for perfect sinter.

Keywords: Sinter, reducibility, iron ore, mineralogical analysis, statistical process control, basicity.

SİNTER KALİTESİNİN İYİLEŞTİRİLMESİ ÜZERİNE ÇALIŞMALAR

ÖZ

Bu tezde; farklı demir cevherlerinin kimyasal ve fiziksel özellikleri ile sinter performansı üzerine etkileri araştırılmıştır. Sinter yapımında üç farklı ithal ve yerli demir cevher kullanıldı. Bu çalışmanın amacı, kompleks sinter mineralojisi ve ürün standardı tanımlanması ile özellikle sinterin kalite parametreleri ve verimlilik anlayışında, yeni bir yaklaşım sunmaktır.

Endüstriyel sinter tesisinde kullanılan demir cevheri kompozisyonlarının, sinter kalitesi üzerine etkileri test edilmiştir. Sinter harmanları, İskenderun Demir ve Çelik (İSDEMİR) sinter şartları düşünülerek hazırlanmış ve deneyler dört aşamada tamamlanmıştır. Bunlar; İSDEMİR'de yapılan sinter uygulamaları, İSDEMİR ile Çin'deki MECC firması arasında yapılan Ar-Ge laboratuar çalışmaları, Sinter numunelerine istatiksel proses kontrol uygulamaları ve mineralojik analiz ile sinter kalitesi araştırmalarıdır. Sinter prosesinde farklı demir cevherleri kullanılarak yapılan çalışmalarda; aynı oranlarda kok tozu, flaks ile yarı yarıya ithal ve yerli demir cevheri kullanılmıştır. Testlerde kullanılan demir cevherleri, kok tozu ve flaks malzemesi İSDEMİR tarafından tedarik edilmiştir. Sinter numunelerinin fiziksel, kimyasal ve mineralojik karakterizasyonları araştırıldıktan sonra teknolojik testler yapılmış ve elde edilen sonuçlar diğer sinter numuneleri ile mukayese edilmiştir. Karakterizasyon çalışmaları, X-Ray Difraksiyon (XRD) ve Taramalı Elektron Mikroskobu (SEM-EDS) yardımıyla yapılmıştır. Sinter ürününün değişen fiziksel, kimyasal ve indirgenebilirlik değerleri istatiksel proses kontrol tekniği ile ölçülmüştür. İdeal sinter üretimi için parametreler belirlenmiştir.

Anahtar Kelimeler: Sinter, indirgenebilirlik, demir cevheri, mineralojik analiz, istatiksel proses kontrol, baziklik.

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CHAPTER ONE
INTRODUCTION

Sintering is the most common agglomeration process. Sintering is a process in which iron bearing materials of fine particle size are converted into coarse agglomerates by partial fusion.

The principle of agglomeration by partial fusion has been known for centuries. Sintering, as we know it today, originated in the nonferrous industry as a batch process in the late 19th. century; in June 1906, Dwight and Lloyd built the first continuous sintering machine, chain grate design. And development of both ferrous and nonferrous sintering in number of machines and capacity had continually increased (Dartnell, 1969).

Prior to World War II, sintering was principally employed to convert blast furnace flue dust and other steel mill iron bearing wastes to a useable blast furnace burden. Sufficient ore fines were added to these dusts to yield a sinter mix capable of being sintered economically. Subsequent to World War II, the bulk of the sintering strands have been installed to agglomerate virgin iron values present as concentrates and fine natural ores (Strassburger, 1969).

The quality of the product sinter is essentially depending on the raw materials used in sinter mix. The sinter mix consists of iron ore or iron ores, coke, return sinter fines, flux materials, and, moisture. The moisture content has a highly significant effect on sinter production. The chief function of moisture is to produce well balled mix for maximum bed permeability (Barnaba, 1985).

Because of the dramatically reduce of the high grade iron ores in the earth, necessity of enrichment of low grade iron ores, makes the particle size below the chargeable size to the blast furnace. While the mining and preparation of ores, undesired dust formation obtained, all of the ore dust which unsuitable for direct use in the blast furnace, can get suitable by agglomeration processes.

The iron ores and other raw materials having too fine particle sizes for blast furnace charging are agglomerated into the lump forms of appropriate size with suitable physical and chemical properties. This agglomeration can be done by sintering at temperatures that is high enough for binding of the fines by partial fusion. The sintered product has a porous, cellular structure, and its mineralogy may be substantially different from that of the original iron bearing fines (Ball, 1973).

Sintering is achieved by the application of heat which results in the conversion of ore fines into large, hard, porous lumps. In sintering, a shallow bed of fine particles is agglomerated by heat exchange and partial fusion of quiescent mass. Heat is generated by combustion of a solid fuel (coke) admixed with the bed of fines being agglomerated. The combustion is initiated by igniting the fuel exposed at the surface of the bed, after which a narrow, high temperature zone is caused to move through the bed by an induced draft applied at the bottom of the bed (Topkaya, Geveci & Aydoğdu, 1991).

Inside this narrow zone, the surfaces of adjacent particles reach fusion temperature, and gangue constituents form a semi liquid slag. The bonding is affected by a combination of fusion, grain growth and slag liquidation. The generation of volatiles from the fuel and flux stone creates a frothy condition and the incoming air quenches and solidify the rear edge of the advancing fusion zone. The product consists of the cellular mass of ore bonded in a slag matrix.

In modern blast furnace operation, careful burden preparation to provide a rigorously sized burden of consistent chemical analysis is essential in order to obtain high furnace productivity. Intensive work into the improvement of burden preparation and quality has identified facets of burden properties which can play an important part in furnace operation.

In the ferrous industry, the essential materials for sintering consist of a mixture of iron bearing fines and solid, particulates fuel. The iron bearing constituents are principally iron ore fines, recycled sinter fines, and flue dust; but may also include mill scale, open hearth precipitator dust, and similar iron bearing materials. Coke is the most common solid fuel used in iron ore sintering process. Also limestone and

dunite are added into the sinter mix as flux materials. The raw mix is tumbled in a rotating drum to ensure complete mixing and water is added during mixing to satisfy balling and to improve permeability. A hearth layer about 2 cm depth is first loaded on to the grate, and then the raw mix charged. Upon ignition, the surface of the bed is heated to about 1200°C - 1300°C, combustion of the fuel is initiated, and the fine particles at the surface are fused together (Topkaya, Geveci & Aydoğdu, 1991).

As it is becoming increasingly important to know the physical and chemical properties of the individual burden constituents, many test procedures have been and are being developed to determine and quantify various properties.

Taking into account that the performance of the materials inside the furnace under reducing conditions, it is accepted that it is difficult to develop tests which can precisely simulate furnace conditions. Many tests have been developed in an endeavor to obtain meaningful information which is of assistance to blast furnace operators. As the blast furnace is final smelter of any material, it is important to carry out blast furnace trials to evaluate a specific material under test conditions, in order to correlate the results with laboratory tests. Various practices in furnace operation and local economics must be borne in mind attempting to determine the acceptable level of any specific property of a burden (Sevinç, Topkaya, Geveci & Timuçin, 1987).

An important characteristic of a burden material is its reducibility, e.g., the ease with which oxygen can be removed. Reducibility data give an indication of the fuel required in the blast furnace and also provide information for determining the optimum size range in which any material should be employed. The ideal method of determining reducibility of iron bearing materials should use the same size grading as that charged to the furnace using gas flow rates, temperatures, and gas compositions, varying in a manner simulating that in the blast furnace. This, however, is difficult to effect for a routine method, as the apparatus becomes sophisticated and the procedure complicated, and several skilled operators are required (Ball, 1973).

The effect three different types of foreign iron ores and domestic iron ores usage in sinter making processes and effects of some sinter properties were examined. For this purpose preparation of sinter mix was done by considering the sintering conditions at İskenderun Iron and Steel Works (ISDEMİR). Experiments were carried out in three stages: preparation of the sinter mix, baking, followed by the treatment of the baked sinter by some standard tests. For the tests iron ores, coke, limestone and dunite were supplied by ISDEMİR. After the physical, chemical and mineralogical characterization of the samples, they were subjected to reducibility tests and the results obtained were compared with those obtained for other domestic and imported ores.

The main subjects of studies in this thesis are given below:

► These results were obtained experiment from production information and also, during sinter production and usage data of sinter technology at Isdemir. The sinter experiments and analysis were made with 40 different sinter productions.

Examples of sinter of obtained with used different domestic and foreign iron ore: These are the effect of sinter production efficiency, mineralogical and mechanical properties of sinter product, and then reducibility effects, and finally the sintering furnace is used in the production of liquid crude iron and the effects of productivity are investigated.

Examples of sinter of obtained with used different foreign iron ore in blend: iron ore A, iron ore B, iron ore C were supplied from Turkey, whereas foreign iron ore (FIO1), foreign iron ore (FIO2) foreign iron ore (FIO3) and foreign iron ore (FIO4) were used in sinter test pogram. These iron ores were uniformly mixed respectively to reduce fluctuation of chemical composition and obtain representative sample.

► R & D Laboratory test work was done operating agreement between China Metallurgical Equipment Corporation (MECC) and the ISDEMIR. The test results are summarized as test report worked out in great detail. Coke breeze consumption, basicity test, moisture test of raw mix, reducibility test, mineralogical analysis etc. of

sinter was investigated in these studies. Also, tumbler index of product sinter in hot pot sintering tests were compared with sinter plants of different countries.

► Statistical process control (IPC) techniques are used for Iron and Steel plants in which data are measured for certain periods to increase physical, chemical and reducibility properties of sinter production. Thereby, some important advantages are obtained for time control and applicability. In this study, Static and Dynamic Test low temperature reduction disintegration and reduction under load of the samples taken out regularly from sinter plant of Isdemir were investigated to determine capability of the process. For each parameter, X-R control charts using moving range method were drawn. Thus, deviations of lower and upper limit values were determined In addition, process capability indices (CP ve C) also were determined for specification limit values. It was observed that quantity of raw ores and grades should be fixed using suitable homogenization

► To present a new approach to the characterization of complex macrostructures and microstructures, especially those found in effect of sinter quality and productivity. Our results, similar to others, suggest that if the coalescence of fine pores was suppressed and the number of small pores was increased in the sintering process, the surface area of sinter could be increased to achieve a substantial improvement in its reducibility. An important feature of this system is the simultaneous use X-Ray Diffraction and Scanning Electron Microscope (SEM-EDS) which enables to determine both the macrostructure and the microstructure of a sinter with high accuracy. The primary aim of this project was to investigate why fine sinter is produced at the ISDEMIR sinter plant, from a phase chemical point of view. Differences between domestic and foreign ore sinter as determined in this project are as follows.According to the microprobe analyses there are small chemical variations between the phases in domestic and foreign ore sinter, however not substantial

The studies showed that a small increase in the alumina content of sinter blends can have a significant adverse impact on the strength and reduction degradation characteristics of final sinter leading to deterioration in gas permeability in the upper part of the blast furnace. Tumbler strength tests also showed that the mechanical

strength of sinter decreased with increase in $A1_2O_3$ content. Calcium ferrite is the main bonding phase in sinter. It increases with an increase of basicity. Generally, the high content of calcium ferrite favors the tumbler strength of sinter, but probably not for the RDI.

A decisive precondition for the production of high quality sinter is a homogeneous sinter raw mix of high permeability. All additives should be uniformly distributed throughout the mixture. The production of high quality sinter depends to a high degree on the chemical composition of the raw materials, especially with respect to the gangue content (SiO_2, MgO, Al_2O_3), interstitial water and the CaO/SiO_2 ratio of the sinter raw mix. Particle grain size is paramount in importance.

Blast furnace ironmaking today, the sinter plant no longer can be seen as a separate production unit, but must be fully integrated with the blast furnace to generate the ideal burden for optimized production and cost efficiency. In this study, sinter products which produced from different sinter blends in Isdemir are examined. ISDEMIR produce their sinter in sinter plant to use in blast furnaces. Properties of sinter product are determined by the required tests in ISDEMIR. Another aspect to consider is that most blast furnaces operate using a mixture of sinter, pellets and lump ore. Pellets and lump ore, however, are normally marketed with specific chemical compositions and qualities. An advantage in the use of sinter is that the burden can be optimized by adjusting the quality and the ratio of the charged sinter in accordance with the composition and characteristics of the other burden materials.

CHAPTER TWO
GENERAL INFORMATION ABOUT ISDEMIR

2.1 İskenderun Iron and Steel (ISDEMIR)

ISDEMIR is located in the Gulf of Iskenderun at the South Mediterranean coast of Turkey; Plants are 17 km to North of Iskenderun in the Payas (Yakacık) region. At the date of establishment is based on totally 16.75 millions m^2 areas together with the social facilities (Factory area 6.8 millions m^2). ISDEMIR is Turkey's third largest integrated iron and steel factory in terms of date of establishment and the largest in terms of long products (Iskenderun Iron &Steel Co [ISDEMIR], 2010).

The foundation works of ISDEMIR was started in 1966, some projects are carried on by Tyazhpromexport firm in the content of Technical and economical Cooperation Agreement that is made with Soviet Union on 25 March 1967, a factory foundation agreement is made with the same firm on 10 October 1969. ISDEMIR was designed to have 1.1 million tons/year bloom capacity and started to construct on 3 October 1970. The plants completed were made operational after individual completion of construction and installation activities, starting from 1975.

While the construction activities of the plants for 1.1 million tons/year capacity are continuing; works are started to increase to 2.2 million tons/year and the second slice credit agreement was signed with Soviet Union on December 24, 1972. The capacity was increased to 2.2 million tons/year steel blooms by implementing the plants starting from 1985 (Isdemir, 2009).

It was operational under the title of “Iskenderun Iron and Steel Works” as a part of the Turkish Iron and Steel Works General Directorate, then still an affiliate of the same General Directorate, and got the present title of Iskenderun Iron and Steel Works (ISDEMIR) in 14 October 1994 (Isdemir, 2010).

The works eventually was given under the control of Turkish Privatization Administration on March 2, 1998. Iskenderun Iron and Steel Works Co. was privatized by the purchase of Eregli Iron and Steel Works (ERDEMIR) on february

1st, 2002, provided with the conversion to flat product. With this transfer, investment activities have been initiated in ISDEMIR. The investment works which is called Modernization and Transformation Investment were planed at 2003 that started in 2004 and will finish at 2012. When these works completed, the capacity of ISDEMIR will increase from 2.2 million ton/year to 5,25 million ton/year intensively flat product production (Isdemir, 2011).

2.2 History of Iron Making

Iron is a metallic element, a metal of transition group VIII of the periodic table, with the symbol Fe from the Latin word ferrum. Iron has an atomic number of 26, an atomic weight of 55.847 and melting point of about 1535 °C (2975 °F) or lowers depending on the purity of the metal. Iron has the property of uniting chemically, also known as alloying, with numerous other elements which may improve its properties or have a deleterious effect. Iron makes up 5% of the earth's crust, second in abundance to aluminium among the metals and fourth in abundance behind oxygen, silicon and aluminium among the elements. Nowhere does iron occur as a usable metal. It is always found as an iron ore, which needs complicated processing before becoming a recognizable iron product (Isdemir, 2010).

All iron ores are basically oxides, which mean iron is chemically united with the element oxygen. These ores also contain small amounts of other elements such as manganese or phosphorus and are mixed physically with earthy materials such as sand, rock and clay. Iron making depends on eliminating the unwanted elements and foreign matter from the ore and controlling the amount of those elements which are beneficial. Because iron has a natural affinity to unite with other elements, it can take many forms but it is possible to divide it into three major categories: wrought iron, cast iron and steel. Today steel is by far the most important form but cast iron is still commercially produced and wrought iron has been resigned to ornamental applications. Wrought iron is the oldest iron product dating back at least four thousand years. It is the commercially pure form of iron and has a fibrous structure. It is strong in tension, that is it resists forces tending to stretch it, but it can be shaped by hammering, squeezing or rolling (Isdemir, 2010).

Cast iron, which dates from the fourteenth century, is crystalline and relatively weak in tension. It cannot be shaped by hammering but it can be melted and poured into a meld, the shape of which it will retain after it cools and solidifies. Cast iron is an alloy of iron and carbon, which may contain up to about 5% of the latter element. Steel is the most widely used and versatile form of iron. It can be of a simple composition chemically or it can become a complex alloy containing a number of other elements, which control the desired properties of the finished product. Bulk steel manufacturing only became possible after the invention of the Bessemer process in 1856, although small quantities of a simple form of steel were made hundreds of years earlier (Isdemir, 2011).

Now that this introduction is complete, the history of iron making will be presented in more detail with a chronological listing of its migration from the cradle of civilization and a description of the evolution to modern production technology.

2.3 Coal Preparation and Coke Manipulation Facility

2.3.1 Coal Preparation Facilities

The Coal Preparation Facilities are built in order to stockpile, prepare, blend, weigh and blast furnace feed the various types of coking coal imported from foreign sources via sea transport.

2.3.2 Coke Oven Batteries

Coke has certain basic functions during the steel production that cannot be replaced by any other material. Some of these functions are its reducibility, and serving as a heat source and forming a structural frame in the blast furnace. Therefore the coke batteries are vital to iron and steel producing plants. Furthermore the iron and steel production process in the integrated mills consumes and stores huge amount of energy. Figure 2.1 indicates that a general view of the the coke oven.

Part of this energy is produced in the batteries during coke production. Metallurgical coke is produced by distilling the coking coal or coal blends, in the coke cabins covered with refractors in airless surroundings, i.e. by decomposing the

volatile matter in a high temperature environment. Coke batteries are formed by the compilation of certain number of coke cabins (Isdemir, 2010).

Figure 2.1 A general view of the coke oven. The oven has just been "pushed" and railroad car is full of incandescent coke that will now be taken to the "quench station" (Isdemir, 2010).

2.3.3 Dry Coke Extinguishing Facility

The red hot coke brought by the extinguishing wagon is taken to one of these five blocks. This process is performed by the cranes, and the red hot coke is extinguished in the extinguishing cabin. 40 atm. pressured steam is produced in the steam boiler from the circulation gas used in extinguishing.

By products; is established with the aim of obtaining clean coke gas by cooling the coke gas produced during the coking of the coal in the coke batteries, decomposing it after condensing with tar and steam and clarifying from ammonia and aromatic hydrocarbon (benzene and its derivatives). This clean gas will be delivered to the steel making units. The by products obtained during the clarification of coke gas process will also be available for sales. In these facilities, items such as raw tar, ammonium sulphate (manure), raw benzene is produced (Isdemir, 2011).

2.4 Sinter and Raw Material Manipulation

The purpose of sintering is to burn the sinter blend constituting of fine iron ore, ferrous and other complementary materials (limestone, dolomite etc) with coking coal so that it can be used in the blast furnace.

Sinter and Raw Material Manipulation Management are functioned in two departments, namely Sinter and Raw Material Preparation. Raw Material Preparation and Manipulation Department; Iron ores from various parts of Turkey are discharged from stockyard build stocks while the rest deliver the ore from the stockyard railway wagons to the discharging unit. The wagons are generally fal and fals type of 50-55 tons. At this unit there are also 4 bunkers. The ore is unloaded from the wagons into the bunkers by cranes. The material is carried to the stockyard via conveyor belts from the bunkers. Furthermore, ore is transferred by the conveyor belts to the stockyard and the breaking unit. Figure 2.2 gives that a general view of the sinter plant. The ores coming from the wagon discharging unit and the port are stockpiled according to their types (Isdemir, 2010).

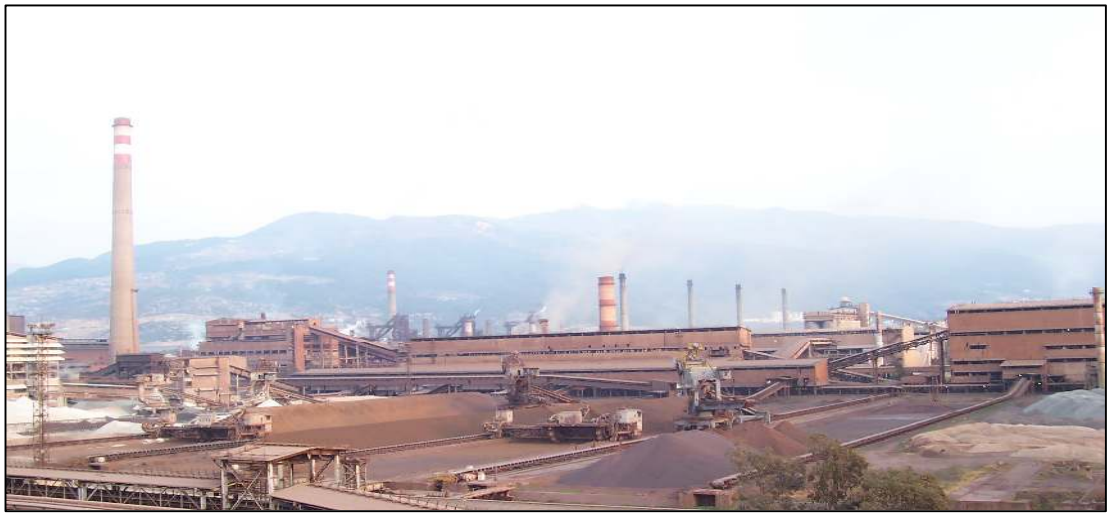


Figure 2.2 A general view of the sinter plant. The field of stock and blending with sintering is seen (Isdemir, 2010).

Universal machines provide ore stocking in the stockyard with 8 halls and also to receive material from the stockyard. This is done with the machines' rotating booms and conveyors.

2.4.1 Sintering Department

Sintering unit has three main sections. These are proportioning unit, sinterin unit and sludge preparing and drying unit.

2.4.1.1 Proportioning Unit

Material is discharged from the bunkers by computerized weighbridge conveyor belts onto the collecting conveyors. The iron ore is taken into the dosing bunkers directly after blending accordingly. Limestone and dolomite stone are taken into the 3 closed circuit bunkers to be broken and screened down to 0-3 mm size.

The screened limestone and dolomite stone powders are stockpiled in the dosing unit bunkers. The coke breeze used as the fuel is obtained from the 4 coke oven batteries and from the coke on stock. These are transferred to the coke breaking bunkers. Here they are broken down to 0-3 mm size by the 4 breakers of 16 ton/hour capacity and then stockpiled at the coke breeze dosing unit. The sinter powder to be combined to the blend comes from the sinter screening unit. Also the sinter powder screened at the blast furnaces is stored in the dosing unit bunkers (Isdemir, 20110).

2.4.1.2 Sintering Unit

Sinter unit is composed of sinter cooling and screening, sinter loading and laboratory, cyclone concrete sides, exhauster fan and electro filter units. The blending material taken from the dosing unit and then to the collecting conveyor are mixed. The water feed amount here changes according to the season and the moisture content of the blend. Figure 2.3 gives that a general view of the sinter machine.

Figure 2.3 A general view of the Sinter machine (Isdemir, 2010).

The material taken out of the mixer is transferred to the sinter unit via conveyor belts. The blend arriving to the sintering unit is delivered to the sintering machines via tripper cars.

Material below 8 mm. taken out of the screen beds is taken back to the sinter facilities. The excess bedding material is then sent to the blast furnaces. The particles below 8 mm. are sent to the dosing unit to be used in the sinter blend. As the sinter gets broken during the transfer to the blast furnaces, another screening under the bunker is performed. The undersize obtained here is also used in the dosing unit.

There are 5 electro filter units to avoid the small particles from spreading around during the discharging of the sinter to the conveyor belts. The particles are collected into the bunkers through pipes in various diameters and hold in the electrodes. These small particles are also sent via the conveyor belts to the dosing units to be used in the dosing unit (Isdemir, 2011).

2.4.1.3 Sludge Preparing and Drying Unit

In Sludge Preparing and Drying Unit, the watery sludge coming from melt shop gas cleaning, blast furnace cleaning, sinter and raw material preparation units are subject to sedimentation via physical and chemical methods and purified to be reused

in blast furnaces, raw material and sinter preparation units. The resulting sludge will be dried at %10 to stocking field (Isdemir, 2011).

2.5 Blast Furnaces

Isdemir Blast Furnace unit has three blast furnaces called Cemile, Ayfer and Gonul. Gonul is the largest blast furnace. Blast furnaces are built to produce the crude steel that the melt shops require. The main duty of these three blast furnaces is to produce liquid crude steel. Sinter, pellet, iron ore and coke are charged to the blast furnace to get crude steel. Figure 2.4 gives a general view of the Blast Furnace.

Figure 2.4 A general view of the Blast Furnace. Photograph of a blast furnace at the end of a tap with the mud gun in position at the taphole. The taphole drill is retracted and ready to be swung into position for use for the next tap (Isdemir, 2010).

Together with the above mentioned raw materials, other ferrous materials and slag making materials such as limestone, silicon dioxide and dolomite are also added to be melted with the metallurgical coke burning in the furnace to obtain the crude steel. It constitutes of a top, body, bosh and a tank. There are vertical and horizontal coolers inside the sheath. The coolers are covered with refractory material. The top bricks are low alumna, and the alumna content of the bricks increase towards the lower parts. The furnace tank is covered with carbon and graphite bricks. Blast furnaces have been preferred for years because of their high efficiency of crude steel production (CSP). Today, the modern furnaces are more efficient with coal injection and the oxygen enriched air inflow. This helps reducing fuel consumption. The existence of blast furnace in the integrated mills is the most important cost reducing

and quality enhancing factor. Charging material is fed from the top part of the furnace. The charge materials move downwards in the furnace. With the high temperature air blown in at 900-1000 °C burning the coke creates CO and H_2 gases and leads to a reaction as a result of contact with ferrous materials. As a result the metal oxides are reduced and converted to crude raw steel. Other oxides that are not reduced form slag. To produce crude raw steel in the blast furnace ferrous materials such as sinter, pellet, and lump ore are used. Sinter contains 55-58%, pellet 65-67% and lump ore 59-60% Fe. Metallurgical coke is used as fuel. The quality of metallurgical coke and ferrous materials is strictly important. Figure 2.5 gives that production flow of Isdemir (Isdemir, 2010).

Coke provides the composition of reducing gases and also forms a layer between the ferrous materials allowing the transition of gases in the blast furnace. At the same time it provides exothermic reaction and secures the necessary temperature in the furnace for the formation of melting and slag production. As a result crude raw steel gathers at the bottom of the tank and slag at the top. Normally 8 or 10 times a day the crude raw steel and slag are discharged from the furnace to separate ladles. During the crude raw steel blast furnace gas and slag are formed as by products. 1/3 of the blast furnace gas is used in heating the stoves and the rest in the Power Plant and Coke Plant. Slag is granulated and sold to cement factories as raw materials (Isdemir, 2010). Figure 2.6 gives capacities of the main facilities.

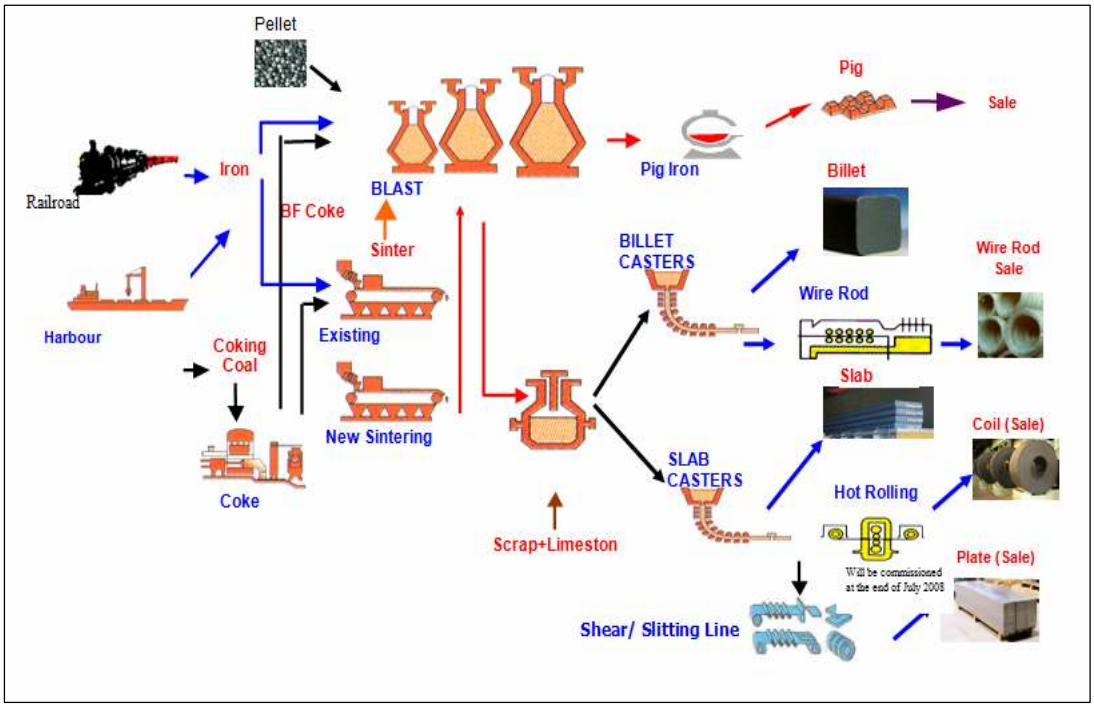


Figure 2.5 Production flow of Isdemir (Isdemir, 2010)

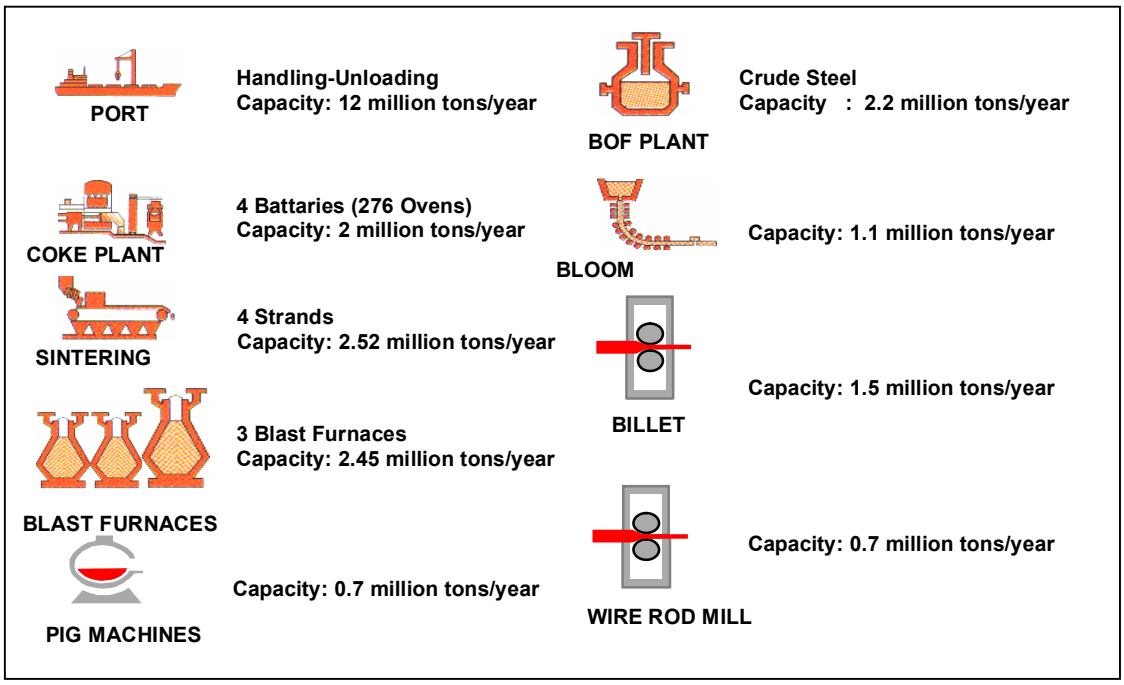


Figure 2.6 Capacities of the main facilities (Isdemir, 2010).

CHAPTER THREE
NATURE AND MINING OF IRON ORE

3.1 Nature of Iron Ore

Iron is only found in metallic state in certain types of meteorites but, in a combined form, it is one of the most common elements, comprising some 5 % of the earth's crust. It is most usually found as an oxide, sometimes a hydrated oxide, but it also occurs as carbonate and sulphide and as a component of a wide range of complex minerals (Boyd & Ferron, 1997).

An iron bearing mineral can only be considered to be an iron ore if the total cost of extracting iron from it is comparable with the cost of extracting iron from other ores. This will be governed by many factors, of which the iron content of the mineral, the nature of the impurities and the location of the deposit are of particular importance (Wens, 1999).

3.2 Geology of Iron Ore

Iron ores have a wide range of formation in geologic time as well as a wide geographic distribution. They are found in the oldest known rocks of the earth's crust, with an age in excess of 2.5 billion years, as well as in rock units formed in various subsequent ages; in fact, iron ores are forming today in areas where iron oxides are being precipitated in marshy areas, and where magnetite placers are being formed on certain beaches (Portillo, 1996).

Many thousands of iron occurrences are known throughout the world. They range in size from a few tonnes to many hundreds of millions of tonnes. Many of the world's largest deposits of iron ore are located in the oldest geologic series the Precambrian. Table 3.1 demonstrates the geologic age for selected iron deposits.

Table 3.1 Geologic age of selected iron ore deposits (Kirk, 1996).

Geologic Age	**Deposit**	**Location**
TERTIARY PERIOD		
Pliocene Epoch	Kerch Oolitic Limonite(a)	Crimea, Russia
	El Tofo Magnetite	Chile, South America
Oligocene Epoch	Cheikh-ab-Charg Hematite	Iran
Eocene Epoch	Upper Assam Clay Ironstones	India
	Bahariya Hematite	Egypt
Mesozoic Era		
CRETACEOUS PERIOD	Salzgitter Limonite and Hematite(a)	Germany
	Bilbao Hematite	Spain
	Algerian and Moroccan Magnetite	North Africa
JURASSIC PERIOD	Minette Limonite and Hematite	France, Germany
	Iron Springs Magnetite	Utah
TRIASSIC PERIOD	Kashmir Calcareous Iron Ore (Hematite)	India
Paleozoic Era		
PERMIAN PERIOD	Damuda Sandstone (Hematite)	India
Pre-Cambrian Era[(b)]		
	Minas Gerais and Serra dos Carajás Hematites[(a)]	Brazil
	Krivoi Rog Hematite[(a)]	Ukraine, Russia
	Bihar, Orissa and Mahya-Pradesh	India
	Labrador Hematite[(a)]	Quebec and Labrador
	Lake Superior Taconites and	Michigan, Wisconsin,
	Jaspilites, Hematites and Magnetites[(a)]	Minnesota, Ontario
	Cerro Bolivar and El Pao Hematites[(a)]	Venezuela
	Kirunavaara Magnetite[(a)]	Sweden
	Hamersley Range Hematites[(a)]	Western Australia
	Nimba Range Hematite([(a)]	Liberia and Guinea
	Sishen Hematite[(a)]	South Africa
[(a)] Well-known, important deposits.		
[(b)] Represents the time span of the combined Proterozoic and Archeozoic Eras.		

3.3 Iron Bearing Minerals

A large number of minerals contain iron; however, only a few are used commercially as sources of iron. Minerals containing important amounts of iron may be grouped according to their chemical compositions into oxides, carbonates, sulfides and silicates. Table 3.2 indicates the oxide and carbonate classes of iron minerals and indicates the mineral species commonly used as sources of iron. Oxide minerals are the most important sources of iron, with the carbonates, sulfides and silicates being of minor importance. In the descriptions of the important iron ore minerals or mineral groups that follow, the chemical compositions are for the pure minerals the iron content of commercial ores or concentrates generally is lower, due to the presence of gangue and other impurities (Boero, 1996).

Table 3.2 Chief iron bearing minerals (Hartmur, Donald, Charles & Jonathan, 1999).

Class and Mineralogical Name	**Chemical Composition of Pure Mineral**	**Common Designation**
Oxide		
Magnetite	Fe_3O_4	Ferrous-ferric oxide
Hematite	Fe_2O_3	Ferric oxide
Ilmenite	$FeTiO_3$	Iron-titanium oxide
Limonite	$HFeO_2$ [(a)] $FeO(OH)$ [(b)]	Hydrous iron
Carbonate		
Siderite	$FeCO_3$	Iron carbonate
[(a)] Goethite [(b)] Lepidocrocite		

All iron ore deposits contain one or more iron-rich minerals and the physical and chemical properties of any specific ore are related to the mineralogy of the deposit. Today, in beneficiation and agglomeration plants, the mineralogy is often more important than the grade of the crude ore, as the concentration and indurations properties are dependent upon the minerals present and on their physical relationship in the ore (Antunes, 1998).

The most important iron ore minerals can be grouped according to their chemical composition into oxides, hydrous oxides, carbonates, sulphides, and iron silicates (see table 3.3). The great bulk of iron ore consists of the first two groups which are

oxides and hydrous oxides. Iron carbonates and sulphides are local sources of iron and, although mined for their iron content, are usually converted to oxides by calcining and roasting before being sent to the blast furnaces (Kruger, 1997).

Table 3.3 Characteristic of iron bearing minerals (Bailey, 1992).

Iron Ore Minerals	Chemical Composition	%Iron	Specific Gravity	Hardness	Crystallization	Color	Notes and Distinguishing Characteristics
Oxides							
Magnetite	Fe_3O_4	72.4	4.5-5.3	5.5-6.5	Isometric Octahedrons	Black	Strongly magnetic
Hematite	Fe_2O_3	69.94	5.2	5 to 6	Hexagonal Rhombohedral	Red to steely blue and brownish black	Red streak, non-magnetic
Martite	Fe_2O_3	70	4.8-5.3	6 to 7	Octahedrons	Black	Pseudomorphic after magnetite
Maghemite	Fe_2O_3	69		5	Isometric Massive	Brown	Highly magnetic
Ilmenite	$FeTiO_3$	36.8-Fe 31.6-Ti	4.7	5.5-6.7	Hexagonal Rhombic	Iron black to brownish black	Slightly magnetic
Hydrous Oxides							
Goethite	FeO(OH)	62.9	4.3	5 to 5.5	Orthorhombic	Yellowish brown to dark brown	
Lepidocrocite	FeO(OH)	62.9	4	5	Orthorhombic	Rudy red to reddish brown	Streak dull orange
Limonite	$FeO(OH)nH_2O$	Variable 52.3-66.3	2.7-4.3	Variable 4 to 5-1/2		Various shades of brown and yellow	A field term for hydrous iron oxides of uncertain species
Turgite	$2Fe_2O_3.H_2O$ (variable H_2O)	40-50	4.2-4.7	2 to 4	Amorphous	Reddish black to dark red	Not a mineral species. in part a metacolloid
Carbonates							
Siderite	$FeCO_3$	43.2	3.8		Rhombic	Light to dark brown	
Sulphides							
Pyrite	FeS_2	46.6	5	5 to 6.5	Isometric	Brass yellow	
Pyrrhotite	$Fe_{1-x}S$	57-63.5	4.6	3.5-4.5	Hexagonal	Bronze yellow	Magnetic

In addition, iron is recovered from a number of minerals from which other saleable products are also extracted: for example, from pyrites (FeS_2) and pyrrhotite (FeS), in addition to sulphur; from ilmenite ($FeTiO_3$), in addition to titanium, and from complex ores, in addition to such metals as nickel, copper, cobalt and vanadium.

3.3.1 Hematite

Hematite has a chemical composition of Fe_2O_3 corresponding to 69.94% iron and 30.06% oxygen, has a color from steel gray to dull red or bright red, can be either earthy, compact or crystalline, and has a specific gravity of 5.26. Common varieties are termed crystalline, specular, martite (pseudomorphic after magnetite), magnetite (magnetic ferric oxide), earthy, ocherous, and compact.

Hematite is one of the most important iron minerals. It has a wide occurrence in many types of rocks and is of varying origins. It occurs associated with vein deposits, igneous, metamorphic, and sedimentary rocks, and as a product of the weathering of magnetite. Some low grade deposits of disseminated crystalline hematite have been successfully treated by both gravity and flotation techniques to produce high quality concentrates (Kirk, 1991).

3.3.2 Magnetite

Magnetite has a chemical composition of Fe_3O_4, corresponding to 72.36% iron and 27.64% oxygen; has a color of dark gray to black, and a specific gravity, 5.16 to 5.18. It is strongly magnetic, sometimes possessing polarity so it will act as a natural magnet. The magnetic property of magnetite is important, for it permits exploration by magnetic methods and makes possible the magnetic separation of magnetite from gangue materials to produce a high quality concentrate. Magnetite occurs in igneous, metamorphic, and sedimentary rocks. It has become increasingly important as a source of iron as a consequence of the continued improvements in magnetic concentration techniques and in the expanded use of the high grade products. At times, magnetite contains titanium in small amounts as inclusions of ilmenite. When the titanium content reaches 2-15 % or more, the magnetite is termed titaniferous magnetite (Bolis & Bekkala, 1987).

3.3.4 Hydrous Oxides

Limonite is the name commonly given to hydrous iron oxides that mineralogically are composed of various mixtures of the minerals goethite or lepidocrocite. The chemical formula for goethite is $HFeO_2$ and that for lepidocrocite is FeO(OH). Goethite contains 62.85% iron, 27.01% oxygen, and 10.14% water; has a specific gravity in the range of 3.6-4.0, is commonly yellow or brown to nearly black in color, and is compact to earthy and ocherous. In non-technical parlance, the term limonite is used to denote unidentified oxides with variable moisture content due to absorbed or capillary water. It is a secondary mineral, formed commonly by weathering, and occurs in association with other iron oxides and in sedimentary rocks (Chavarri, 1997).

3.3.5 Siderite

Siderite has a chemical composition of $FeCO_3$ corresponding to 48.20 % iron, 37.99 % CO_2 and 13.81 % oxygen, a specific gravity of 3.83-3.88, and a color from white to greenish gray and brown. Siderite commonly contains variable amounts of calcium, magnesium or manganese. Siderite varies from dense, fine grained and compact to crystalline. The siderite ores are sometimes termed spathic iron ore or black-band ore. Carbonate ores are commonly calcined before they are charged into the blast furnace. They frequently contain enough lime and magnesite to be self-fluxing (Antunes, 1998).

3.3.6 Ilmenite

Ilmenite has a chemical composition of $FeTiO_3$, corresponding to 36.80% iron, 31.57% titanium, and 31.63% oxygen. This is commonly considered an iron titanate. Ilmenite is often associated in small amounts with magnetite. Although generally mined as a source of titanium rather than as an ore of iron, iron may be recovered as a byproduct (Skillings, 1996).

3.3.7 Impurities

All iron ores contain impurities, which are collectively known as gangue. The complex mineral chamosite can be regarded as a mixture of iron oxide (FeO) and gangue (SiO_2 and Al_2O_3). These impurities may be divided into the following categories (Tripathi, 1997).

3.3.8 Slag Forming

These are, in the main, four oxides namely silica (SiO_2) and alumina (Al_2O_3) which are acidic and lime (CaO) and magnesia (MgO) which are basic. Of these oxides, only SiO_2 can be reduced during ironmaking and usually only to a very limited extent. In conventional ironmaking by the blast furnace process, a liquid slag is formed in which the ratio by weight of bases (CaO+MgO) to acids (SiO_2 + Al_2O_3), called the basicity, is normally about one. Most ores have an excess of SiO2 and

Al_2O_3 and the ash of the coke used for fuel is mainly composed of these oxides, so a basic flux, e.g., limestone, must be added (Hammond, 1999).

3.4 Mining of Iron Ores

3.4.1 Planning and Development

The general occurrence, size and shape of an iron ore deposit is determined during the exploration phase, which is discussed above. Knowledge of the deposit is determined in more detail through development work. It is often necessary during the development of a mine to determine, in considerable detail, the position and nature of geological structures which affect ore distribution and availability. Currently most deposits being exploited in the United States are of low grade materials called taconites containing about 30% iron, all of which must be beneficiated or concentrated to produce acceptable shipping products. In such ore deposits, the results of laboratory concentration tests of drill samples, supplemented by pilot plant tests and by experience with commercial plant results, are used to determine the economics of ore treatment (Prior, 1998).

After sufficient detailed information is obtained, various combinations of operating plans are studied using maps and sections prepared for this purpose. These show the size and shape of the ore body, ore compositions and laboratory test results. From these graphic representations, the quantities of ores and waste materials are determined by the application of volume weight factors. Computers are commonly used in the preparation of tonnage estimates and in the preparation of detailed mining plans. Through the use of these systems, comparative evaluations of various mining methods and plans may be made to determine the most favorable plan for each particular deposit and to schedule the mining of the deposit (Meyers, 1996).

3.4.2 Open Pit Mining

Electric power shovel in an open pit mine loading iron bearing material into a diesel powered truck for transfer to a processing plant. Inasmuch as open pit mining provides the lowest cost operation, it is employed wherever the ratio of overburden, either consolidated or unconsolidated, to ore does not exceed an economical limit. Nearly all the large iron ore mines in the world, with the exception of some in Europe, are worked by open pit methods (Lungen, 1996). Figure 3.1, figure 3.2 and figure 3.3 show examples of open pit mining on the Mesabi Range.

Figure 3.1 Electric power shovel in an open pit mine loading iron bearing material (Hartmur et al., 1999).

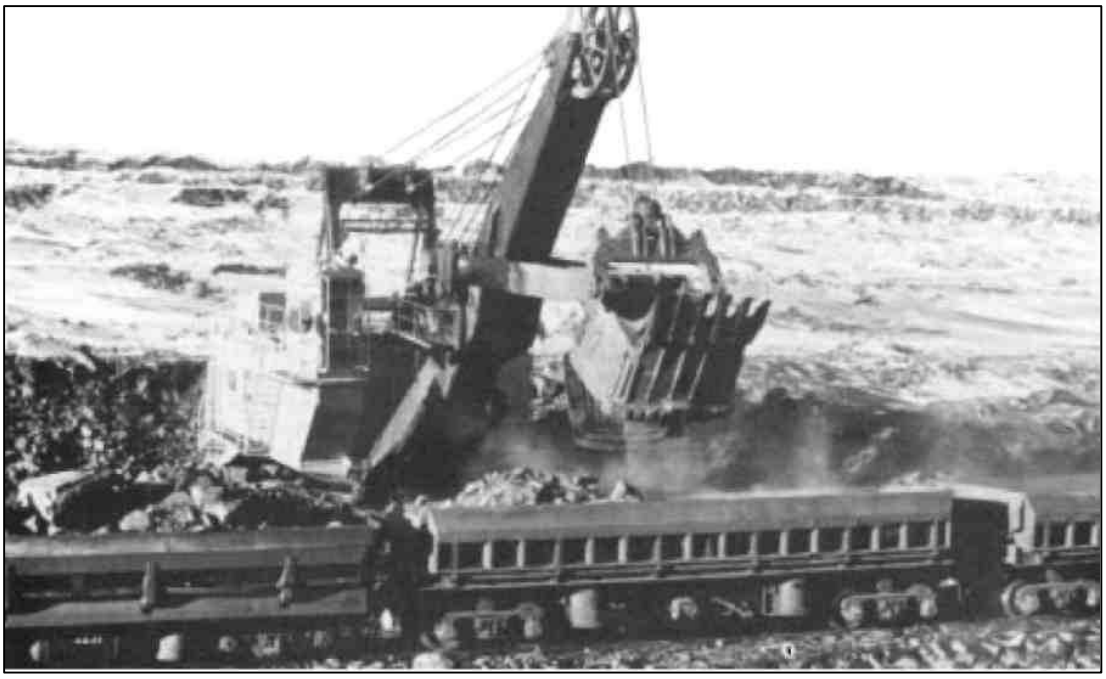


Figure 3.2 Electric power shovel loading material from an open pit iron ore mine for transfer to an ore processing plant (Hartmur et al., 1999).

Figure 3.3 Electric power shovel on caterpillar tracks, making a sinking cut and loading ore into steel dump cars for haulage from an open pit mine to a beneficiation plant. Diesel-electric locomotives handle the dump cars while a cable carries power to the shovel (Hartmur et al., 1999).

The depth to which open pit mining can be carried depends upon the grade of the ore, the nature of the overburden and the stripping ratio. The stripping ratio is the amount of overburden and waste that has to be handled for each unit of ore mined. The economic stripping ratio varies widely from mine to mine and district to district, depending upon a number of factors. In the case of direct shipping ores, it may be as high as 6:1 or 7:1; whereas, in the case of taconite, a stripping ratio of less than 1.5:1 may be the economic limit (Faria, 1997).

Overburden (stripping) may consist of unconsolidated material, rock, or lean ore material. In open pit mining, removal of overburden may continue through a large part of the life of a mine as the pit walls are cut back to permit deepening of the mine to recover ore in the bottom. Unconsolidated materials are excavated by power shovels, draglines or power scrapers, depending on local conditions. Other materials are generally excavated with power shovels (Bartholomew, Craig, & Metz, 1989).

Drilling and blasting is done to break consolidated materials into sizes capable of being handled by mining equipment and beneficiation facilities, and is also done to loosen ore banks ahead of power shovels to increase the efficiency of loading. Figure 3.4 shows that a typical picture of iron ore mining (Perry, 1994).

Taconite ore is loaded by power shovels equipped with buckets ranging in capacity up to 26.8 m^3 (35 yd^3). The ore is transported out of the pit by railroad cars, trucks, belt conveyors or combinations of these, to a crushing plant for size reduction and then to a beneficiation plant for fine grinding and concentration by magnetic, gravity or flotation techniques. The concentrates are usually agglomerated into to 9.5 mm (1/4–3/8 in.) balls and fired into hard durable pellets which will withstand handling, shipping and furnace charging (Feinman & MacRae, 1999).

The mining of taconite poses special problems because of its extreme hardness, which necessitates considerable additional drilling and blasting and more specialized and rugged equipment, as compared with the techniques and equipment used in mining most oxidized ores. Also, the relatively low iron content of taconite makes it necessary to handle two to four times as much mined material to obtain the same quantity of iron in ore as from higher grade ore deposits (Boyum, 1983).

Water causes a variety of problems in iron ore mining operations. Except in rare instances, such as in hilltop mining or mining under desert conditions, water must be collected in sumps, wells or underground workings and pumped out of the mine. Such drainage water is often utilized directly to make up for water losses in concentration operations. Figure 3.5 shows that a typical picture of ship loading.

Figure 3.4 A typical picture of iron ore mining (Iron Ore Manual, 1998-1999).

Figure 3.5 A typical picture of ship loading. Ships can carry about 170,000 tons of iron ore (Skillings Mining Review, 1996).

3.4.3 Underground Mining

When the stripping ratio becomes too high for economical open pit mining, underground mining methods may be employed. In most cases, access to underground mines is obtained through inclined or vertical shafts sunk adjacent to the deposit but far enough away to avoid the effects of surface subsidence resulting from mining operations. Some deposits are so situated that shafts can be driven directly into them from hillsides or from open pit banks (Benison, 1954).

Underground mining requires a larger capital investment per ton of annual capacity than open pit mining because it depends upon costly shafts or tunnels, underground haulage and development workings, and elaborate pumping facilities. Moreover, the production of iron ore per man per day in an underground mine is only a fraction of that in open pit operations, whereas the cost of supplies, maintenance, hoisting, and pumping, are all higher (Eggert, 1992)

In underground mining, several methods of ore extraction may be used. Among the most common, in order of increasing cost, are: block caving, sub level stopping, sub-level caving, top slicing, and modifications or combinations of these. All of these methods involve: drilling; blasting; transportation within the mine by rail tramming, trackless shuttle cars, scrapers, or conveyor belts; and hoisting or hauling to the surface. On the surface, the ore may be crushed, sized or concentrated prior to shipment. In general, the higher costs of underground mining has limited its use to ores requiring simple crushing or sizing, special ores such as open hearth lump ore, or low grade ores located so near a consuming market that transportation costs to the point of use were not significant. There is one iron ores operating underground mine in the United States, located at Pea Ridge, Missouri, where a magnetite ore body is being mined at depths of +600 m (+2000 ft). The ore is crushed, concentrated and had been used as a pellet feed. Examples of successful underground iron ore mines are the LKAB operations in Sweden where high quality, economy of scale and proximity to markets overcome the economic challenges of underground mining (Vialls, 1980).

3.5 Blending of Iron Ores

Furnace practices, as described elsewhere in this volume, require the production of natural ores and/or concentrates to meet the physical and chemical specifications required in ironmaking processes. Prior to the shift in the last two decades to the large scale production of iron ore concentrates and pellets, the natural ores were graded by the producers and shippers to meet furnace demands for particular and uniform chemical composition and structure. Iron ore merchant companies could satisfy grade requirements by purchase or exchange of ores, which were then mixed together in the correct proportions. Recognition of the importance of uniformity led to the use of elaborate ore blending facilities at some producing and consuming points, involving systematic layering in stockpiles and recovery for shipment or consumption by cross cutting the layers, see figure 3.5. The blending, stacking and reclaiming process illustrated in figure 3.6 is prominently used to prepare a uniform feed blend for modern sinter plant operations (MacLean, 1981).

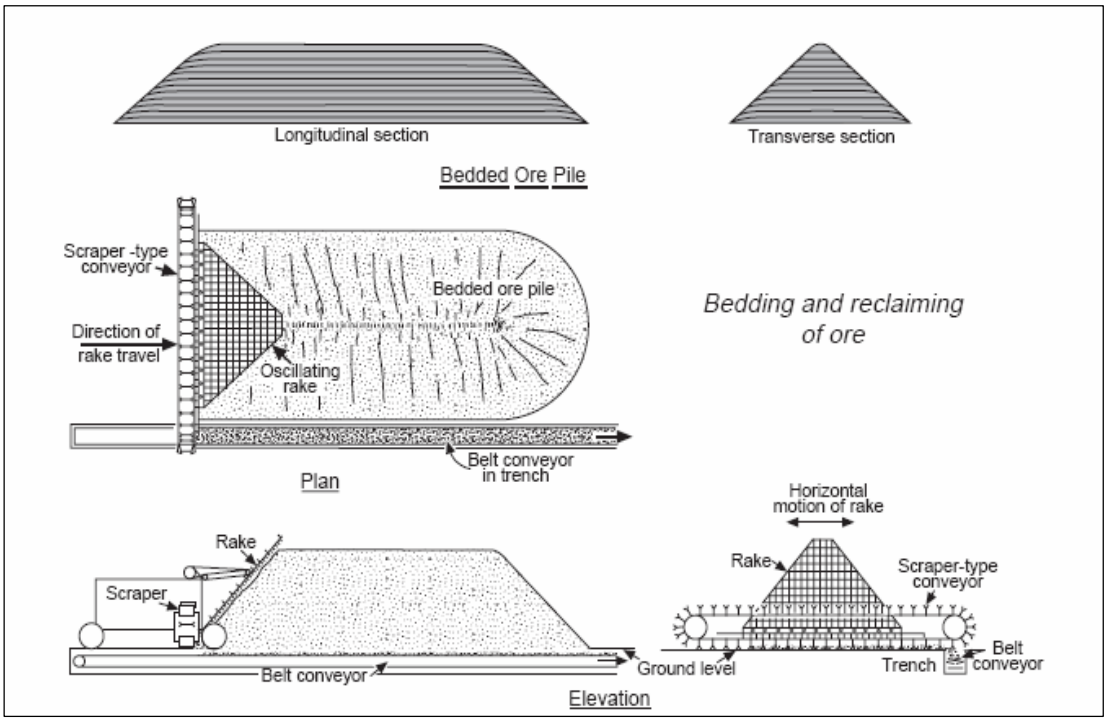


Figure 3.6 Schematic representation of the principle of operation of a stacking and reclaiming system (Hartmur et al., 1999).

On a worldwide basis, many iron ore districts continue to produce natural ores which are usually mined to supply a uniform quality of ore. Some producers ship natural ores along with sized products, concentrates and pellets while others ship only concentrates and/or pellets. In general, it has been demonstrated that the added cost of utilizing a uniform high grade iron ore product as furnace feed can result in significant economies in the cost of the resulting hot metal (Lewis, 1976).

Of particular significance in grading of ores, aside from the iron content, is the content of silica, phosphorus, manganese and alumina. A high lime content makes some ores self fluxing. Sulfur, copper, nickel, titanium, sodium, potassium and other deleterious constituents may require close control in some producing areas. applied for varying chemical and structural quality. Practices vary considerably in the various world iron ore markets (Mousset, 1994).

CHAPTER FOUR
BENEFICIATION OF IRON ORES

4.1 Beneficiation Process

The term beneficiation in regard to iron ores encompasses all of the methods used to process ore to improve its chemical, physical or metallurgical characteristics in ways that will make it a more desirable feed for the ironmaking furnace. Such methods include crushing, screening, blending, concentrating and agglomerating.

Most iron ores fall within one of the three following categories. (1) Direct shipping, or high grade merchant ores, which contain enough iron to be charged to the blast furnace directly and may only require crushing, screening, and blending. (2) Associated low grade merchant ores which occur around the high grade ores that can be mined simultaneously and which require minor upgrading by washing, or gravity separation techniques to increase their iron content. (3) The underlying iron formations, or taconite, from which most of these deposits have been derived; a hard, dense, low grade material that requires extensive crushing, grinding and concentration to produce an acceptable concentrate (Boero, 1996).

The iron ores that fall within these three categories have quite different processing requirements and discussion of the appropriate beneficiation steps have been grouped accordingly. The following brief and generalized descriptions are intended only to describe how the basic types of ore are beneficiated and are not to be interpreted as suitable for all ores.

4.1.1 Crushing and Screening

Iron ore of merchantable grade must be properly sized prior to charging to the blast furnace. Present blast furnace technology commonly requires crushing and screening of direct charge lump ore finer than 6 mm and coarser than 30 mm.

The specific size selected is based on the characteristics of the ore and is specified so as to maintain high stack permeability and also allow sufficient time for reduction of coarser material. Consequently, crushing and screening are an integral part of ore producing facilities.

Crushing commonly involves a primary jaw crusher with secondary gyratory, or cone crushers operating in closed circuit with vibrating screens. Roll crushers, hammer mills, or vertical impact crushers have been used. Equipment selection is determined largely by the friability of the ore. Most of the screening operations on high grade ores are dry except when the fines fraction can be effectively upgraded by de-sliming (Boyd & Ferron, 1997).

The –6 mm (1/4 inc.) fines produced by crushing and screening are most commonly agglomerated by sintering, or sometimes reground and pelletized. The sinter produced is also crushed and screened to meet size specifications compatible with the other charge components.

4.1.2 Low Grade Merchant Ores

The earliest iron ore beneficiation techniques were developed to upgrade the lower grade ores associated with direct shipping ores. The natural ore forming process produced layers of relatively pure iron oxides interblended with partially decomposed silica rich layers. Ores in which the silica layers had been completely decomposed could be easily upgraded by simple washing techniques where the fine-grained friable silica particles could be separated from the heavier, denser, and coarse iron ore particles by hydraulic classification. Ores in which the silica rich layers had not been weathered as intensely and were still relatively cohesive and had to be broken by crushing were upgraded by gravity concentration techniques such as jigging and heavy media separation with the finer fractions upgraded in spiral concentrators. The technology for beneficiating low grade ores was largely developed on the Mesabi Range in Minnesota in the 1950s and 1960s and applied world wide to similar deposits (Portillo, 1996).

4.1.3 Washing

Washing is the simplest iron ore concentration process that takes advantage of the high specific gravity and comparatively coarse size of the iron bearing minerals to separate them from the finer, lighter, siliceous gangue which is predominantly quartz and clay minerals. The ore is prepared for washing by crushing in one or two stages finer than 50 mm (2 in.). The intense agitation of the ore by the paddles (similar to a modern pug mill) combined with the counter flowing water efficiently commuted and removed the fine silica to leave a coarse residual iron rich product. Some washing plants employed spiral classifiers in one or two stages without a log washer on ores containing a minimum amount of sticky clay gangue (Kruger, 1997).

Hindered settling classifiers of various types, hydrosizers, pocket seizers and Wilfley shaking tables were also used to recover fine iron before the development of the Humphrey spiral.

4.1.4 Jigging

Jigging is a more complex form of beneficiation than simple washing and is used on the more refractory ores that require crushing to break up the silica rich layers. Jigs used for iron ore beneficiation are basically horizontal screens which carry a bed of ore some 15–25 cm (6–10 in.) deep. The ore is fed at one end and is stratified by the pulsing action of water, either caused by an oscillating pump or by physical up and down movement of the jig screen itself. As the ore moves down the deck the pulsing allows the lighter silica rich particles to work their way to the top of the bed, while the heavier iron rich particles segregate along the base. The two products are separated at the end of the jig, the lighter silica-rich particles over the top of the discharge weir and the iron ore concentrate under the bottom. Iron ore jigs worked best on the particles ranging from 25 mm to 1 mm (1 in. to 1/16 in.) (Kirk, 1991).

4.1.5 Spirals

The Humphrey spiral, first developed for the treatment of beach sands, is used in iron ore concentration to treat -6 mesh X 100 mesh ore. Efficiency below 100 mesh decreases rapidly and spirals are ineffectual on finer materials. Spirals are widely used for the supplementary recovery of fine iron from merchant ore types and are the primary concentration device for the specular hematite ores and similar ores that can be liberated by grinding no finer than 20 mesh. A typical concentration plant flowsheet for specular hematite ore is shown later in figure 4.2.

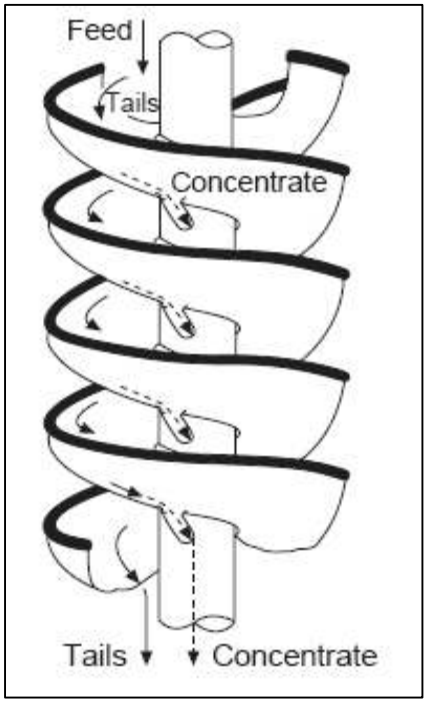


Figure 4.1 Schematic of a spiral concentrator (Antunes, 1998).

The spiral concentrator is a curved-bottom trough, wound around a vertical axis in the form of a helix, (seen in Figure 4.1). When fed at the top with slurry of iron ore and gangue, the less dense gangue, being more readily suspended by the water, attains greater tangential velocity than the iron minerals, and migrates toward the outer rim of the spiral trough. Wash water added along the inside rim helps wash away the lighter gangue. After a few turns, a band of iron mineral forms along the inner rim, and the gangue forms bands toward the outer rim. Ports are spaced along the inner rim to collect and remove the iron minerals. The gangue remains in the spiral and discharges at the bottom (Bolis & Bekkala, 1987).

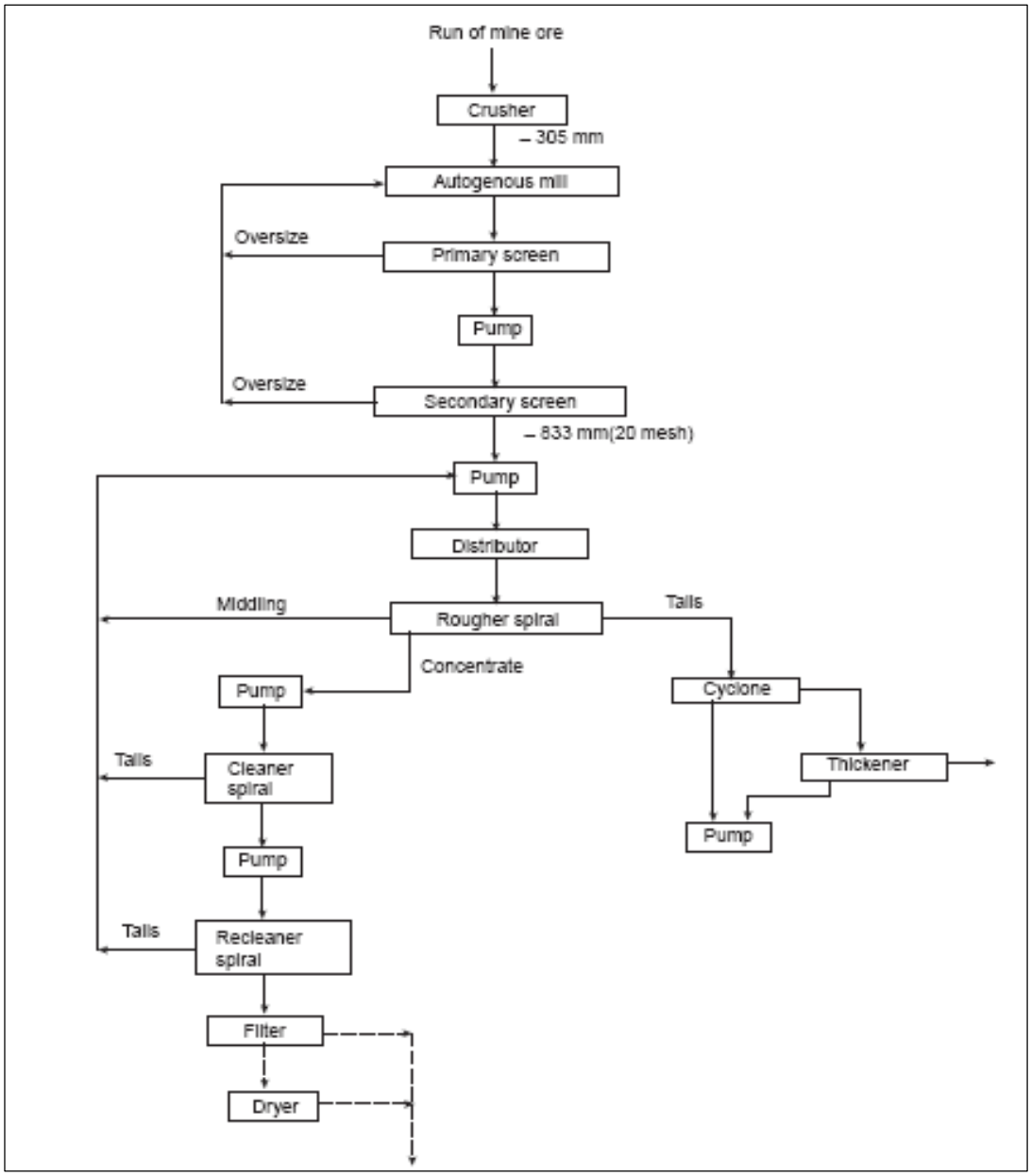


Figure 4.2 Flowsheet for spiral concentration at Quebec Carter Mining Company (Kirk, 1996).

4.1.6 Reichert Cone

The principle advantages for the Reichert cone are capacity and the ability to recover fine heavy minerals efficiently down to about 325 meshes, finer than is attainable in spirals. A single Reichert cone has a capacity of up to 100 tonnes per hour and can be used effectively to recover specular hematite fines as well as merchant ore fines. De-sliming of the feed is desirable, and essential for merchant type ore fines (Antunes, 1998).

The Reichert cone, figure 4.3, is a flowing film concentrator. The denser particles concentrate at the bottom of a flowing film of slurry having a solids content of about 60% by weight. The separation mechanism is a combination of hindered settling of the dense particles and interstitial trickling of the fine particles. The separation element in the Reichert unit is an inward sloping 1.9 m (6.25 ft) diameter cone. Feed pulp is evenly distributed around the periphery of the cone (Chavarri, 1997).

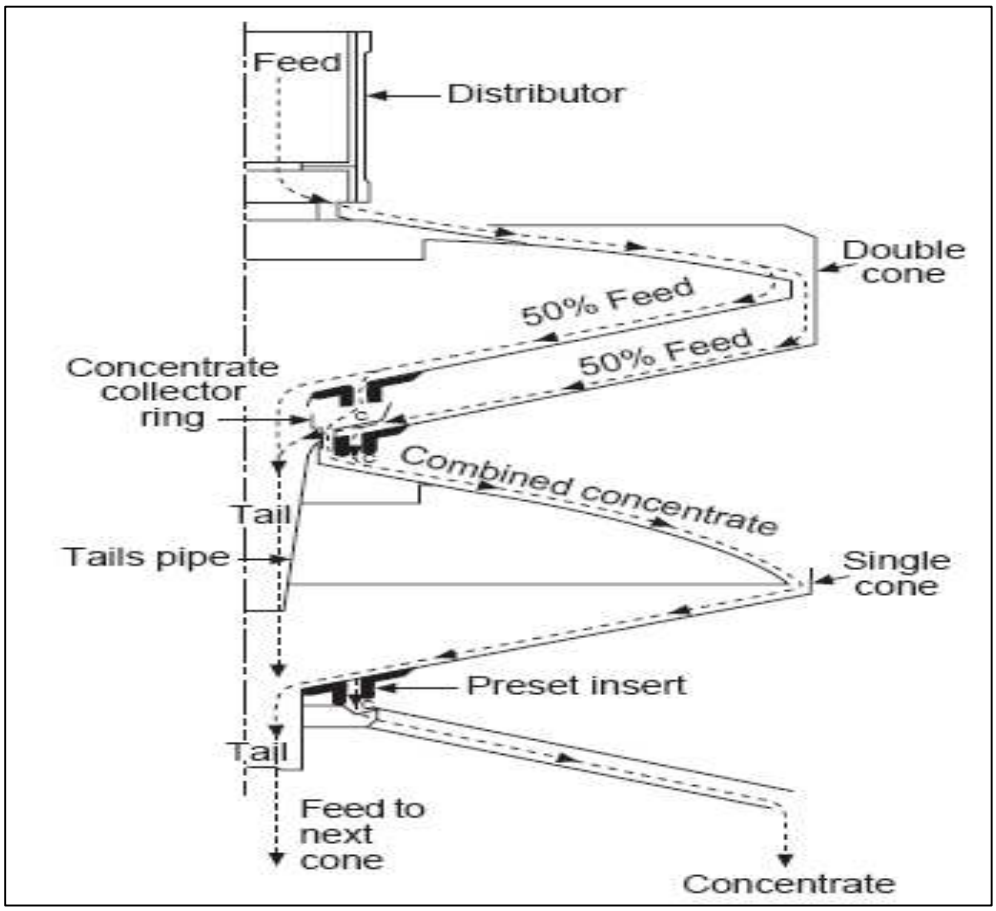


Figure 4.3 Schematic of a section of a Reichert cone concentrator (Bailey, 1992).

As the pulp flows by gravity toward the center, the fine and the heavy particles concentrate on the bottom and are removed through an annular slot near the apex of the cone. The tailing flows over the slot and is collected at the apex or center of the cone. Because the efficiency of this separation process is relatively low, it is repeated several times within a single stacked arrangement of cones to increase the recovery. Generally, the highest grade concentrate is produced in the primary separation cone.

4.1.7 Wet High Intensity Magnetic Separation (WHIMS)

Wet high intensity magnetic separation machines, WHIMS, were developed to recover non-magnetic iron units, They can be effectively applied across a wide particle size range from 10 mesh to 500 mesh depending upon the matrix used. WHIMS applications include recovery of iron from natural ore fines, upgrading of spiral concentrates for direct reduction feed, and recovery of hematite from tailings.

In wet high intensity magnetic separations (WHIMS), electromagnets produce a very high strength magnetic field that is applied to a matrix consisting of steel balls, spaced grooved plates, steel wool or pieces of expanded metal, figure 4.4. The matrix is contained in an annular ring that is rotated between the high intensity magnets. The iron ore slurry is introduced at a point where the matrix is in the field. The high magnetic gradients developed around the matrix hold the hematite while the siliceous gangue is washed through. The hematite concentrate is released and discharged as the matrix moves out of the magnetic field (Skillings, 1996).

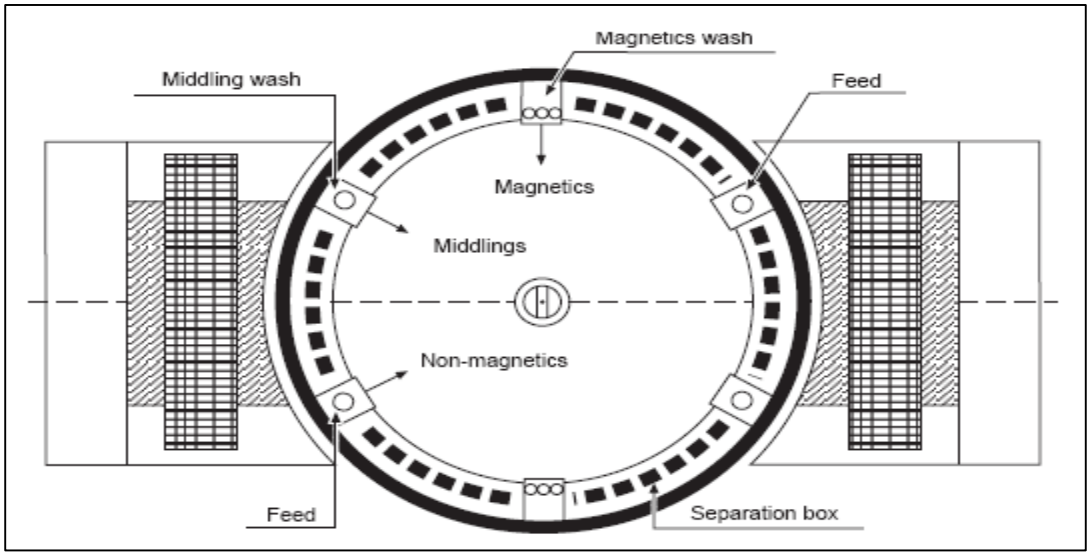


Figure 4.4 Schematic diagram of a wet high intensity magnetic separator (Iron Ore Statistic, 1997).

4.1.8 Flotation

Froth flotation is effective for the concentration of fine, –100 mesh, iron ores. It can be used for the supplementary upgrading of magnetite concentrates, as currently practiced at four taconite operations in Minnesota, or as the primary recovery method for hematite ores, as currently practiced at the Tilden mine in Michigan.

Flotation processes depend on the fact that certain reagents added to water suspensions of finely ground iron ore selectively cause either iron oxide minerals or gangue particles to exhibit an affinity for air. The minerals having this affinity attach to air bubbles passing through the suspension and are removed from the suspension as a froth product.

The reagents added to induce the preferential affinity for air are commonly called collectors or promoters; substances added to cause stable bubble or froth formation, are known as frothers; other substances added for control purposes such as pH adjustment, or to cause better dispersion or flocculation are known as modifiers, dispersants, and depressants (Prior, 1998).

The main application of anionic flotation is to float iron bearing minerals away from gangue material. The most common collectors used are fatty acids or petroleum sulfonates. Fuel oil is often added along with the collectors to promote recovery of iron oxide particles finer than about 10 μm.

Hematite Ores: The basic selective flocculation, cationic silica flotation flowsheet was developed specifically to concentrate the ores of the Tilden Mine. Refer to the plant flowsheet shown as figure 4.5. Fine grinding of this somewhat soft ore produces a significant percentage of hematite slimes. These are flocculated by adding starch products to the slurry. This allows rejection of siliceous slimes by classification and retention of the flocculated fine iron minerals. The de-slimed ore is then concentrated by froth flotation using cationic collectors to remove additional silica. The separation is enhanced because the starch depresses (inhibits) flotation of the flocculated iron minerals (Hammond, 1999).

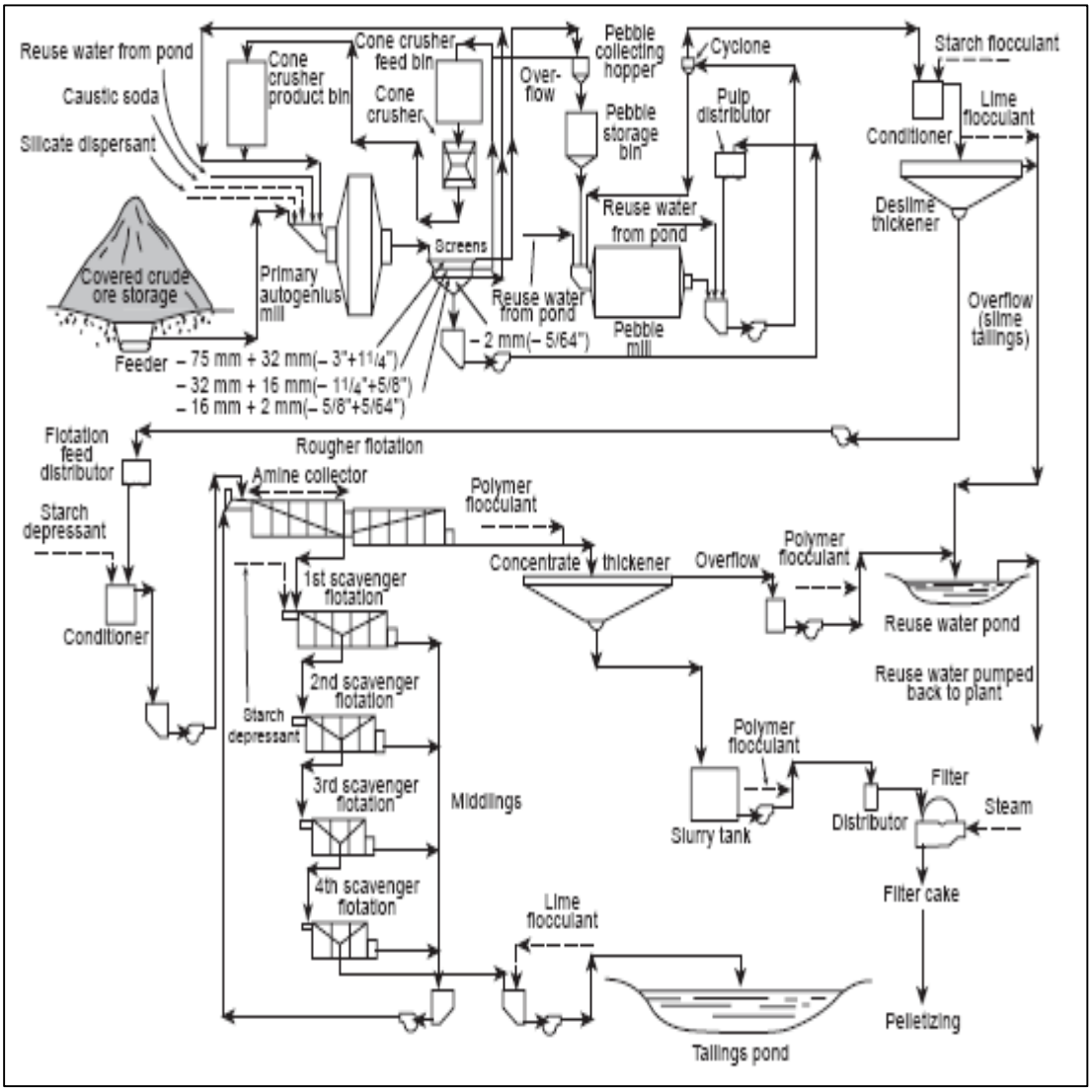


Figure 4.5 Concentrator flowsheet of the Tilden Mine (Skillings Mining Review, 1996).

CHAPTER FIVE
INFORMATION ABOUT SINTERING TECHNOLOGY

5.1 Sintering

Sintering has been referred to as the art of burning a fuel mixed with ore under controlled conditions. The flexibility of the process permits conversion of a variety of materials, including natural fine ores and ore fines from screening operations, flue dust, ore concentrates, and other iron bearing materials of small particle size into a clinker like aggregate that is well suited for use in the blast furnace.

The continuous sintering process shown schematically in figure 5.1 is carried out on a traveling grate that conveys a bed of ore fines or other finely divided iron bearing material, intimately mixed with approximately 5% of a finely divided fuel such as coke breeze or anthracite. Near the head or feed end of the grate, the bed is ignited on the surface by gas burners. Sintering machine in operation shown of a diagram in figure 5.2. The ignition furnace in the center background ignites the fuel in the surface layer of the sinter mix. As the bed of sinter progresses into the foreground, air is pulled down through the bed to cause the burning zone to move downward through the bed by igniting fuel in deeper and deeper zones of the mix. By the time the discharge end of the machine is reached, sintering will have taken place throughout the depth of the bed (Meyer, 1980).

In figure 5.2, the mixture moves along on the traveling grate, air is pulled down through the mixture to burn the fuel by downdraft combustion. As the grates (or pallets) move continuously over the windboxes toward the discharge end of the strand, the combustion front in the bed moves progressively downward. This creates sufficient heat and temperature, about 1300-1480 °C (2370-2700 °F), to sinter the fine ore particles together into porous clinkers. That location along the traveling grate where the combustion front touches the bottom of the bed is called the burn through point (Turkdogan, 1996).

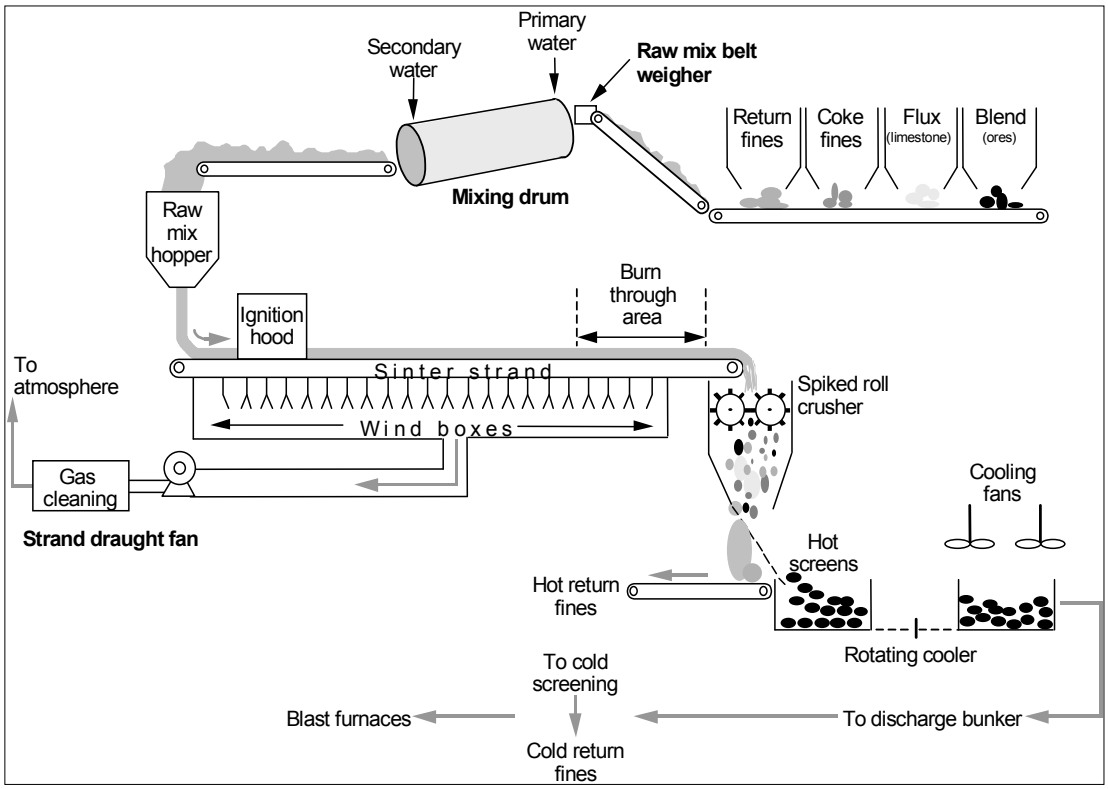


Figure 5.1 Schematic flow diagram of continuous iron ore sintering process (Kasai & Komarov, 2005)

Figure 5.2 Photograph of a sinter strand with the charging facility (drums or chutes) and the ignition canopy at the starting end (Integrated Pollution Prevention and Control [IPPC], 2001).

Although simple in principle, sintering plants require that a number of important factors in their design and operation be observed to attain optimum performance.

Intimate mixing of the feed materials is one of the most important. In most modern sinter plants, bedding and blending operation, as shown in figure 5.3, is used to pre-mix the sintering ores, steel plant waste oxides, fluxes, and some solid fuels. This blended feed is then supplemented by small trim amounts of flux and solid fuel. This total feed mixture is the subject to a water addition within a mixing device such as a balling drum or disc, in figure 5.3, or pug mill. These mixers are operated to produce small rice size nodules that significantly improve the permeability of the sinter bed. Improved permeability, in turn, results in more rapid and uniform sintering. Desirable mixer retention times vary from about one minute for sticky hematite ores to four minutes for more difficult to ball ores. Figure 5.4 shows the typical material routes in the ore preparation and sintering plant, which are continuously supervised

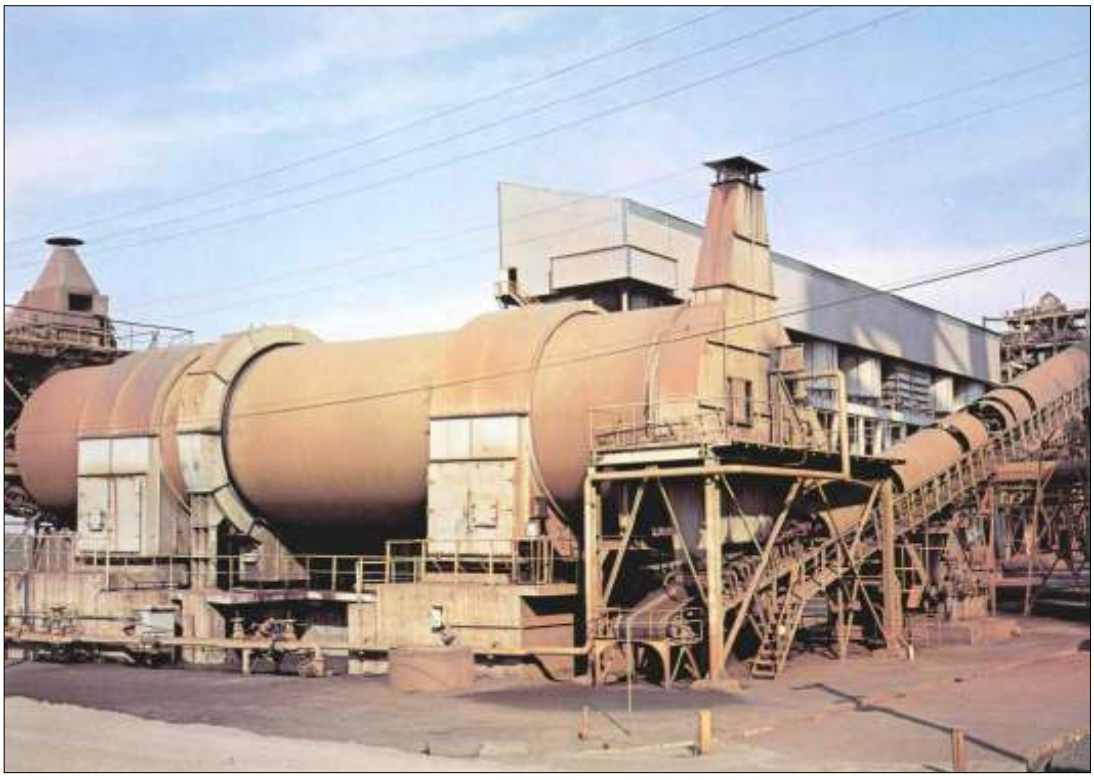


Figure 5.3 Two disc-type pelletizing machines. Ore fines are fed to the discs by the belt conveyors (behind the operator). As the discs rotate, there is a balling action that causes the fines to agglomerate into pellet-like masses that are discharged from the discs over their lip at the bottom onto to the belt conveyor shown at floor level that carries the pellets to the bins of the sinter machine (Iron Making Volume, n.d.).

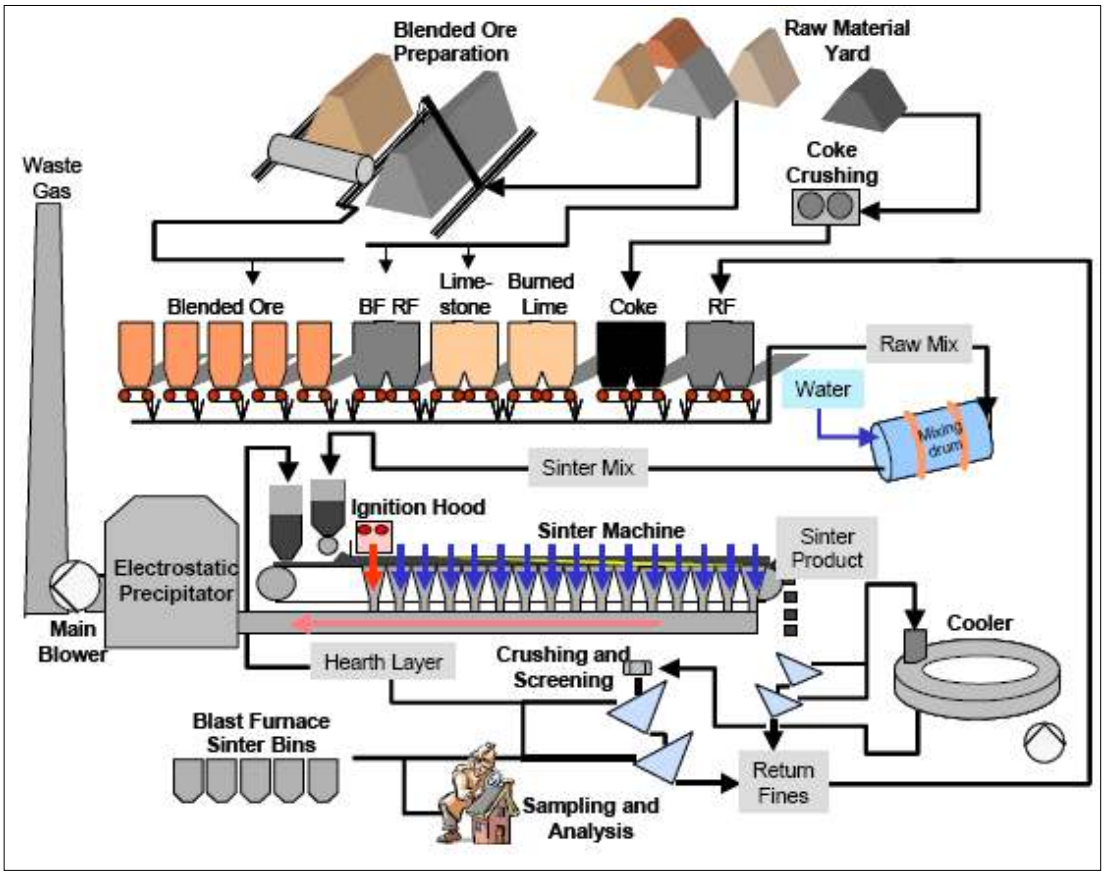


Figure 5.4 Material Flows in Sintering (Voest Alpine, 2008).

In transferring the prepared mix from the mixer to the grate of the sintering machine it is essential to feed the material carefully to provide a uniform, homogeneous bed and to prevent compacting of the bed. To avoid a direct drop of feed onto the grate, a hearth layer of about 25–50 mm of already sintered material is fed first onto the traveling grate. Feeding devices typically include a roll feeder in conjunction with chutes which act to avoid compacting the feed material. Design of surge bins and feeders for distributing the prepared mix into these bins is equally important because, if the prepared mix is compacted or segregated during handling and loading onto the grate, all of the advantages gained through good feed preparation may be lost. Once the feed is charged onto the grate, metal bars or rods already inserted longitudinally along the grate for a distance of about 2–4 m (6–13 ft) help to loosen up the mixture to enhance permeability (Sasaki, Ishii, & Wan, 2001).

Proper ignition of the sinter bed is also important. Poor ignition results in spotty burning and may leave unsintered material over the surface of the bed. Conversely, too intense an ignition flame can result in sagging over the bed and reduced sintering rates.

The radiant hood ignition furnace provides good ignition. Replacing part of the solid fuel with gaseous fuel results in sinter having a slightly improved strength and reducibility without affecting sinter production rate. This practice is termed mixed firing. Where a shortage of solid fuel exists, and gas is available, use of increased amounts of gaseous fuel should be desirable. Plants using increased ignition (extended firing) have approximately 25% of the length of the sinter bed covered by a gas fired ignition type hood. The temperature in this hood ranges from about 1150 °C (2100 °F) in the first section where ignition begins to approximately 800 °C (1500 °F) at the exit end of the hood. Depending upon the characteristics of the ore materials and the sintering conditions, daily average production rate of 22.4-42.9 tonnes/m^2/day (2.3-4.4 net tonnes/ft^2/day) of grate area are expected, and individual daily rates in excess of 48.8 tonnes/m^2/day (5 net tons/ft^2/day) have been attained (Sasaki, Ishii, & Wan, 2001).

Cooling of the sinter below 150 °C (300 °F) so that it can be handled on conveyor belts is an important part of the operation. Sinter coolers, such as the rotary type, figure 5.5, are usually used; it is desirable to avoid a water quench as the quench adversely affects sinter properties. The exhaust air from these coolers is normally at too low a temperature to permit the economical recovery of heat, although such systems have been installed in countries with high energy costs as in Japan (Kimura, 2005).

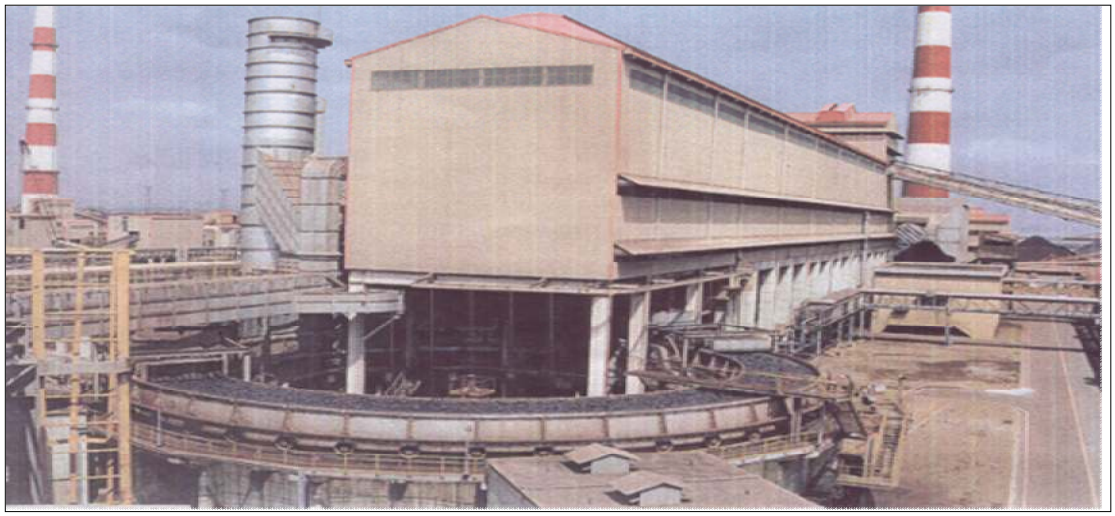


Figure 5.5 Induced draft rotary or circular sinter coolers, shown in process of construction (Poveromo, 1999).

Sinter represents an improved blast furnace material as compared to lump ore or acid pellets as follows. Improvements have been obtained by incorporating the blast furnace flux into the sinter rather than charging it separately to the top of the furnace, as was formerly done, by use of sized sinter and by virtue of improved high temperature properties. The available data on the use of fluxed sinter, sometimes called self-fluxing sinter, indicate that for each 200 lbs of limestone per net ton of hot metal removed from the blast furnace burden and charged into the sinter plant to make a fluxed sinter, approximately 9.1 kg (20 lb) of metallurgical coke per net ton of hot metal are saved. The coke savings results primarily from calcining of limestone on the sintering grate rather than in the blast furnace. Limestone in the form of fluxing fines for the production of sinter is made by crushing and screening methods that result in a product meeting size specifications (Sasaki, Ishii & Wan, 2001).

Use of sized sinter is desirable because iron production rates in the blast furnace are further increased. Plant tests have demonstrated significant increases in iron production rate as a result of screening out small-sized material in sinter before it is charged to the furnace. Other tests have shown that sized sinter, which contains 85- 90 % of 25 mm x 6 mm (1 in. X l/4 in.) material as compared with 60% in unsized sinter has a much higher permeability than unsized sinter and performs as well as pellets of comparable size. It also appears that crushing to (25 mm (1 in.) size at the sinter plant yields a more stable sinter because the smaller size fractions are more resistant to degradation.

In the last several decades researchers have shown that fluxed sinter (and also fluxed pellets) has superior high temperature properties in the blast furnace as compared to lump ore and acid pellets. These improvements include higher softening and melting temperatures and higher levels of reducibility.

The above description of sintering only provides an introduction to this topic. The earlier (before 1980) references are useful for North American and former Eastern bloc sinter plant operations, most of which were built in the 1950s and 1960s. Some more recent references describe advances in sintering relevant to large, modern sinter plant operation which are predominant in Europe and the Asia Pacific area.

In North America and in the Scandinavian countries it has been demonstrated that fluxed pellets; fluxed with either limestone/dolomite mixtures or olivine, and also properly sized, can achieve blast furnace performances comparable to that of fluxed, sized sinter (Sasaki, Ishii & Wan, 2001).

5.2 Process Description

Sintering is the most common agglomeration process. It is achieved by the application of heat which results in the conversion of ore fines into large, hard, porous lumps. In sintering, a shallow bed of fine particles is agglomerated by heat exchange and partial fusion of quiescent mass. Heat is generated by combustion of a solid fuel (coke) admixed with the bed of fines being agglomerated. The combustion is initiated by igniting the fuel exposed at the surface of the bed, after which a narrow, high temparature zone is caused to move through the bed by an induced draft applied at the bottom of the bed.The bonding is affected by a combination of fusion, grain growth and slag liquidation. The generation of volatiles from the fuel and flux stone creates a frothy condition and the incoming air quenches and solidify the rear edge of the advancing fusion zone. The product consists of a cellular mass of ore bonded in a slag matrix (Meyer, 1980).

Decide the application to make complete the iron ore concentration by pelleting or sintering, additionally decide the particle size of concentration. If ore can be very tiny to enrich, namely toughening become by very tiny matter, there are pelletizing for the ore. Suitable size of particle for pellettizing and sintering shown below:

Sintering: - 100 μm. should be 20% max. Max particle size shouldn't be more from 8 mm. Grade of Fe is between 58 and 62%.

Peletting: - 44 μm should be min. 70% and 90 μm should be max .15%.

Grade of Fe: It needs above the 62 %.

Lump iron ore must be specific size to send to blast furnace. So that the big particles break down and also during this application there is powdering as a specific ratio. As well as during the transportation, charging and discharging there is a tiny fine ore which

needs to sintering is occurs. As a result of this most blast furnace have own sinter plant (Larrca, 1992).

5.3 Reason of Sintering

Materials that making iron ore + coke + cinder goes down, and hot gases goes up in the blast furnace which is product a metallic iron form iron ore. The best attachment between the solids and reduced gases is provided by a bed which has minimum channelling and ordinary flow of gases which has the highest permeability. If iron ore that the used for charging to blast furnace is smaller than the specific size (usually -8 mm), then it is decreases the permeability of blast furnace and increases the gas channelling, so that it don't charge directly to furnace.

Take the elements, which have harmful amounts, away from the ore chemical configuration during the sintering. Discharging of the sulphur, arsenic, and humidity that has present in the ore. Some elements such as sulphur and arsenic should not being in steel structure as over the define ratio. If this situation is not provided, so that the steel become more brittle. Because of that these harmful materials evaporate as far as possible during the sintering (Egundebi, 1989)

Using raw ore without sintering decreases the capacity of blast furnace. So blend materials becoming half products to prevent these situations.

Charging of sintering products to blast furnace increases the raw iron capacity of blast furnace, also consumption of coke, cinder which is occur, amounts of flue dust decreases.

However the iron smelting plant accumulate 0-30 mm coke dust without appreciate during the production in blast furnace and during the transfer. The dust is 30.000- 40.000 tones in Eregli Iron and Steel Plants. Sintering is providing appreciate the coke dust.

The characteristics of sinter of paramount importance in that to a high degree they determine the performance of the blast furnace. The objects of sintering are;

- To increase the particle size,
- To form a strong reducible agglomerate,
- To remove volatiles and sulphur,
- To incorporate flux into the blast furnace burden.

The first sinter plants were designed and operated with virtually the sole object of increasing the particle size, and the blast furnace was then burdened with an appropriate amount of flux. With the realization that the use of sinter vastly improved furnace performance, the physical properties and chemical constitution came to be examined much more closely.

5.4 Sintering Steps

5.4.1 Preparing Sinter Blend

The aim for preparing sinter blend is make a suitable of iron ore to blast furnace. Although some elements and components which is present in ore, put out from the context provided by matters that put into the ore during the sintering. These matters look like limestone and dolomite. Also the sinter dust which is come from the sinter structures is added. The sufficient amount of solid fuel should mix to blend to become sintering. The sinter product become so quality if we know the properties of matters which is feed to blend and pay attention on these conditions (Ball, 1973).

5.4.2 Ore Properties

Ore appreciate if iron grade of ore is increase. Because the amounts of mixing which is feed the blast furnace to product one tone pig are decreases by increasing of grade. This means that lower cost and fuel against to higher production. The important things are the ratio of acid and base which are in ore.

Iron ore has S, and As elements in its structures. These elements oxidise at sintering label and leave from sinter. K_2O and Na_2O stayed in sinter. These components evaporate

on blast furnace at 900-1000 ^{o}C high temperature, and come down when the tall boy of furnace temperature is 700 ^{o}C.

These operations repeated continuously so that there is a consumption of Na_2O and K_2O components regularly. This is because that obtaining potassium oxide crown at the top label of blast furnace. Sodium oxide and potassium oxide are called alcali (Cohen, 1953).

5.4.3 Dunite or Dolomite

The aim of adding dolomite to sinter blend is reach the acid base equilibrium in blend. First level is equilibrium of MgO and Al_2O_3 at acid base equilibrium. If using ore have high amounts of clay, so we should use dolomite to reach the equilibrium. MgO/Al_2O_3 should be equal to 1. Dolomite should be estimate to complete the lack of MgO, and then add to the blend (Ball, 1973).

5.4.4 Limestone

Limestone that is used for sinter is a half crystal structure. It has 52-55% CaO, 0-2% MgO, 1-1.5% SiO_2 and 1-2% humidity in its chemical structure. It has high amounts of CaO, as a result of this it is add to the sinter blend to reach the CaO/SiO_2 equilibrium. This ratio should be 1.22. And it is called as low basic ratio. Also CaO+ MgO+ CuO / $SiO_2 + Al_2O_3$ ratio called as high basic ratio. Both of them can call as basic ratio.

If low basic ratio is bigger than the 0.90, so S which is occurs during the sintering, it forms CaS without combustion. The substance will have permanent property. But it is start to melting at 1570 ^{o}C. So that it is invalid substance for sinter. As a result, we have to estimate the ratio of basic and combusting of sulphur (Strassburger, 1969).

5.4.5 Coke Dust

Coke dust is used as solid fuel in sintering. It is come from the coke plant which is also in Isdemir. It has 12-17% humidity. And also it obtains 14-20 % ash as dry. The ash has SiO_2 50 %, CaO 5 %, Fe 5 %, Al_2O_3 7-8 %. If it is a good coking it has not any volatile substance such as benzene and phenol. The amount of the coke is 4% to add the blend. Coke has also a physical effect on sintering. If coke is big, there will not be good

combustion and so sinter will not be good. The optimum size should be 0-3 mm of coke (Strassburger, 1969).

5.4.6 Sinter Dust

Sinter dust obtains from the hot and cold underscreen materials, electro filter dust, chain conveyor's dust, and also the dust which are taken from the blast furnace and from the underscreen that place under the bunker. The sinter dust has the all physical and chemical properties of sinter. Sintering provided that these sinter dust made bigger particles, ingot (Voest Alpine Industrieanlagenbau [VAI], 2006).

The amount of sinter dust that is in sinter blend and its qualities has a big effect to sintering results. Because of that sinter dust should be cold to 80 ^{o}C and its humidity ratio should not pass over the 5 % (Strassburger, 1969).

5.5 Sinter Stage

The necessity additive such as CaO+MgO and coke dust added into the iron ore dust for sintering. And after that mixed put into the trommel index and concrete type mixer. The solid components mixed as well and also the dust becomes floccules (pelletizing). Because of that humidity that has define ratio (8%) is add into the mixer (Wild, 1953).

Raw materials required blending prior to the sintering operation, initially; iron ore is crushed to a suitable size is put into the blending by a stacker. Some flux material, coke and return dust can also be added at dosage unit. In addition, collected dust and mill scale, flue dust and sludge from gas cleaning in steel making may be added to the ore blend at the mixing stage. At the start of the sintering operation the ore blend is transferred from the beds to storage bunkers at the start of the sinter plant. The ore blend and the coke breeze are weighed on conveyer belts and loaded into a mixing drum (given in figure 5.6). Here, they are blended completely and the mixture is moistening to enhance the lumpy, which improve the permeability of the sinter bed (Portillo, 1996).

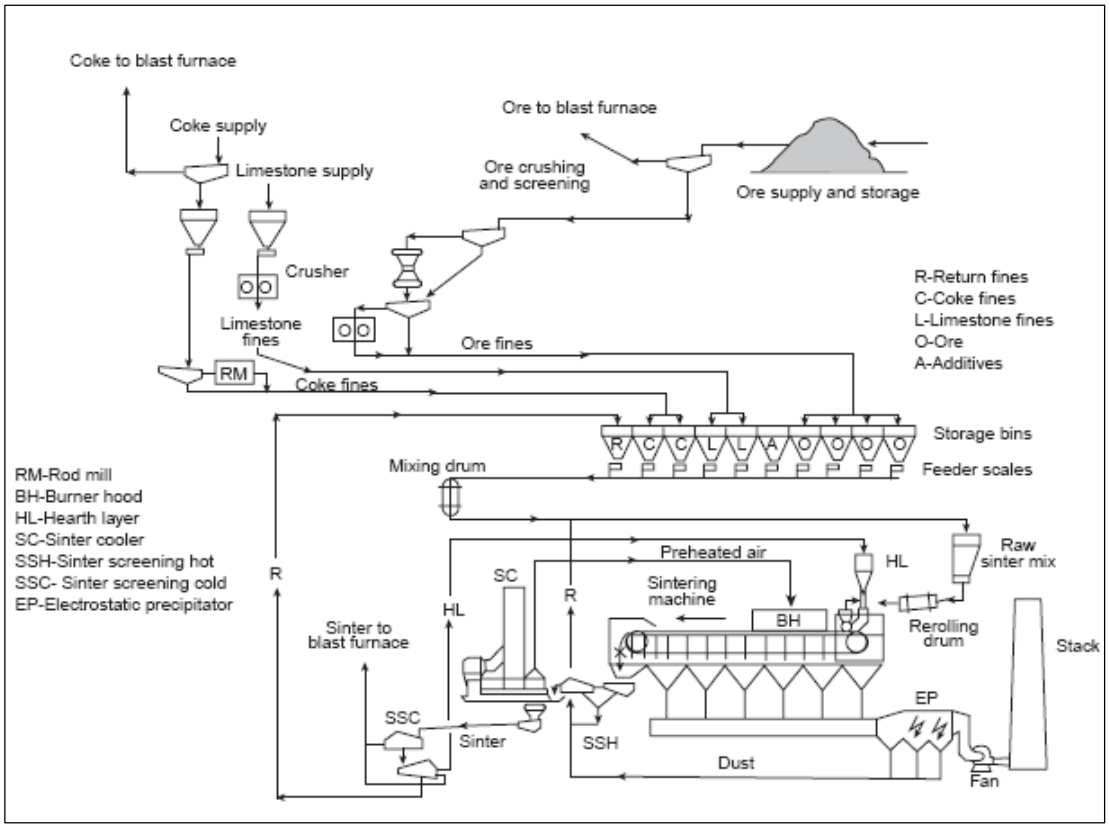


Figure 5.6 Sinter Operation Flow Diagram (Oone, Kawaguchi, Hoshi, Hadano & Umezak, 1998).

The pelletizing materials are going to blast furnace by conveyor. The furnace usually has rotary grate and its type is band furnace. Firstly 10-20 mm thickness fine sinter sends to the grates. So that is providing that closed of the grates blanks and also it saved grates from the high temperature.

Coke which is in mixed become firing and starts to combust when the loading materials pass under the firing dazzle. The grates move ahead, at that time the air sends on grates, this cause that combustion is passing through to the base of the layer, (given in figure 5.7). The temperature rises up to the 1300-1400 ^{o}C after the combustion. This high temperature provoke that the breaking down of ores and additives structures, and consequently the sinter bond obtain between the particles. These broken parts of materials are sulphur which is invalid materials for blast furnace. Otherwise after the leaves of invalid materials from the sinter cause that increasing of the sinter porosity, and also reaction ability of ores is increase. The sinter application completes towards rotary grate furnace endings, (given in figure 5.8). After the sinter leaves from the furnace the breaking and sieving application applied on sinter. The more fine parts (less than 6mm)

send to sintering again, and the other particles which are between 6 and 20 mm fraction are used for grate saddle. The big particles (20-60 mm) send to the blast furnace.

12-30 mm thickness materials put on sinter grate furnace. After the specific time there will be zone, which are define in below, on this sinter materials (Portillo, 1996).

- Sinter zone
- Combustion and sintering zone
- Calcinations zone
- Drying and preheating zone
- Wet materials zone

Figure 5.7 Circle Cooler of the sinter plant (Voest Alpine Industrieanlagenbau [VAI] ,2008).

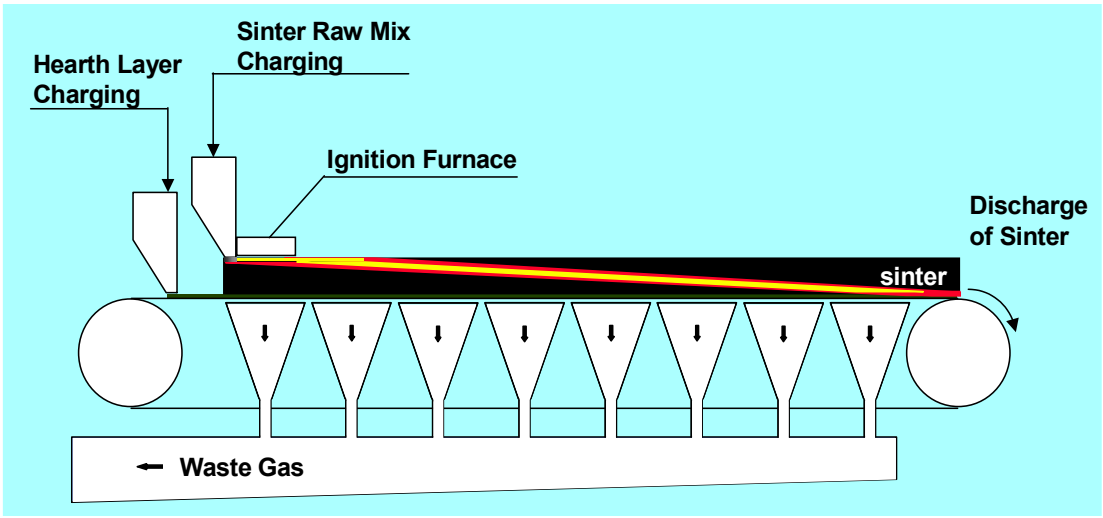


Figure 5.8 Zone that form on Sinter Conveyor (VAI, 2006).

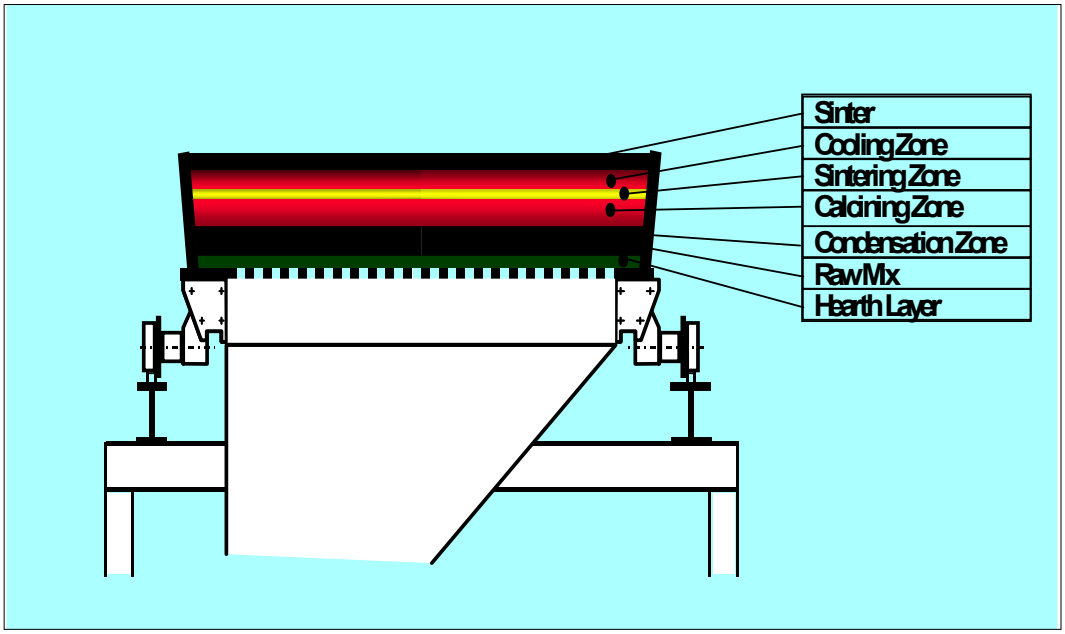


Figure 5.9 Zone that form on Sinter Conveyor (VAI, 2006).

a) Sinter zone: Sintering is complete and sucked air from the base is cooled of zone.

Combusting and Sintering Zone: Coke dust is burn and this causes that heating of zone (1300-1400^{o}C). Cinder makers fused by effect of temperature and the cinder bridge forms between the ore particles, (given in figure 5.9). However, sinter bridge present between ore particles. Additionally CO_2 which forms after combusting, make a reaction with C that present in surroundings, and they form CO after the reaction, also this gas reduce the some parts of hematite (Fe_2O_3) to magnetite (Fe_3O_4). Furthermore sulphur oxides to SO_2 and throwing away are become in this zone (Portillo, 1996).

b) Calcinations Zone: Carbonates break down and CO which is in hot gas, reduce the hematite to magnetite in this zone. Also the big particles start to crack by effect of heat.

c) Drying and Preheating Zone: Materials that will sinter are drying firstly and then heating to specific temperature. After this heating some thermal cracking occurs.

Wet Materials: This zone is not drying and also the gas which is loading by humidity give some humidity from itself to this zone. This zone should have good air permeability (Portillo, 1996).

5.6 The Effect on Sintering

Characteristic of sinter is important because of mainly determining the performance of blast furnace. The reasons of sintering are raising the particular size, forming high reducibility agglomerate, removing sulphur and volatiles and forming flux for blast furnace charge. There are no criteria or tests for determining the sinter quality. Efficiency performance of blast furnace is equal to performance of sinter plant. However, the operator of sinter plant creates the prescription for product which will be accepted for blast furnace. The most common approaches about this are blending size distribution, strength, reducibility, chemical compound (Cohen, 1953).

5.6.1 Coke Effect on Sintering

Two types of fuel are used for sintering. One of them is coke dust as solid and the other one is coke gas. Coke dust control in sinter is made by with visual control at refractive wharf. The 2/3 parts of sinter will become black and the other parts of them will be reddish, if there is normal coke. Also if there are some unbaked parts so this means that there will be a few cokes. In addition if there is a burning continuously in wind clack, there will be much coke. So the coke amounts should be arranged .

5.6.2 Sinter Size

The sinter size has the big effect on sintering. Normally the ore low and high size should be between the 6.5-12.5 mm. The particles which are bigger than the 12.5 mm cause the lack of homogeneous sintering. Also this size are increased the permeability of sinter but it's prevented the excellent bonding. The big particles can not create the full attachment. And sometimes they play a role to disintegration. The specific sizes of ores are suitable for sintering.

Sinter size has an important effect on performance of blast furnace. There is no sinter size as a universal however; it should not forget that fine powder is harmful for blast

furnace processes. The gas permeability of blast furnace charge is decreased by fine powder, loss of powder is increased. Enormous sinter is also not suitable, too. Especially if its both reducibility and strength is low, during processes in the blast furnace there is physical breaking because of this reason (Portillo, 1996).

Sinter size is controlled at sinter processby only making limited changes. Fine sizes can be decreases by making special arrangement as better screening, increasing strength (by arranging fuel quantity), improving ignition, increasing returned sinter, height of the blend and speed of sinter machine.

Observation of plant and laboratory tests indicates that when bacilty ratio is increased between 0.5 -1.2 , the +25 mm of ratio decreases.

The most effective way to remove the fine powder can be screening in the blast furnace stokes so this product will be recycled to the process economically (Strassburger, 1969).

5.6.3 Sinter Dust

The total number of sinter and its types has the effect on sintering. There is a specific value for sintering. The increases of sinter dust quantity are increased the rate of combustion and the resistance to degradation by impact. Because the dust become sintering at previous stage. There is no need to another stage to sintering. Many of the plants add sinter dust as 20-40% to sinter blends.

The blend temperature before the ignition increased to 60-70 ^{o}C by using sinter dust. And this cause that increasing homogeneous of humidity of blend and also decreasing the humidity denser on ore beds base at first level sintering and so provides higher permeability (Cohen, 1953).

5.6.4 The Quality of Base Effect on Sintering

Sintering becomes faster when the quality of base is increased to 1.0. The quality of base increasing supplied the ordering of reduce properties. This properties make sinter more infrequent structure and bonded to more reducible mineralogical phase shape.

Many times when the quality of base is increased to the 0.6-0.8, there is a important strength increasing. The experiments show that adding $CaCO_3$ provides more strength than adding $MgCO_3$ (Camci & Aydin, 2000).

KNEPPER developed the effect of quality of base on strength, and found the decreasing of strength between the 1.0-1.5 zones. There is discussing about the effect of limestone on strength. Dispersion of limestone particles has the effect on strength.

5.6.5 Effect of Humidity on Sintering

The amounts of humidity have the big effect on sinter production. The aim of humidity is obtain well pelleted blends for maximum ore bed permeability. The humidity is given according to blends properties. The small changes of humidity amount might have negligible effect on sinter quality. But it has the big effect on production rate. Also the methods of giving water and where it is giving have important role on sintering. Water should have good sucking so another spray should use. The ultra humidity provokes cancellous earth, and this is separate during the sintering and transporting. Additionally it is causing that increasing of density and decreasing of permeability (Sevinc & Topkaya, 2001).

5.6.6 Sinter Strength:

Strength is the most important factor which supplies the sinter quality. Usually, it is the single index for sinter quality. If the sinter strength gets increase, the fine powder will get decrease and it will be supplied increase in output of iron blast furnace as noticeable.

There are some methods for operator of sinter machine to control the sinter strength (Dawson, 1985). Here are the most suitable parameters of set and control;

a) The fuel quantity for heat input
b) Ratio of air /gas and degree of ignition
c) The moisture quantity in the blend for permeability
d) The speed of machine supplying suitable and complete combustion
e) The quantity of sinter coming back
f) Height of the blend

In addition to these, there are process variable usually not related to operator of machine, for routine control, however affecting sinter strength in different degree and needed to exist during the control measurement (Cohen, 1953). These are;

a) Suitable design and operation of mixing equipments.
b) Consistency proportion of raw materials.
c) Suitable design of ignition unit.
d) Using preheated or completely heated air
e) Using hot, coming back material (as sinter, flue dust etc.)
f) The fuel having suitable dimension
g) The lime stone having suitable dimension(for sinter with flux)
h) The maximum dimension for ores which will be sintered.

In addition to factors making strength, mostly of these factors has to be provided in planet process. The deviation which occurs because of normal operation condition deliberately or failure of apparatus should be minimum (Strassburger, 1969).

5.6.7 Composition of Sinter

The criterions for sinter is basically resultant of maximum content of iron and minimum content of gang, reasonable strength and reductable and performs of blast furnace. Content of iron almost completely is related to ores which is selected by operation of institution and this selection is done because of economic reasons with sintering.

The chemical analysis of fuel is important for selecting fuel. Especially, fuel with low content of ash which causes decrease of content gang in sinter should be selected. The effects of components of sinter on economy and sinter fuels are examined and an expression is produced according to quantity of carbon and ash values. The quantity of ash affects the relative values of fuel as factor of 1.4 at least (Dawson, 1985).

Close control of blending raw material is obligated. Furthermore, operator should work for optimum sinter machine. Therefore, disorder of treatment and deviation of process variable is restricted and the effect of chemical composition is cut down minimum (Dawson, 1985).

5.7 Factors Effecting Reducibility

There are many factors which affect the reduction speed of ore and sinter. Here are the important ones:

5.7.1 Ore Quality

Although hematite includes 12% more oxygen which should be removed than magnetite, reducibility of hematite is better. Figure 5.10 indicates that increases in oxidation degree of magnetite due to increases of reduction speed. The reason of that hematite is reduces well may be these:

a) Reduction gas permeability of porosity wustite formed from hematite is more than concentrated wustite formed form magnetite.
b) Hematite has more wide and longer micro vacancy, magnetite has more wide grain dimension and its structure is tighter package.

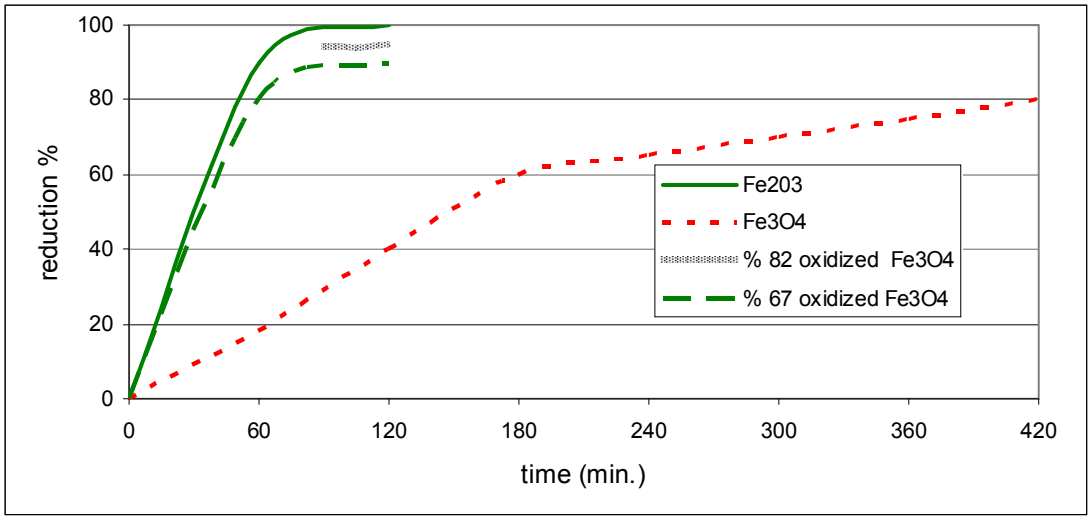


Figure 5.10. According to the increase in degree of oxidation of magnetite which reduction rate increases (Biswas, 1985).

c) The reduction speed constant of wustite formed from hematite has high value because it shows disorder at higher degree than formed form magnetite (Biswas, 1985).

5.7.2 Sinter Quality

During making acid sinter, fayalite reduced difficultly is formed related to ore of silica content the ratio of coke in sinter blend. Thus, quantity of fuel and so quantity of fayalite increase, after that the reducibility decreases. When quantity of fayalite increases, sinter strength increases, too. Therefore, when strength and reducibility are obtained enough, making acid sinter can be started (Jasienska & Durak, 1999).

Here are probable reasons of low reducibility in acid sinter:

a) Low activity parametre of FeO inside fayalite
b) Closing vacancies with liquid slag and passing reduction gases rarely
c) Low oxidation degree, low ratio of Fe^{+3} / Fe^{+2}, if the ratio is high, the content of hematite will be more, during reduction cracks ,splits and vacancy are occurred.

If lime is mixed into sinter blend, the sufficient strength and reducibility sinter is obtained. Increasing basicity causes decreasing in quantity of FeO. Figure 5.11 indicates this situation.

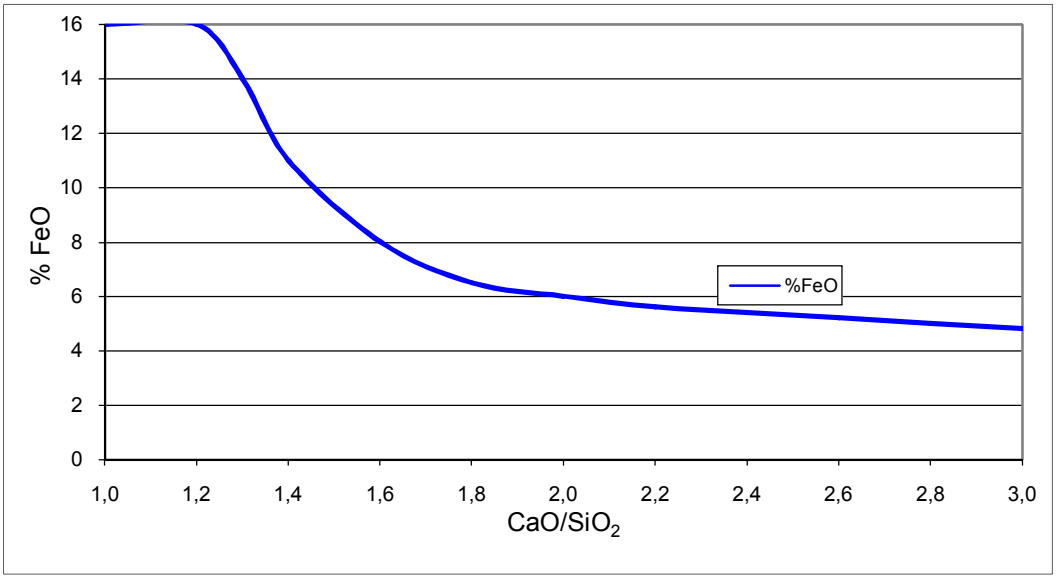


Figure 5.11. The decrease of FeO content with increasing amounts of basicity (Jasienska & Durak, 1999).

At figure 5.12 reducibility and content of FeO indicate as function of CaO/ SiO_2 and coke ratio, dR/dt (40%) or R_{60} which is ratio of removed oxygen quantity at 60^{th} minutes of reduction can be indicated removal per minutes 0% as quantity of reducibility. These can be the reasons of increase reduction in limy sinter (Kimura et al., 2005).

a) Increases of lime in silicate including iron can be decreased by forming calcium silicate. Besides, it also raises the activity coefficient of wustite.
b) Calcareous hematite tends to form irregular and therefore more reactive wustite.
c) Firstly, ferrous oxide and rich ferrous ferrite are reduced to FeO and $2CaO.Fe_2O_3$. Then, reduction gas preferably turns into dicalcium ferrite as in the reaction below.

$$2CaO.Fe_2O_3 + 6CO = 2CaO + 2Fe + 6CO_2 \quad (5.1)$$

Ferrite regenerates by reaction of re release lime and wustite.

$$2CaO + 3FeO = 2CaO.\ Fe_2O_3 + Fe \quad (5.2)$$

Reformed ferrite is reduced with CO immediately because being high reactive condition.

d) Contrary to lime free wustite, reduced Wustite covered with high porosity iron layer, including lime has high reaction speed. The connection is found by experiences done by Keddeinis and Sticker (1998). According to the results of these experiences, the connection of sinter with FeO content reducibility between 0.25 -2.0 range of basicity is found as below;

$$\frac{dr}{dt}\ (\%/\text{min}) = 2.43\text{-}0.12\ (\%FeO) \quad (5.3)$$

According to the connection, 1% change inside FeO quantity will cause 0.12/ min % change in reducibility.

5.7.3 Oxidation Degree

The oxidation degree of magnetite is 88%, the oxidation degree of hematite is 100%. Reduction of hematite is better than magnetite. Thereby, increases of oxidation degree causes increases of reducibility. This situation is indicated in figure 5.12 with different couple basicity lines. The figure also indicates that reducibility can increase by basicity.

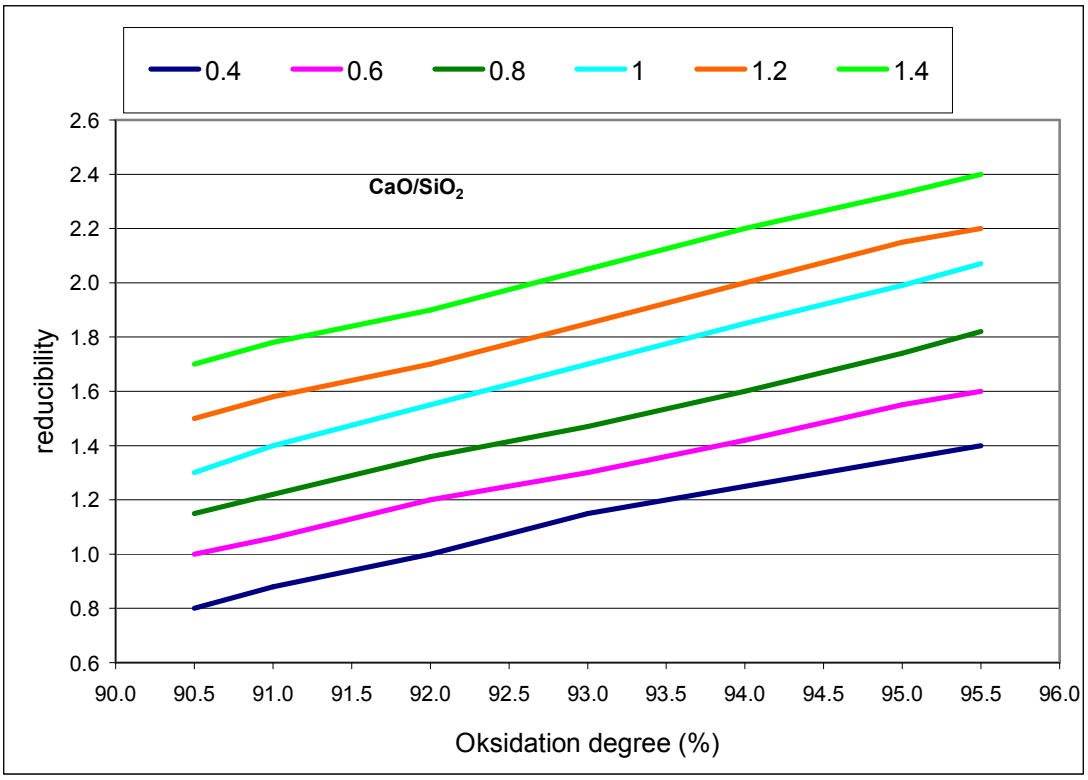


Figure 5.12. Increasing of oxidation degree and basicity effect on sinter reducibility (Ball, 1973).

The decrease in coke content of sinter causes decrease in FeO content. Similarly, when the coke content decreases, oxidation degree increases or the oxidation degree increases by increasing basicity for steady coke content. Consequently, high degree of oxidation sinter can be produced with relatively low content of coke ratio and increasing lime content (Ciocan, 2000).

5.7.4 Porosity

When the gas comes into sinter material, sinter can be easily reduced with reduction gas. If the porosity is high, the inner surface will be wide because of occurring reduction. Thereby, the higher porosity, the higher the rate reduction is expected (Egundebi, 1989).

Porosity always changes when the reduction improves; because some events occur like a leaving oxygen from oxide cage, moving gas products away, cage of broken, phase transformation, weakening and volume alteration, compression load, iron softening and recrystallization, gang softening and filling into the porosity.

5.8. The Effect of Blending Composition on Sinter Production and Quality

5.8.1 The Effect of Blending Composition on Sinter's Chemical Composition

At sinter production, chemical analyses on 12 productions at different times in daily production are made. According to these chemical analyses the changing at production that has FeO and sulfur content, sinter basicity and returning sinter dust amount is examined (Kimura et al., 2005).

a) The effect of basicity on FeO content in sinter production. The changing of the FeO amount with basicity rate at production is given at figure 5.13

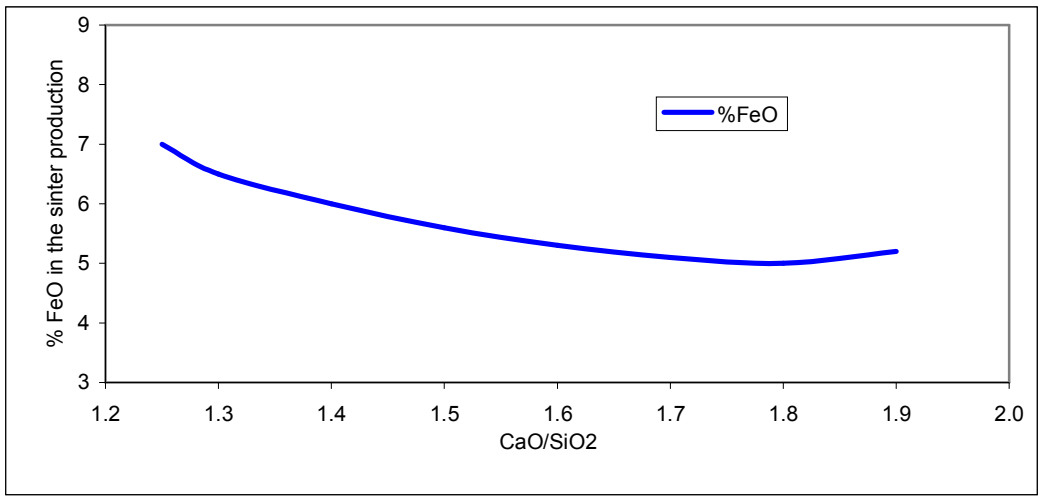


Figure 5.13 The variation of rate of FeO with basicity of sinter production (Egundebi, 1989).

It is seen at figure 5.13 that the FeO rate at sinter's decreasing with increasing at basicity and with increase at BOF slag amount added to blending, FeO amount in sinter take high values.

b) The effect of basicity on sulfur content in the sinter: The changing of sulfur at sinter production, according to basicity is given at figure 5.14. As seen from the figure as basicity increases sulfur content increases at production sinter. The sinter including high FeO, sulfur amount is less low.

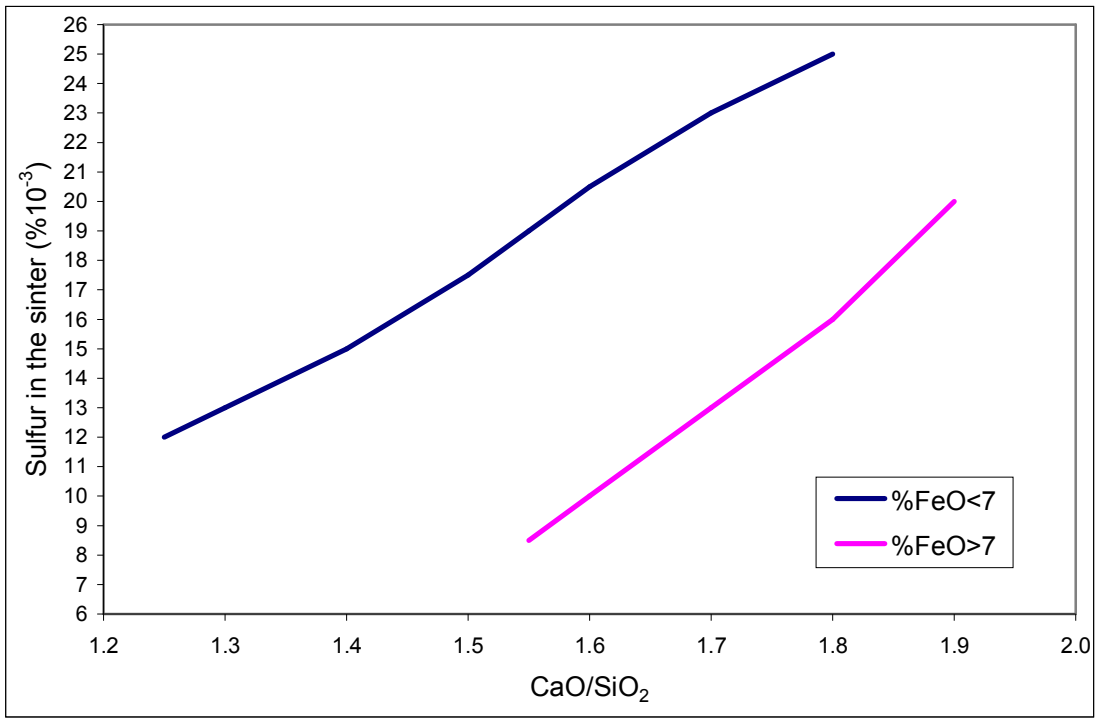


Figure 5.14 The changing of ratio of sulfur at production sinter depend on sinter basicity (Günaydin, 2002).

c) Sinter dust returning to blending effect on sulfur content at sinter: With the rate of sinter dust returning to sinter blending the changing of the sulfur content at production sinter is given at figure 5.15

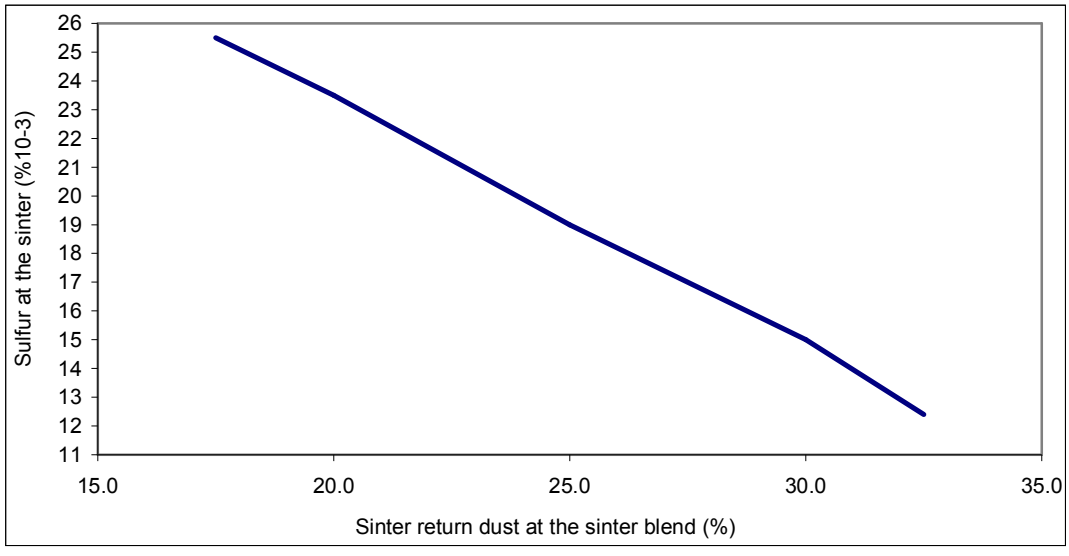


Figure 5.15 Changing of sulfur content to sinter production with add of BOF slag in the sinter blend.

As seen from the figure, with increasing of returning sinter dust, the sulfur content of sinter decreases.

d)The effect of returning sinter dust on FeO content at production sinter: With the returning sinter dust rate, the changing of FeO amount at production sinter is given at figure 5.16. As seen from the figure as the sinter dust rate returning to the blending increases, FeO rate increases at sinter.

Figure 5.16 FeO content at sinter production changing of the rate of return sinter dust to sinter blending

5.8.2 The blending compositions' effect on sinter's physical properties

At sinter production in daily production taken 12 productions at different time tumbler strength tests are made. With the composition of production sinter the changing of tumbler strength values are examined (Kirk, 1997).

a) The effect of basicity on sinter strength

The effect of basicity on sinter strength is given figure 5.17

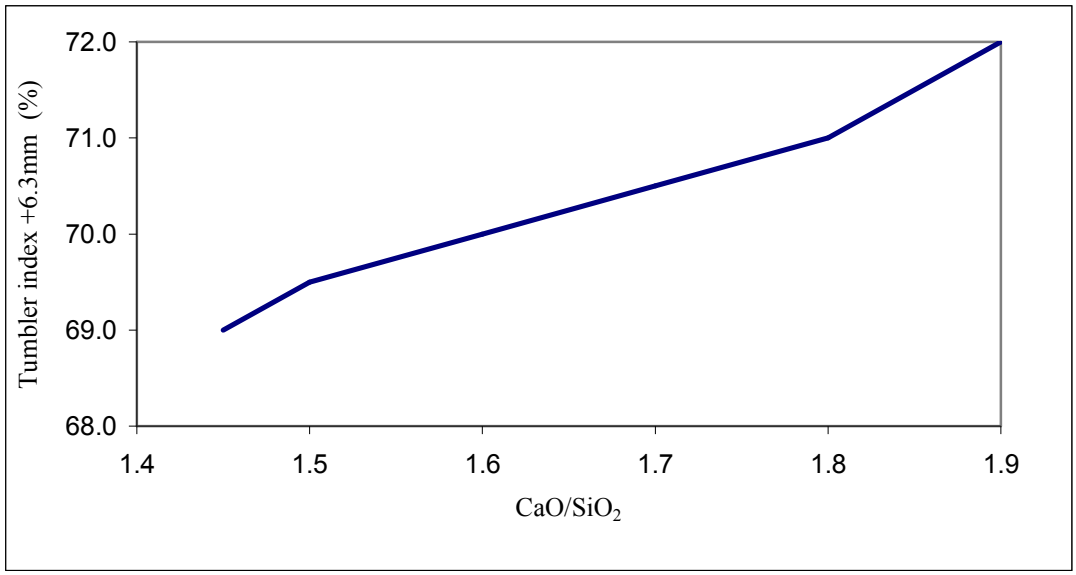


Figure 5.17 The variation of sinter strength with respect to total basicity of sinter

In the case of sinter the rate of limestone added to blending being more than % 2.5, strength decreases regularly with the increase for basic provided with BOF slag because of slag's complex structure an systematic changing is seen.

b) The effect of FeO in the sinter on Strengths

The rate of FeO at sinter composition's effect on sinter strength is given at figure 5.18

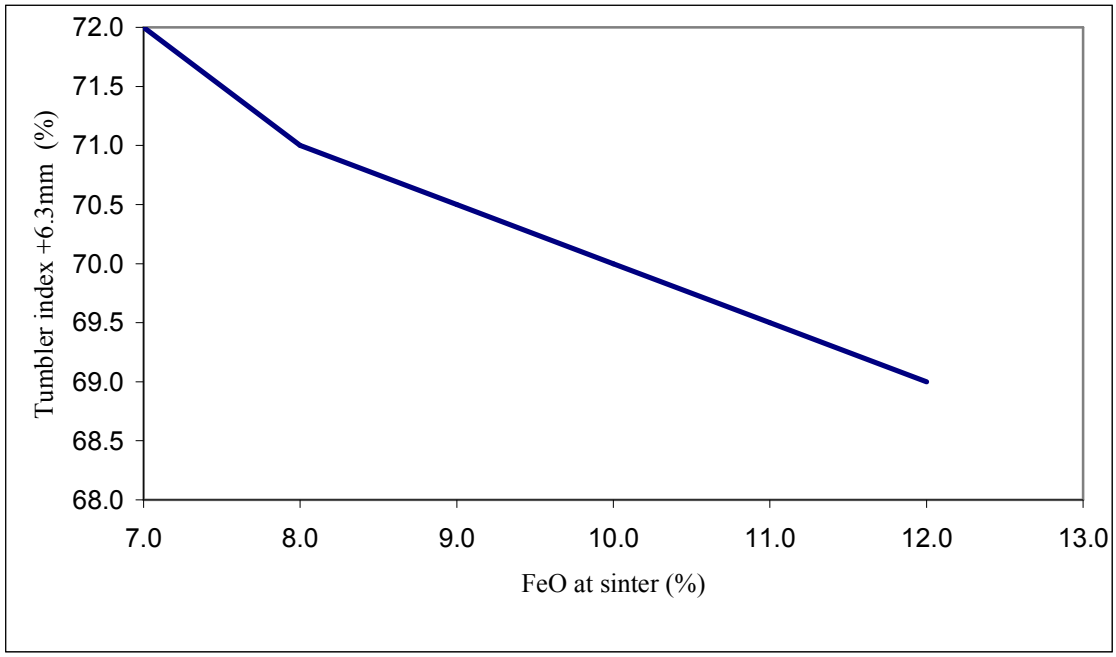


Figure 5.18 The variation of sinter strength with respect to FeO content of sinter production

5.8.3 Mineralogy of Sinter

The nature of agglomeration effected by connectioning the economy, transporting, storage and processing and finaly affects reduction of blast furnace. Connection basically is provided by the melting of the slag and oxide particles. This connection is created in combustion zone in which the ore components and partial melting of gang compounds. Diffusion bond or grain growth within and between ore particles has similar importance (Strassburger, 1969).

Transportation of many minerals can be seen related to chemical compounds of flux and ore as fuel ratio in sinter product. Properties such as resistance and a good reduction of good sinter, kinds of produced mineral and melting temperatures are important. If degree of oxidation or quantity of hematite (Fe^{+3} content) is high and quantity of magnetite or wüstite (Fe^{+2}) are low, reductionablity will become higher at the same rate (Biswas, 1985).

Oxidation degree is determined as below:

Oxidation degree = oxygen quantity related to iron / amount of oxygen required for the formation of Fe_2O_3

% Oxidation degree = [l-(Fe^{+2} /3ΣFe] x100. (5.4)

Oxidation Degrees of three iron oxide;

FeO = 70%, Fe_3O_4 = 88.9%, Fe_20_3 =% 100,

Sinter material includes three types of material:

- The original unchanged material.
- The original mineral compounds which change their structures and shapes with recrystallization when they are solid phase.
- During the sintering of the components are molten or dissolved material..

These components can stay as dissolved materials or precipitate. the mineralogy of sintering basically is related to the chemistry of main material, character of minerals and the maximum process temperature. Moreover, the solidifying

temperature, cooling rate, and atmosphere of sintering sometimes affect the final mineralogy as seen in the examples. Therefore, knowledge of detailed process cycle should be learnt well to connect the relationship between the final sinter mineralogy and evolution of process (Nosovitskii & Zhilin, 1980).

The equilibrium condition is rarely obtained because of speed of the sintering process and presenting kinds of materials as relatively wide size. The most important part during the sintering is start of transformation to magnetite because burning of gases can reduce slightly the hematite. Although temperature of the sintered layer decrease despite to the cold air passing, the changing amount of magnetite reoxide to secondary hematite. Presenting hematite according to type and quantity makes the property of breaking into pieces worse (Nosovitskii & Zhilin, 1980).

Sinter strength are provided as following below:

a) Slag or melting bond:These are occurred when crystal is partially or completely buried into molten, glassy slag according to being able to wet and volume and of molten phase. Bonding strength depends on amount of crystals in glassy phase and types and amount of compound. These also depend on fuel ratio, and presenting or addition of SiO_2, CaO, MgO.

b) Diffusion bond: These occurs with recrytalization of hematite and magnetite and grain growth. At high temperatures, especially over 1250–1300 ^{o}C, diffusion of hematite has important role This kind of bonds are derived from ores with the high iron low SiO_2. Because gang in the poor ores is formed before the fluids with low melting temperature reach diffusion temperature. Slag bond with rich sinter has less important. Principle connection is obtained by recrytalization of hematite and magnetite and grain growth.

The change of mineralogy in low content of MgO and $A1_2O_3$ with CaO/SiO_2 ratio at the property of optimum sinter is shown in the figure 5.19.

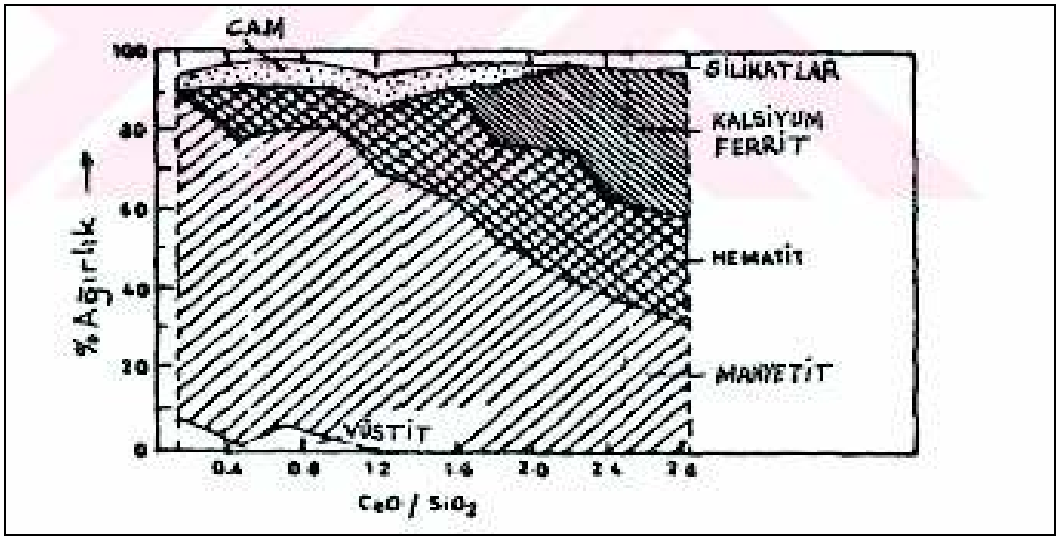


Figure 5.19. The change of optimum strength mineralogy with ratio of basicity (Pramanik, 2000).

5.8.4 The Quality of Sinter Mineralogy and CaO/SiO2 ratio

5.8.4.1 Ratio of CaO/SiO_2 =0.0

If there is not any lime in acid sinter, the main compound is fayalit ($2FeO.SiO_2$) which can be formed from magnetite or hematite directly. If there is SiO_2, Fe_3O_4, Fayalite (melting point 1210 ^oC) forms multi-component liquid with low melting point when presenting increasing viscosity with silica content. If there is more Silica and fuel, the huge amount of water will be formed. In this situation, the sinter which is low grade oxidation and weak reducibility, high strength, contains 20-30 Fe %. Sinter obtained from Containing low SiO_2 and/or low ratio of fuel is weak but it can be reduced with high rate of oxidation well and content 10-15 % Fe^{+2}. Thus, diffusion can have an important role in bond strength. Coke ratio, strength, reducibility, oxidation degree and fayalite content of acid sinter can be seen in figure 5.20 clearly (Pramanik, 2000; Nosovitskii & Zhilin, 1980).

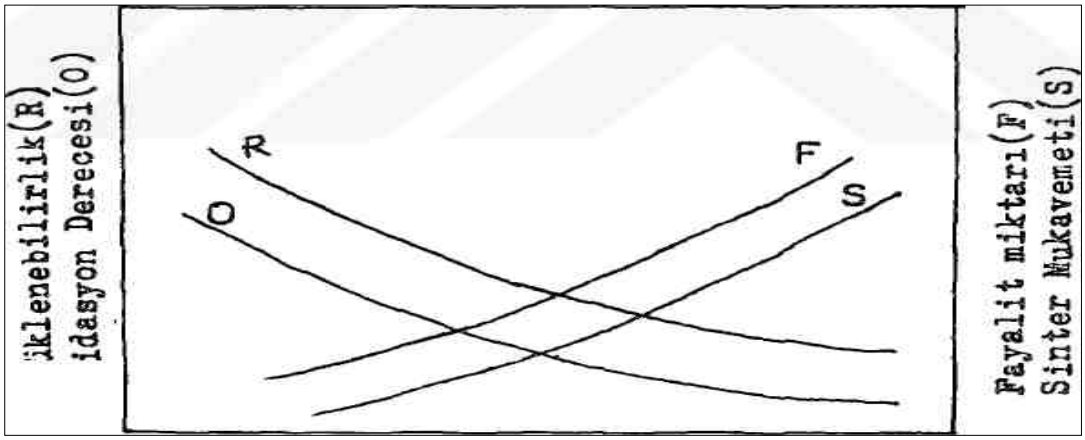


Figure: 5.20. The relationship between Rate of Coke Strength, reductionable, oxidation degree and content of Fayalit (Biswas, 1985).

5.8.4.2 Ratio of CaO/SiO_2 = 0.0-0.8

When the basicity gets increase, the volume of slag get increase and glass takes place instead of crystal and silicate as increasing. Phase of slag mainly includes silica glass(combinations of FeO,CaO,Al_2O_3,SiO_2) fayalite, Wollastanite ($CaO.SiO_2$) and anortite ($CaO.Al_2O_3.2SiO_2$). Lime makes the glassy structure which needs less heat and fuel less viscose. Thus, the oxidation degree get increase and content original and secondary rate of some scoria CaO/SiO_2 volume increases and the increasing availability of crystal Silicates take the glass. Major slag phase basically includes silica glass (FeO, CaO,Al_2O_3,SiO_2 combinations), fayalite, Wollastanite ($CaO.SiO_2$) iron montiçellit (CaO. $FeO.SiO_2$) and anortit ($CaO.A1_2O_3.2SiO_2$) . Glassy structure and low temperature fuel lime requires more, makes less viscous, so the degree of oxidation increases and original and secondary hematite content increases. With increasing basicity, the amanount of fayalite decreases and phase of rahkinite ($3CaO.2SiO_2$) is, come across more. Mono Calcium ferrite ($CaO. Fe_2O_3$) which settles in sinter pore and is dispersed from the sinter pore can be encountered. The grains of magnetite are embedded into the slag matrix of calsium ferrite Silicate. During cooling, magnetite particles are net to the wide pore including secondary hematite ribbon which is formed by reoxsidation. There are little magnetite crystals in large part of matrix of slag, dendiritic orientation which is formed by precipitating during cooling. Dendirtic structure is seen in all the basic values of sinter (Biswas, 1985).

5.8.4.3 Ratio of CaO / SiO_2 = 0.8 -1.4

With increasing basicity, the amount of glassy phase increases and amount of fuel decreases. Thus, strength keeps on decreasing and the reductionablity continues increasing (sometimes it can decrease). Knepper between 1.0-1.2 ratios of basicility notifies devitrification of bond of glass silicate with the way of separating dicalsium silicate crystals and occurring calcium ferrite. In the range of 1.4-1.6, β-2 $CaO.SiO_2$ crystallization occurs. Volume of water becomes maximum and strength becomes minimum in this stage. This situation probably occurs because of during cooling comprising of phase transformation of β dicalsium silicate which forms weak point and cracks (Pramanik, 2000; Nosovitskii & Zhilin, 1980).

5.8.4.4 Ratio of $Ca0/SiO_2$ = 1.4-2.8

While the basicity increases above 1.4, calcium and ferrite takes the place of glassy matrix. These form progressively the structure of bond. There is stable incensement in strength, needle ferrite precipitate and grains of ore are hold on together. Besides, reductinable continuous increases. This basic rate of sintering is super flux sinter (Nosovitskii & Zhilin, 1980).

5.8.4.5 Reducibilities of different phases

Fe_2O_3> $CaO.2Fe_2O_3$> $CaO.Fe_2O_3$> $2CaO.Fe_20_3$> Fe_3O_4> Olivine> Fayalit

Little rate of fuil may be enough for providing the same value of strength because ferrites form strong bond. The viscosity of ferrite is around 0.35 – 0.15 poise at 1300-1600 ^oC, this is 10 times higher than silicates. The problem of decreasing sinter can be made decrease by using MgO and Al_2O_3. Sinter including % 4.5 MgO and % 6.5 Al_2O_3 can give a correct result in low temperature crashing test. Alumine makes range of solidification of dolomite sinter larger, cooling stress lighter, mechanic strength stronger, reductable better, and oxidation degree increase. Approaching the mineralogical composition of the product sinter to the equilibrium conditions depends on temperature and sintering time. If the cooling is fast, top of sinter cake break easily and the structure of it becomes glassy because of inadequate preheating. Because of that at 800-900 ^oC temperature crystallization finishes, applying cooling at 600-700 ^oC temperature

produces better sinter strength. Since, the thermal stress increases and the glassy phase in microstructure disappear (Pramanik, 2000).

5.8.5 Evolution of Sinter Microstructure

According to the lots of researches done on sintering, the relationship between the formation of microstructure and sinter reactions is shown in the figure 5.21

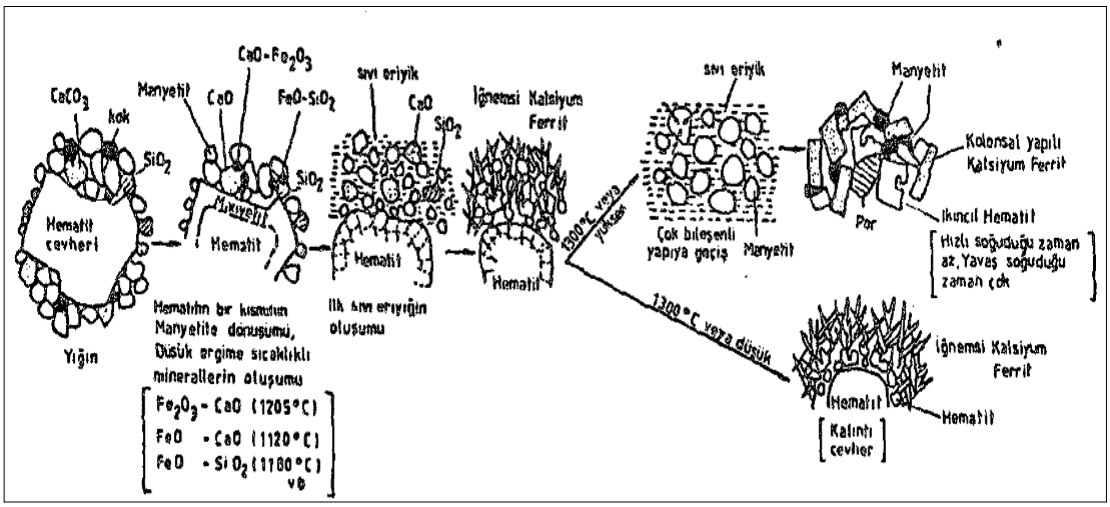


Figure: 5.21. Evolution of Sinter Microstructure (Şeşen, 1985)

- The molten (basically including Fe_2O_3 and CaO) is formed approximately 1200 ^oC temperature and iron oxide and subagent are absorbed with it. If the molted enters among boundary of hematite grains in the big ores, cracks are occurred and the granular hematite is won (Günaydin, 2002).

- The molten reacts with the iron oxide because of that CaO and Al_2O_3 are absorbed in definite measurement and acicular ferrite containing Al_2O_3 and SiO_2 is formed by taking solid solution.

- When the temperature is higher than 1300 ^{0} C calcium ferrite dissolves and turns into molten hematite or magnetite and components of slag. If content of Al_2O_3 is so little or the duration is short, after cooling, secondary hematite causing reduction crushing (Günaydin, 2002).

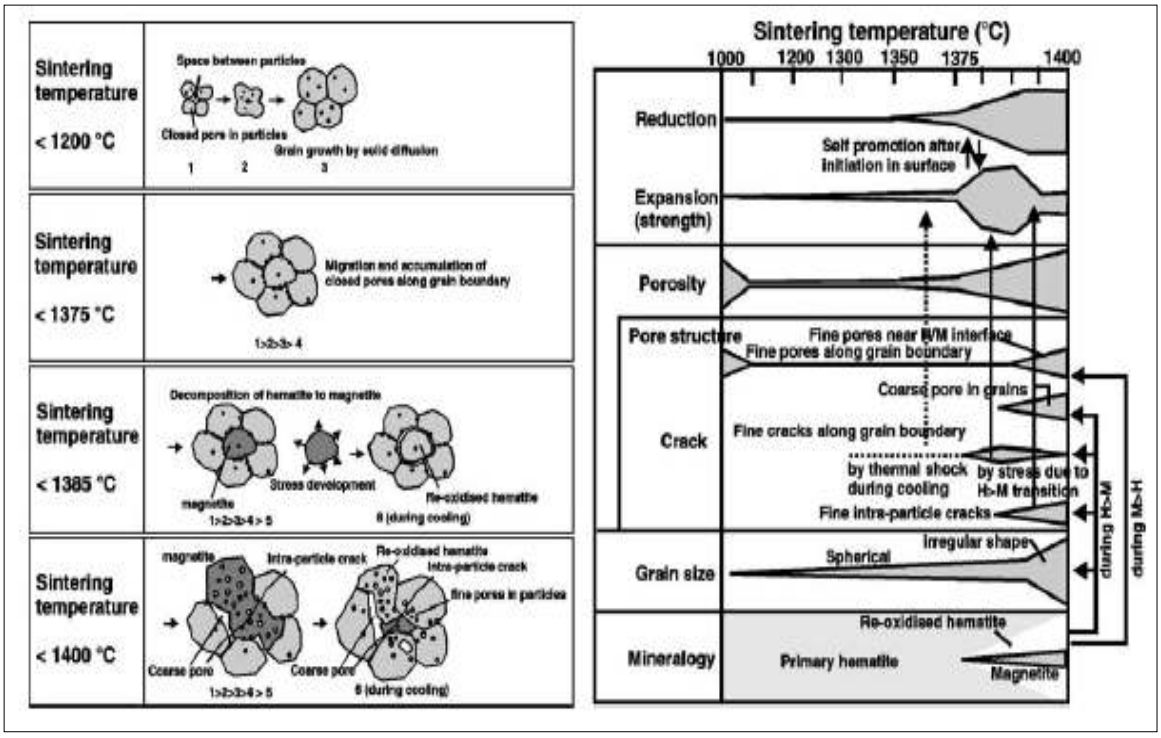


Figure 5.22 Schematic overview of the influence of the sintering temperature of pure hematite on microstructural characteristics and on behaviour during reduction (Sasaki et al., 2005)

From microstructural analysis, the mechanisms occurring during sintering, i.e., the evolution of microstructures of pure hematite, are proposed and summarized in figure 5.22. Below 1200 ^{o}C, the grain growth rate is low, resulting a small grain size of less than 5 μm. Fine pores from space among initial particles, and original fine closed pores within particles are distributed homogeneously. At 1200–1375 ^{o}C, further grain growth proceeds and the crystallite grain size reaches approximately 30 μm. Many fine pores (<2 μm) are trapped, mainly at grain boundaries of the hematite. Some fine pores remain within the interior of the grains. Above 1300 ^{o}C a small number of cracks are formed along grain boundaries due to thermal shock during cooling. At 1375–1385 ^{o}C, hematite grains near the surface of the tablets decompose to magnetite. Most of them are reoxidised to hematite again during cooling. Due to the formation and subsequent re-oxidation of magnetite, stress develops due to volume increases and results in creation of a small amount of space between reoxidised hematite and primary hematite grains (Chaigneau, 1994).

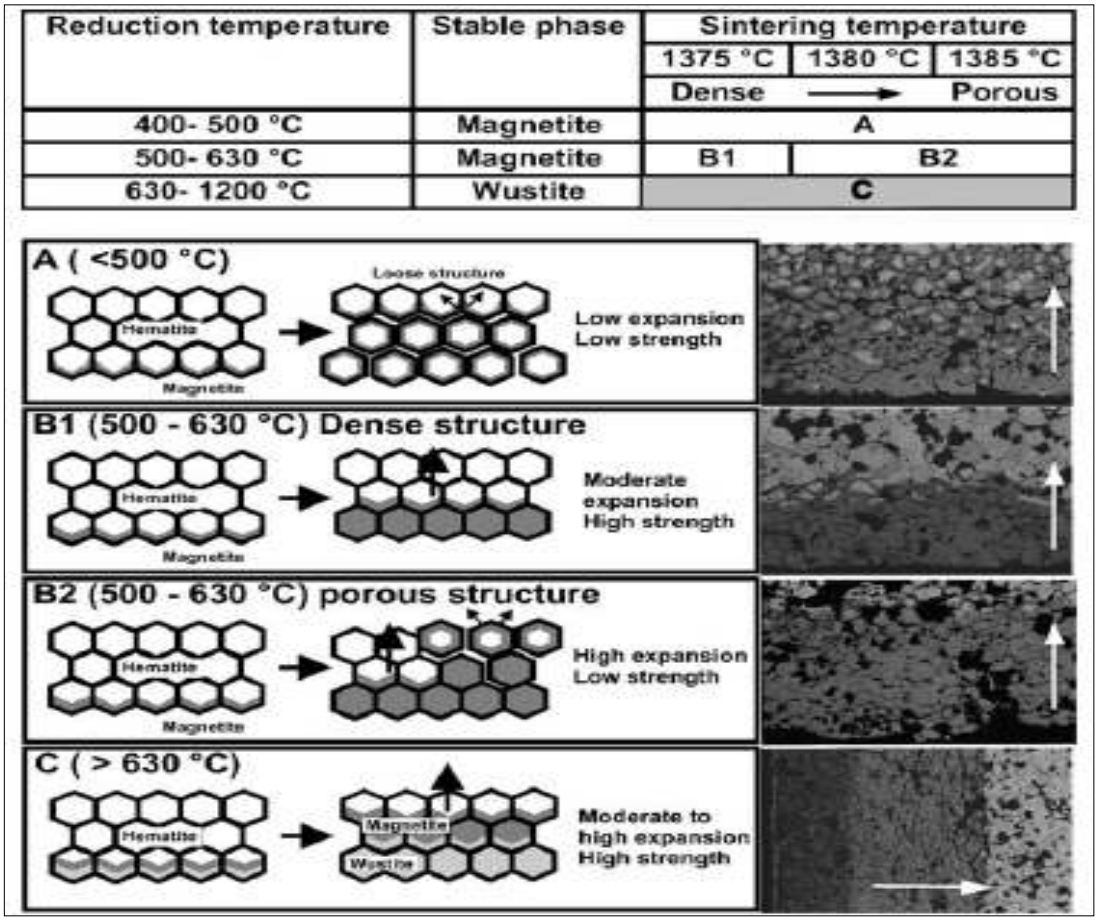


Figure 5.23 Schematic classification of reduction behaviour mechanisms (Sasaki et al., 2001)

Figure 5.23 shows a summary of the proposed reduction mechanisms. The reduction mechanism changes with the reduction temperature. Furthermore, during reduction to magnetite at 500-630°C, the reduction mechanism was also affected by the sintering temperature (Hosotani et al., 1996).

Mechanism A: During low temperature reduction to magnetite (400-500°C), reduction gas diffuses into and inside tablets via pores and cracks along the grain boundaries. However, reduction into the interior of individual grains is limited by the low chemical reaction rate at the interface between hematite and magnetite. Porous magnetite surrounding hematite cores in the individual grains causes a loose structure of the tablets, which results in low strength. Stress developing in the surrounding magnetite, however, is distributed homogeneously, which results in low expansion.

Mechanism B: During moderately low temperature reduction to magnetite (500-630 °C), two types of reduction mechanism are identified. Chemical reaction rate is increased

by the increase in reduction temperature, resulting in formation of a well-defined topo-chemical layer of magnetite from the outside inward. The straightened reduction front causes anisotropic stress, resulting in increased expansion. However, in the case of reduction of dense tablets, reduction proceeds only by chemical reaction between hematite and magnetite. This reduction is rather slow. Therefore, the formed magnetite is annealed and the stress is eased, which results in a moderate level of expansion and relatively high strength the of reduction of more porous tablets, pores and cracks along the grain boundaries promote reduction. Therefore, the reduction rate is high, which results in high expansion levels and low strength (Mechanism B2).

Mechanism C: During high temperature reduction to wustite (630-1200 °C), the high chemical reaction rate results in concentric topochemical layers of magnetite and wustite. Reduction rate is controlled by the chemical reaction rate. Because both the formed magnetite and wustite layers are porous, reduction gases readily diffuse into these regions. The thus formed, more or less straight reaction front causes anisotropic stress, which results in high expansion (Hughes, 1982).

5.8.6 The Important Components of Basic Sinter Microstructure

Calcium ferrite (SFCA), unresolved collapsed iron ore particles from the liquid solution oxides, such as magnetite and hematite as a dicalcium silicate matrix.

Lime forms the first liquid by reacting with pieces iron ore (1200 ^{o}C). This liquid forms a molten by absorbing the silica alumina and pieces of iron ores. During solidification, this molten makes bond between the unasimileted particles.

Magnate and hematite occur after passage the high temperature period. Then, calcium ferrite occurs above the hematite or develops or it can be on more magnetite, moreover, it can be in the silica matrix as a phase.

Recrystalized original pieces of ore ties with calcium ferrite (SFCA) as a dominant phase. Ideal structure of sinter with flux includes it. The structural appropriateness between SFCA with magnetite affects the sinter strength positively.

Vacancy and cracks is important for ore flux. Reducibility is increasing with these properties (Hosotani et al., 1996).

5.8.7 Designing High Productivity Sintering Operations

5.8.7.1 Design of Voids, Pores and Micropores

Each university had been engaged in basic research such as granulation, agglomeration, evaluation of reaction in a blast furnace to propose a new process for "porous meso-mosaic texture sinter" which was anticipated simultaneously to realize high strength and high reducibility under the conditions of low slag and high goethite ratio in sintering.

By combining these basic researches, the theory of agglomeration and the engineering foundation for raw material/sintering/blast furnace process were discussed and a new process for “porous meso-mosaic texture sinter” based on improving granulation process of raw material was searched. As a result, proposed was the MEBIOS (Mosaic Embedding iron Ore Sintering) method which involved arranging dense pregranulated pellets (firing cured bed) in a sinter induction bed with a raw material composition capable of forming a suitable void network under normal sintering conditions.

Figure 5.24 shows the design concept of a porous mesomosaic texture sinter cake. If the non-solid parts present in the sinter bed are classified into voids, pores and micro pores, then the presence of micro pores is strongly desired in terms of reducibility while the removal of pores is strongly desired in terms of strength and the presence of voids is strongly desired in terms of ventilation properties. These voids, pores and micro-pores are formed by the combination and growth of non-solid parts inside ore particles or between ore particles that are present in or created in the raw material zone. It is thus desirable to suppress the combination and growth of micro pores into pores, and to promote the combination and growth of pores into voids (Kasai et al., 2005).

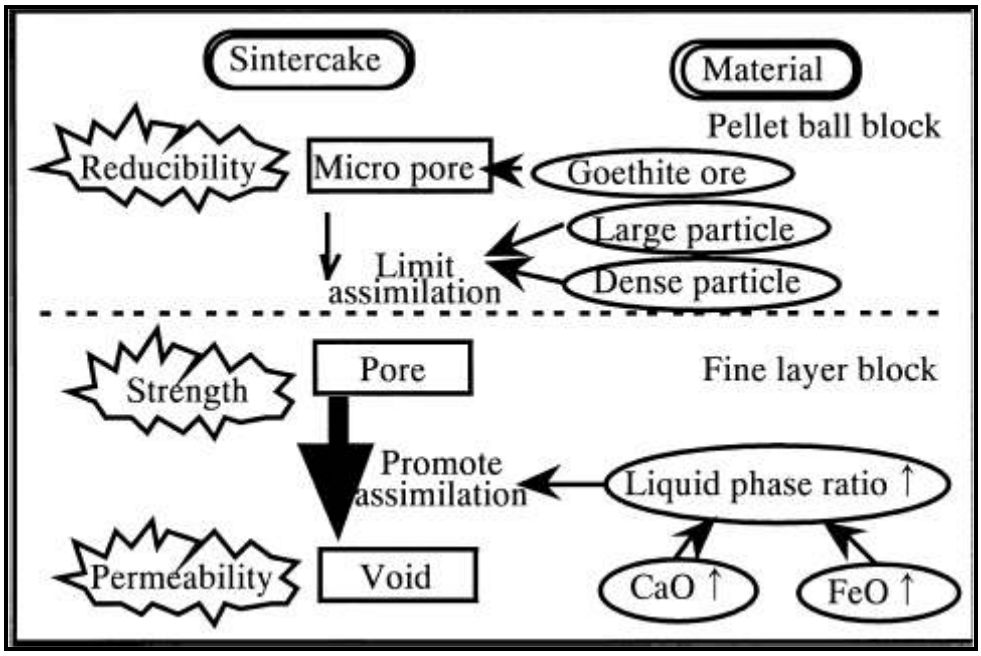


Figure 5.24 Design of sinter cake and material (Kawaguchi & Usui, 2005).

In sinter processing schemes where fuel is charged inside the particles, voids must be kept in the sinter bed as a prerequisite for maintaining ventilation. Accordingly, even if the combination and growth of pores into voids is promoted, a bed column structure that maintains voids must be maintained at the minimum level. The combination and growth of nonsolid parts is governed by the quantity of assimilated melt and the fluidity of the molten parts. The fluidity of the molten parts coexisting in the same liquid are strongly affected by the ratio of solid (liquid) phase, and it is essential to control the melt composition (Otomo, 2005).

5.8.7.2 Non-uniformity of Sinter Raw Materials and Sinter Beds

Because sintering is a process in which heat is accumulated from top to bottom, differences in heat patterns also exist in the vertical direction of the bed. Accordingly, even with the same raw material state, there are many differences in smelting assimilation and flow conditions inside the sinter bed that accompany differences in vertical position in the bed (heat patterns). If the temperature is increased, the solid phase ratio tends to decrease and the fluid state tends to be promoted.

On the other hand, sinter raw materials are vary widely both in terms of their composition and particle size. Particulate raw materials with a small particle size have a

greater degree of surface contact with the melt and are more readily assimilated. Accordingly, as the raw material particle size decreases, the quantity of assimilated melt increases and the melt composition and raw material composition tend to become equal, but conversely as the raw material particle size increases, the quantity of assimilated melt decreases and the melt composition becomes significantly different from the raw material composition (Kasai et al., 2005).

When limonite is subjected to heat treatment, its combined water dissociates, resulting in porous properties with minute pores. These porous ore properties readily promote the assimilation reaction with the sinter melt. The sinter composition has a higher content of iron oxide than the eutectic composition produced in the initial melt, so by uniformly assimilating the ore, the solid phase ratio of the melt increases and the fluidity of the melt decreases. We found that in sinter reactions using hematite ore as the raw material, the initial melt formed from fine-grained components containing CaO and SiO_2 is assimilated by melting the nucleating particle ore, while the non-assimilated nucleating particles are left behind. On the other hand, in sinter reactions using limonite ore as the raw material, smelting takes place after the initial melt has assimilated porous nucleating particles by permeating into them, so that fragments of ore become suspended in the melt and reduce its fluidity (Kasai et al., 2005).

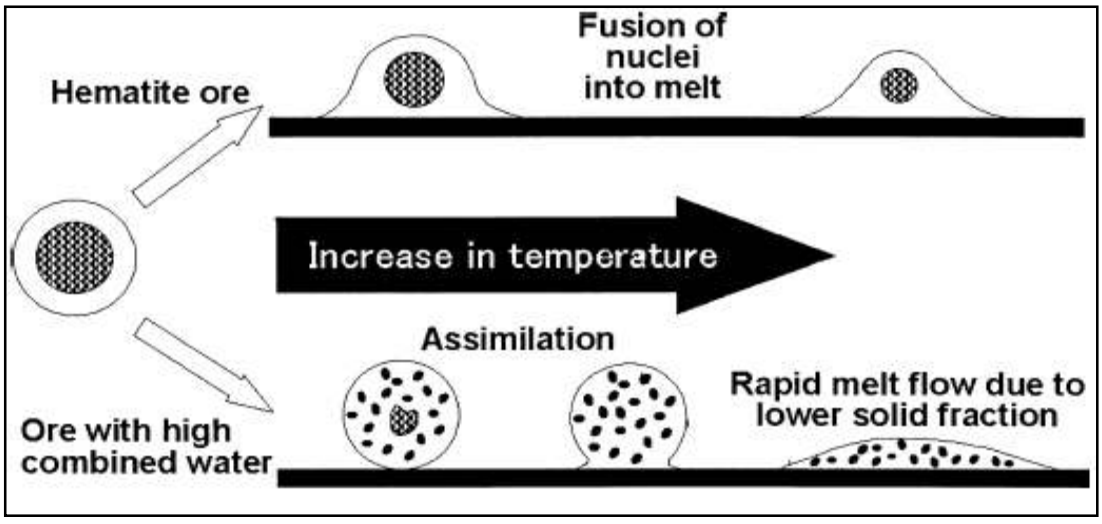


Figure 5.25 Assimilation of iron ore during sintering (Kawaguchi & Usui, 2005).

Thus, as shown in figure 5.25 as the temperature increases, the limonite particles are assimilated more easily than the hematite nucleating particles, but have an adverse effect on the fluidity of the melt. When the temperature is increased until the solid phase has disappeared, the melt becomes highly fluid and is no longer able to maintain voids. So even with the same solid phase ratio, differences in the amount of solid phase present lead to different smelting assimilation behavior and different sintered cake structures.

5.8.7.3 Designing Sinter Raw Material Beds

By taking the abovementioned characteristics into consideration to obtain an assimilation control method for forming sinter beds with well developed voids and few pores, we propose a MEBIOS composite granulation loading method in which the raw materials are distributed in the sinter bed in the form of a meso-mosaic (Otomo, 2005). The concept of a practical system is shown in figure 5.26

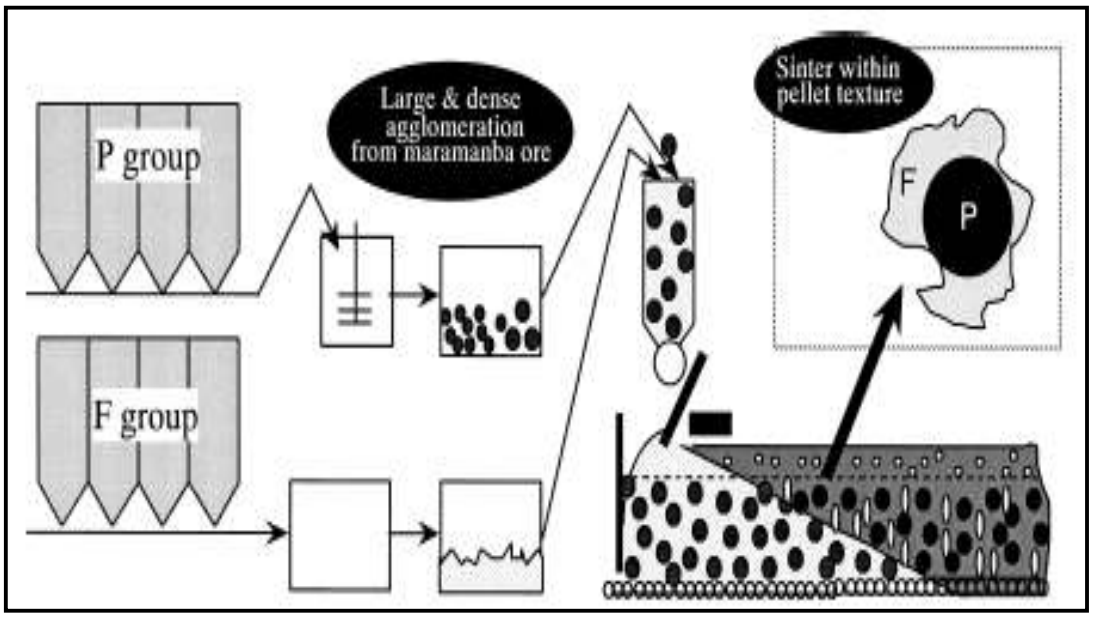


Figure 5.26 Image example of MEBIOS (Mosaic EnBedding Iron Ore Sintering) process (Kasai et al., 2005).

According to a study by Otomo & Takasaki (2001), the pores produced by thermal decomposition of Marra Mamba are mostly micropores measuring 20 μm or less, and the strengthening mentioned in the study by Sasaki et al. (2005) has little effect. Also, the micro-pores of 10 μm or less reported by Sasaki et al. (1996) do not readily combine regardless of the melt viscosity or liquid phase ratio, unlike the macro pores, and are

retained in the texture of the sinter. On the other hand, according to Tsukihashi et al. (2001)'s increasing the content of CaO or FeO is an effective way of reducing the solid phase ratio of the melt and changing pores into voids. As a bed structure column assuming a blend with a large amount of limonite, Otomo (2001)'s study showed that there must be at least 20% or so of fine grains that do not assimilate, and that it is important to maintain a solid phase ratio by mixing together fine and rough grain sizes so that thermal cracks do not readily form in order to maintain the solid phase ratio.

The raw material which is assumed to contain a large proportion of limonite is granulated in two separate blends and a bed of raw material is formed by distributing the two separate raw material beds in the form of a mosaic. One of these is a raw material bed that promotes smelting assimilation, which is based on pisolitic limonite and blended with a reducing agent (coke, scale) and CaO. The other is a raw material bed that suppresses smelting assimilation, which is based on Marra Mamba and consists of fine and coarsely granulated material (Otomo, 2005).

5.8.7.4 Sinter Composition Designs for Improved High Temperature Properties

According to a study by Nakamoto *et al.* (2004) the ventilation properties and reducibility in the high temperature part (welding zone) inside a blast furnace contribute to the quantity of melt produced and to the permeation blockage state, and it is important to reduce the quantity of melt and alleviate blockages by reducing the permeation forces.

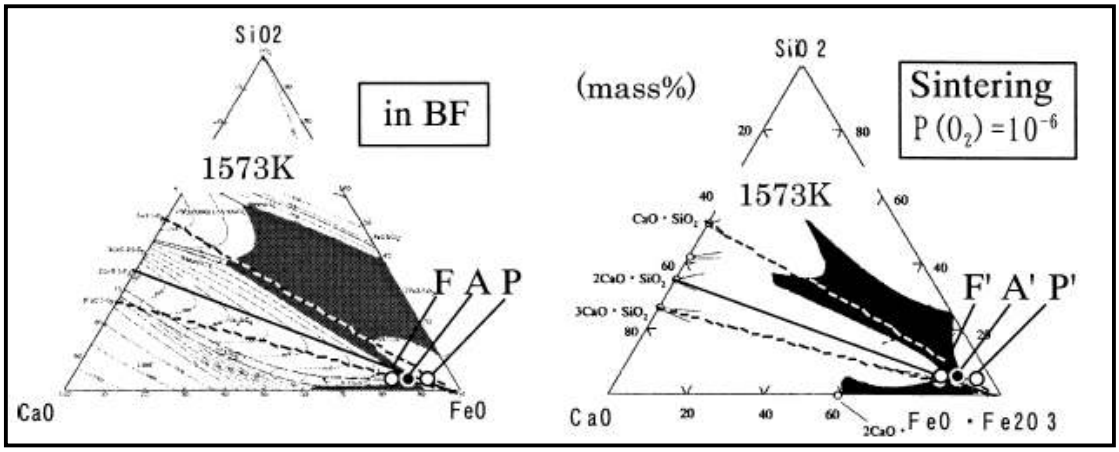


Figure 5.27 Liquid phase at CaO-SiO2-FeOx system ratio in BF and sintering process condition. (Left: Phillips, 1959) right: by Tukihashi, 2005).

The diagram on the left side of figure 5.27 shows the equilibrium liquid regions of the CaO-SiO_2-FeO system at 1 300°C, which represents the slag composition of sinter in a blast furnace in a high-temperature reducing state. The sinter composition in which iron oxide is reduced to FeO (point A) lies in the liquid phase region, while the liquid phase ratio is smaller in a low slag (CaO, SiO_2) composition (point P) where the high-temperature ventilation properties and reducibility are improved. Furthermore, as mentioned above, the increase of micro-pores in blends with a large limonite content allows the high-temperature reducibility to be further improved. This requires the use of a Marra Mamba ore with a low SiO_2 content rather than a pisolitic ore with a high SiO_2 content (Sasaki, & Wan, 2001).

Next, we will consider applications of the MEBOIS method. The MEBIOS method is characterized in that assimilation between the two separated raw material systems is suppressed. Accordingly, this method results in the formation of sinter textures corresponding to the two separate raw material compositions. In the diagram on the left side of figure 5.27, examples of the two separate conjugate compositions (points P and F) are shown together based on the sinter composition (point A). The sinter composition (point P) lies in a region where solid and liquid coexist, and the solid phase is FeO. If the state of reduction (i.e., the FeO component) is assumed to be constant, then the liquid phase ratio is determined by the CaO/SiO_2 ratio with the lowest liquid phase ratio occurring at CaO/SiO_2=1. When the liquid phase ratio is obtained by dividing the conjugation of point P at CaO/SiO_2=1 and point F at CaO/SiO_2=3, these points obey the relationship point A>point F>point P, and it is predicted that the liquid phase ratio will be reduced by separating the composition. Thus even when the overall compositions are identical, uneven distribution of the sinter composition can suppress melt formation inside the blast furnace, thereby improving the high-temperature ventilation and reducibility (Otomo, 2005).

5.8.7.5 Sinter Composition Designs for Improved Reduction Properties

Accoring to the liquid phase diagram produced by Tsukihashi et al. (2005) (see figure 5.27, right side), reducing the CaO and SiO_2 constituents in the sintered ore results in a smaller liquid phase ratio and lower strength in the sinter production process, but it is thought that the liquid phase ratio can be increased and the strength can be maintained by

reducing the oxygen partial pressure. According to a study by Nakazato et al. (2001) into the mineral texture analysis of sinter with a low oxygen partial pressure (i.e., a high FeO content), silicate slag minerals ($2FeO.SiO_2$ and $FeO.CaO.SiO_2$) produce textures that are very difficult to reduce in the low-temperature reducibility evaluation of mineral textures, and are regarded as textures whose generation should be suppressed. Accordingly, if reducing the oxygen partial pressure and slag content of sinter (i.e., increasing the FeO content and reducing the SiO_2 content) suppresses the formation of slag minerals that are difficult to reduce and increases the $Fe3O_4$ mineral content, then it should effectively improve the strength and reducibility. If it can be assumed that blast furnace slag design is feasible, then according to crystal analysis performed by Sugiyama et al. (2000), increasing the MgO constituents, Al_2O_3 constituents and the CaO/SiO_2 ratio should be an effective way of increasing the amount of SiO_2 in solid solution in calcium ferrite minerals with multiple constituents (SFCAM), reducing the levels of silicate slag minerals ($2FeO.SiO_2$ and $FeO.CaO.SiO_2$), and improving the reducibility (Sasaki & Wan, 2001).

5.8.8 Desired Microstructure in Sinter

Firstly, the formation of magnetite should be prevented to produce high reductanable and high resistant sinter in high reductanbality and dispersion as a good quantity sinter. In the other word, the temperature of sinter bed should be more than 1300 ^oC. Holding the maximum temperature around 1250 ^oC to be formed ascilur calcium ferrite and granuler hematite as a stable should be controlled frequently (Otomo, 2005).

The researchers who study in the relationship between morphology of calcium ferrite and T^{max} determine that structure changes from needle form to wide form like plate. Besides, they determine that when the temperature increases above 1250 ^oC, Al_2O_3 and $Si0_2$ take the place of Fe_2O_3 and volume of calcium ferrite decreases. As another from that, they also determine that calsiım ferrite shows informatical differences in between different form. Structure of calcium ferrite is seen in lots of sinter association. While asicular occurs next to reducted hematite, primary calcium ferrite occurs next to pieces of CaO. Calcium ferrite ($CaO.2Fe_2O_3$) is connected with complex quartet or SFCA (silico ferrite calcium and aluminum) (Barnaba, 1985).

CHAPTER SIX
EVALUATION PROPERTIES OF SINTER

6.1 Sinter Trommel Test

Firstly measure the resistance to degradation by abrasion of sinter that is occurs in sinter flat. After that define the resistance to degradation by impact, therefore take a sample as underscreen and chemical analyses, and make analyses to sieve and prepare the chemical sample. Also make the drum strength test, which is suitable to ASTM E279, and afterwards define the resistance to degradation by abrasion of domestic and import pellet (Powers, 1952).

Firstly take the sample by half automatic machine from the conveyor, which carries sinter, and drop to the hand truck every 1.5 hours. Secondly it is divided to two by sample quadruplicating methods. Figure 6.1 shows a typical ISO Drum. Take a sample from one. The other is waiting for underscreen analyses and trommel test. Second application is occur when the second sample is taken (International Organizationfor Standardization 3271 [ISO], 2007).

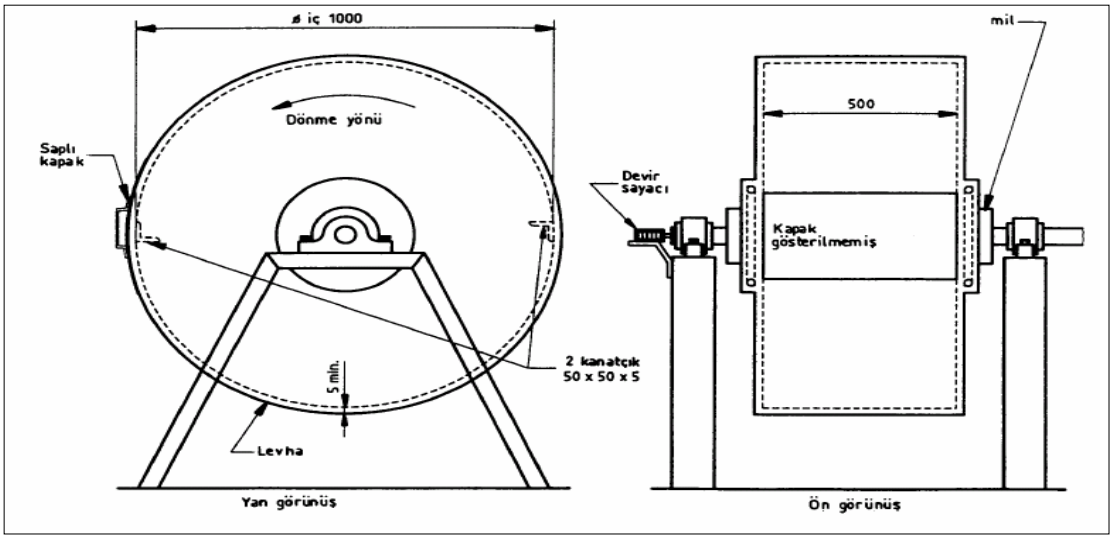


Figure 6.1. ISO Drum (Günaydin, 2002).

Inner Diameter: 1000 mm.

Width: 500 mm.

Inner Cylinder Dimensions: 50x50x5

Rotation Speed: 25 rpm

Supply Sizes: +6.30 mm - 40.0 mm.

Before the taking four sample from the each shift endings (approximately 140 gr), making underscreen test. Then sieve by 50 mm, -10mm, -6mm underscreen on jerking sieve. After that result of the analyses are used for drum test. Drum test is making as shown in figure as below (ISO 3271, 2007):

Calculation of Sinter Trommel, amount of fine at sinter:

1) TM= (US/M)*100 (6.1)

TM=Amounts of Fines %

US=Fine Sinter the 6.35mm underscreen (kg)

M=Total mass of sinter which is sieved (kg) (approximately 60 kg)

2) SM= (OS/T)*100 (6.2)

SM= the resistance to degradation by impact

OS=mass that the +6.35mm shorts after the drum. (11.3kg)

T=mass for drum test. (11.3kg)

3) TI= (US/T)*100 (6.3)

TI= the resistance to degradation by impact

US=Mass that the 0.6mm underscreen after the drum. (11.3kg)

T= mass for drum test. (11.3kg)

10 mm underscreen can use instead of 9.52 mm underscreen.

6.2 Technological Test

Taken a Sample: Prepare a sample for technological test from pellet, lump, and sinter sample which must be suitable to ISO 4696-1, ISO 7992, ISO 13930 and ISO 4698 standards.

Application: If Sinter Plant wants to technological test, then take the 50 kg sample. Sieved at 10 mm and 12.5 mm underscreen. Sample that is stay over the 12.5 mm is broken for passing through the 16 mm underscreen. After that take a broken sample and sieve them at 10 mm and 12.5 mm. underscreens again. Than mixed the two sample. Take 5kg sample for RUL (Reduction Under Load) test, 2 kg sample for LTD (Low Temperature Disintegration) test, and 2 kg sample for RDI (Reduction Degradation Index) by separator (by quadruplicating method). Finally send to test room. Figure 6.2 indicates technological test sample preparation (International Organizationfor Standardization 7992 [I.S.O], 2007).

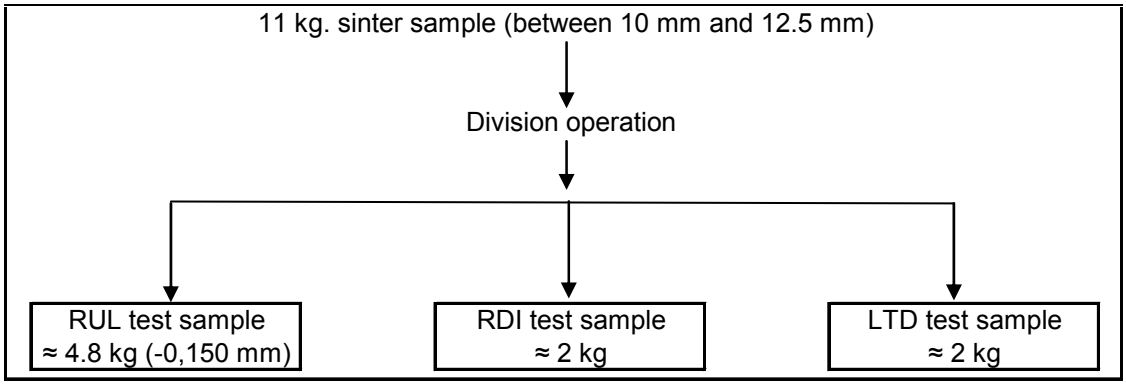


Figure 6.2 Technological test sample preparation.

Firstly put a stick to the sample, and dry 2 hours at 105 ^{o}C. Secondly wash the sample for Free Swelling Test, and than clean the fine, which is, cover the sample, and dry them. And finally make tests that are LTD, RDI, Free Swelling Test, RUL Test.

Preparing the Chemical Samples: Pellet, lump ore, mixer materials and sinter iron ores' samples decrease by quadruplicating method at disk grinder. It become under a 160 micron. Then it stuffed by 100 gr of them in labelled envelope. Finally it sends to spectral laboratory.

6.2.1 Determination of Reduction Properties under Load (RUL)

This method is based on the following steps (I.S.O 7992, 2007).

a) Reduction of a bed of the test portion (test bed) with specified size by a carbon/hydrogen gas mixture at a temperature of 1050 ºC, whilst applying a static load.

b) Monitoring, at regular intervals, the loss in mass of test portion, and the differential gas pressure across the test bed and the height of the test bed.

c) Determination of the differential pressure and the change in the height of the test bed at an 80 % degree of reduction.

The reducing gas shall consist of:

CO 40 % (V/V) ± 0.5 % (V/V)

H_2 2.0 % (V/V) ± 0.5 % (V/V)

N_2 58 % (V/V) ± 0.5 % (V/V)

The reducing gas flow rate shall, during the test period, be maintained at 83 l/min ± 1 l/min. The reducing gas should be preheated before entering the test portion at 1050 ºC ± 10 ºC during the entire test period. Weigh, to the nearest 1 g, approximately 1200 g of the test sample. During the entire test period, the test portion shall be under a constant load of 50 kPa measured at the surface of the bed. The pellets shall have the particle size between 10.0 mm and 12.5 mm. The sinter sample shall have the particle size between 10.0 mm and 12.5 mm (I.S.O 7992, 2007).

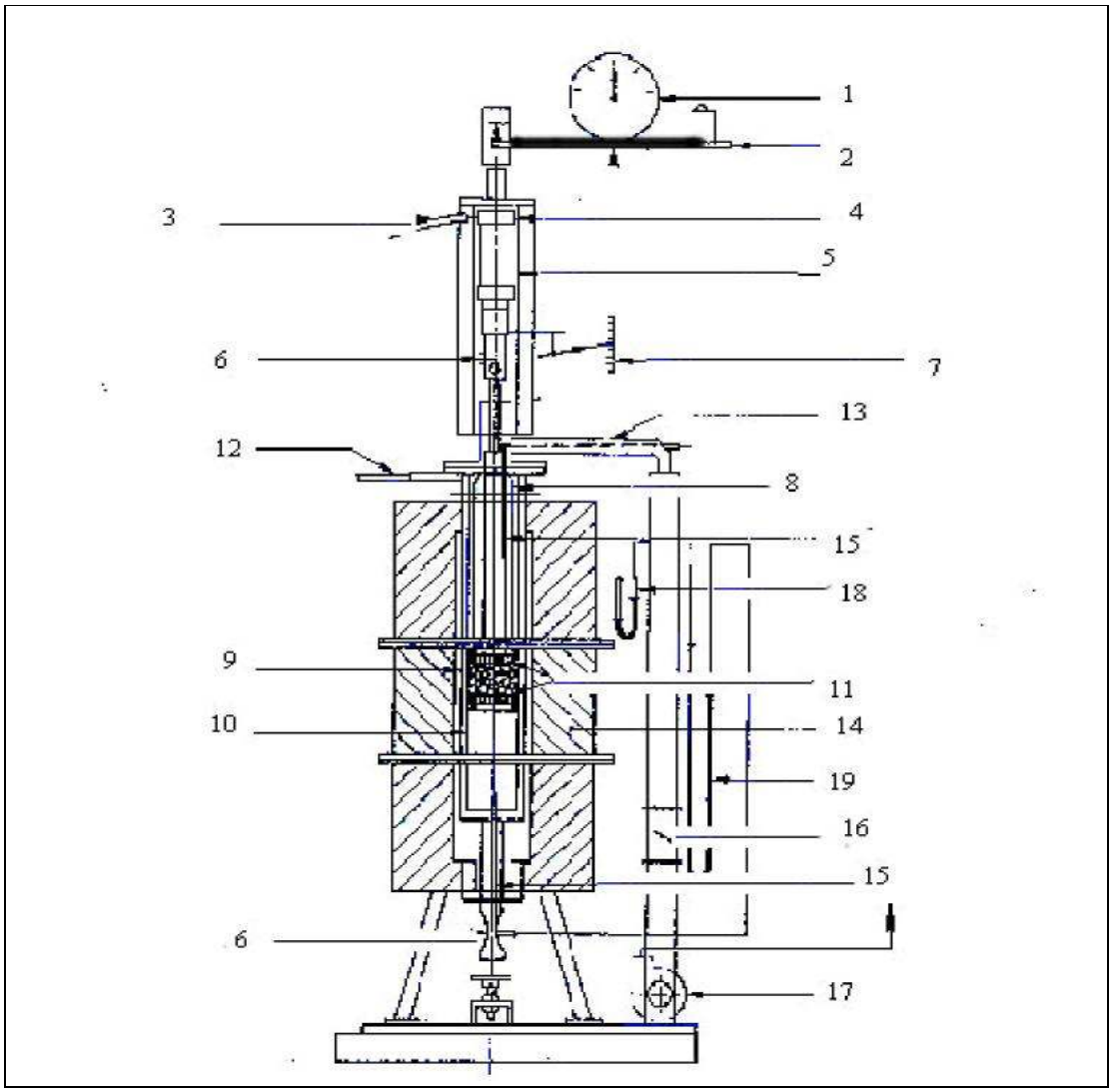
Place a double layer bed of porcelain pellets with a size between 10.0mm and 12.5 mm on a perforated plate in the reduction tube, resistant to deformation, made of non-scaling, heat resistant metal to withstand temperatures of 1050 ºC with a diameter of 125 mm ± 1 mm, in order to achieve uniform gas flow. After leveling, measure the height of the top surface of the porcelain layer. Place the test portion on the bed of porcelain pellets. After leveling, measure the height of the top surface of the bed of the test portion (test bed). Place a further double layer of porcelain pellets

with a particle size between 10.0 mm and 12.5 mm on the test bed. After leveling, measure the height of the top of the surface of the porcelain layer again.

Close the top of the reduction tube by connecting the heat assembly containing the loading device to the reduction tube. Insert the reduction tube assembly into the furnace, having a heating capacity sufficient to maintain the entire test portion and the gas entering the bed 1050 ºC ± 10 ºC, and suspend it by the weighing device, capable of weighing the load to a sensitivity of 1g, centrally, ensuring that there is no contact with the furnace or heating elements. Figure 6.3 shows Reduction under load testing device.

Figure 6.3 A picture of reduction under load testing device.

Connect the thermocouple and the measurement devices for the differential pressure and for the change in the height of the test bed. Connect the loading device and apply a load of 50 kPa ± 2 kPa. Figure 6.4 shows apparatus for determining reduction properties under load for I.S.O. 7992 (I.S.O 7992, 2007).



1-Scale	6-Thermocouple exit	11-Upper and lower perforated plates comprising the test portion	16-Throttle valve
2-Balance	7-Linear scale	12-Reducing gas inlet	17-Waste gas fan
3-Compressed air inlet	8-Loading ram	13-Reducing gas outlet	18-Suction gauge
4-Pressure cylinder	9-Outer reduction tube	14-Main furnace body	19-Differential pas pressure manometer
5-Frame for pressure cylinder	10-Inner reduction tube	15-Differential gas pressure upper and lower probes	

Figure 6.4 A overview of apparatus for determining reduction properties under load for I.S.O. 7992

Pass inert gas through the reduction tube at a flow rate of 50 l/min ± 1 l/min. When the temperature of the test portion approaches 1050 ºC, increase the flow rate

to 83 l/min. Continue heating until the mass of the test portion is constant and the temperature is constant at 1050 ºC ± 10 ºC.

Introduce the reducing gas to replace the inert gas at a flow rate of 83 l/min. Measure and record the differential gas pressure across the test bed, the height of the test bed and the mass of the test portion at least every 5 min for the first 30 min and thereafter at 10 min intervals.

Terminate the reduction when the oxygen loss reaches 80 % of the value theoretically expected with an assumption that the entire test portion is pure iron (III) oxide. Calculate the degree of reduction, R_t , relative to the iron (III) state after t min, expressed as a percentage, using the following equation (I.S.O 7992, 2007):

$$R_t = \left[\frac{0.111 * w(FeO)}{0.430 * w(Fe)} + \left[\frac{m_1 - m_t}{m_0 * 0.430 * (Fe)} * 100 \right] \right] * 100 . \qquad (6.4)$$

where;

m_0 is the mass, in grams, of the test portion.

m_1 is the mass, in grams, of the test portion immediately before starting reduction.

m_t is the mass, in grams, of the test portion after reduction time t.

w(Fe) is the total iron content, expressed as a percentage by mass, of the test portion. w(FeO) is the iron (II) oxide content as a percentage by mass, of the test sample prior to the test and is calculated from the iron (II) content by multiplying by a factor of 1.286, determine in accordance with ISO 9035.

Using the reduction curve, R_t versus time, and the values of the differential gas pressure (ΔP), in kPa, obtained at different times (Δt), plot the differential gas pressure against the degree of reduction. Read the differential pressure (ΔP80) corresponding to an 80 % degree of reduction.

Using the height measurements taken during the test, calculate the percentage change in the height of the test bed (Δh) obtained at different times. Plot the percentage change in the height of the test bed against the degree of reduction using the reduction curve. Read the percentage change in the height of the test bed (Δh_{80}) corresponding to an 80 % degree of reduction (I.S.O 7992, 2007).

6.2.2 Dynamic Test for Low Temperature Reduction Disintegration (LTD)

A test portion with a specified size range is reduced in a rotating tube at a temperature of 500 ºC using reducing gas consisting of carbon monoxide (CO), carbon dioxide (CO_2), hydrogen (H_2) and nitrogen (N_2) International Organization for Standardization 13930 [ISO], 2007).

The reducing gas shall consist of:

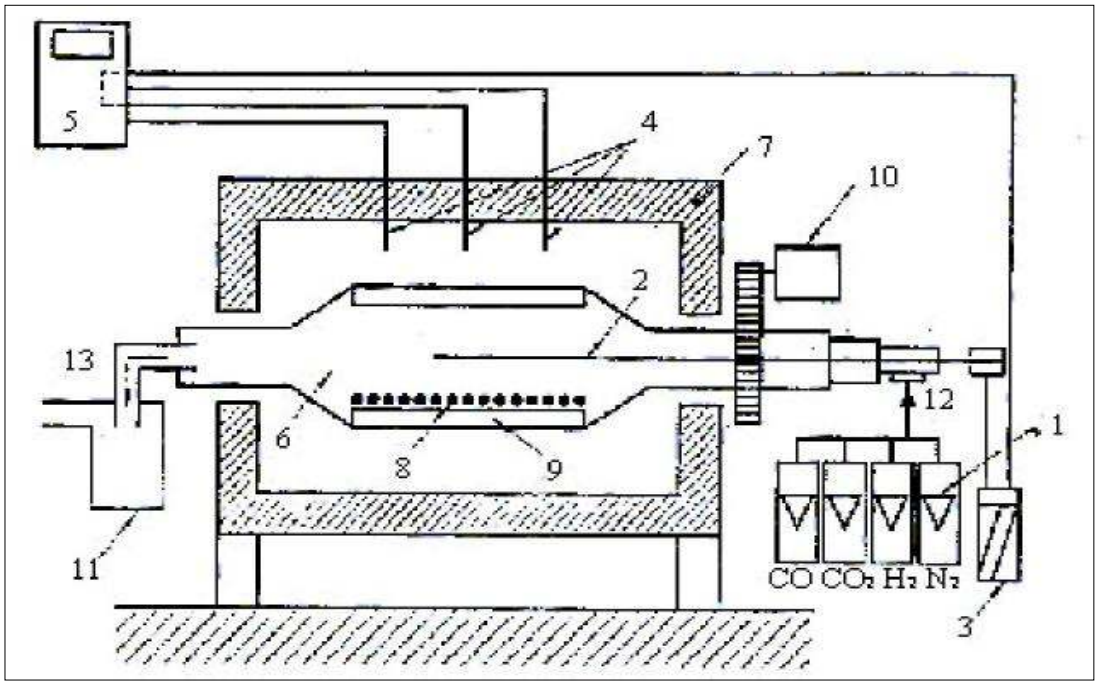
CO 20 % (V/V) ± 0.5 % (V/V)

$CO_2$20 % (V/V) ± 0.5 % (V/V)

H_2 2 % (V/V) ± 0.2 % (V/V)

N_2 58 % (V/V) ± 1 % (V/V)

The reducing gas flow rate shall, during the test period, be maintained at 20 l/min ± 0.5 l/min. The reducing gas should be preheated before entering the test portion at 500 ºC ± 5 ºC during the entire test period. Weigh, to the nearest 1 gr, approximately 500 g (± 1 particle) of the test sample (mass m_o). Pellets in the size range of 10.0 mm to 12.5 mm and 12.5 mm to 16.0 mm shall be used. Lump ores in the size range of 10.0 mm to 12.5 mm shall be used. Figure 6.5 low temperature disintegration test apparatus for I.S.O. 13930 (ISO 13930, 2007).



1-Gas flowmeters	8-Test portion
2-Thermocouple for measuring reduction tube temperature	9-Lifter
3-Temperature recorder	10-Electric motor
4-Thermocouple for measuring furnace temperature	11-Dust collector
5-Temperature control unit	12-Gas in
6-Reduction tube, 540 mm * 150 mm, four lifters	13-Gas out
7-Furnace	

Figure 6.5 Low temperature disintegration test apparatus for I.S.O. 13930

Place the test portion in the reduction tube, made of non-scaling, heat resistant metal to withstand temperatures of higher than 500 ºC. Insert the reduction tube into the furnace and connect the thermocouple and the gas flow system to the reduction tube. Commence rotation of the reduction tube at 10 rpm ± 0.2 rpm.

Replace the air in the tube with inert gas. Heat the test portion and, while heating, pass a flow of inert gas through the reduction tube at a flow rate of approximately 20

l/min. Bring the temperature inside the reduction tube to 500 ºC within 45 min and stabilize the temperature within the next 15 min. If this requirement is not met, will discontinue the test and start a new one (ISO 13930, 2007).

After a total time of 60 min, start the reduction. Introduce the reducing gas at a flow rate of 20 l/min ± 1 l/min to replace the inert gas and to reduce the test portion.

After 60 min reduction time, stop the flow of the reducing gas, stop the rotation of the reduction tube and cool the test portion to a temperature below 350 ºC in the reduction tube under a flow (20 l/min) of inert gas. Then lift the reduction tube from the furnace and cool further, still under the flow of inert gas. Figure 6.6 demonstrates low temperature reduction disintegration testing device.

Figure 6.6 A picture of low temperature reduction disintegration testing device.

At a temperature below 100 ºC, remove all the material from the reduction tube. Add the dust trapped in the dust collector to this material. Determine the mass (m_o) to the nearest 1 g. Sieve mechanically on 6.3 mm, 3.15 mm and 0.5 mm sieves. Determine and record the mass of each fraction retained on the 6.3 mm (m_1), 3.15 mm (m_2) and 0.5 mm (m_3) sieve. Material loss during sieving shall be considered to be -0.5 mm (ISO13930, 2007).

The low temperature disintegration indices $LTD_{+6.3}$, $LTD_{-3.15}$ and $LTD_{-0.5}$, expressed as a percentage by mass, shall be calculated to the first decimal place from the following equations (ISO 13930, 2007):

$$LTD_{+63} = \frac{m_1}{m_0} * 100 \tag{6.5}$$

$$LTD_{-3.15} = \frac{m_0 - (m_1 + m_2)}{m_0} * 100 \tag{6.6}$$

$$LTD_{-0.5} = \frac{m_0 - (m_1 + m_2 + m_3)}{m_0} * 100 \tag{6.7}$$

Where;

m_o is the mass, in grams, of the test portion after reduction including the dust trapped in the dust collector.

m_1 is the mass, in grams, of oversize fraction retained on the 6.30 mm sieve.

m_2 is the mass, in grams, of oversize fraction retained on the 3.15 mm sieve.

m_3 is the mass, in grams, of oversize fraction retained on the 0.5 mm sieve.

LTD_{+63} the disintegration index expressed in terms of the mass % of the +6.3 mm sieve fraction (the so-called disintegration strength).

$LTD_{3.15}$ the disintegration index expressed in terms of the mass % of the -3.15 mm sieve fraction (the so-called disintegration index).

$LTD_{0.5}$ the disintegration index expressed in terms of the mass % of the -0.5 mm sieve fraction (the so-called disintegration abrasion).

6.2.3 Reduction Degradation Index Static (RDI)

A test portion with a specified size range is subjected to static reduction at a temperature range of 500 ºC using reducing gas consisting of CO, CO_2, H_2 and N_2 (International Organization for Standardization 4696-1 [ISO], 1998).

After 1 hour reduction time, the test portion is cooled to a temperature below 100 ºC and tumbled by using a small tumbler drum for 300 revolutions in total. It is then sieved with test sieves having square mesh apertures of 6.30 mm, 3.15 mm and 500 µm.

The reduction disintegration index (RDI) is calculated as a quantitative measure of the degree of disintegration of an iron ore that has been reduced and tumbled: the percentage masses of material greater than 6.30 mm, less than 3.15 mm and less than 500 µm, respectively, are related to the total mass of the test portion after reduction and before tumbling. The reducing gas shall consist of (ISO 4696-1, 2007).

CO 20 % (V/V) ± 0.5 (V/V)

CO_2 20 % (V/V) ± 0.5 (V/V)

H_2 2.0 % (V/V) ± 0.5 (V/V)

N_2 58 % (V/V) ± 0.5 (V/V)

The reducing- gas flow rate shall, during the test period, be maintained at 20 l/min ± 1 l/min. The reducing gas shall be preheated before entering the test portion to maintain the test portion at 500 ºC ± 5 ºC during the entire test period. Weigh, to the nearest 0.1 g, approximately 500 g (± 1 particle) of the test sample. The pellets in the size range of 10.0 mm to 12.5 mm shall be used for the test. As same as pellets, sinters and ores in the size range of 10.0 mm to 12.5 mm shall be used for the test. Figure 6.7 indicates arrangement of a test unit for I.S.O. 4696-1

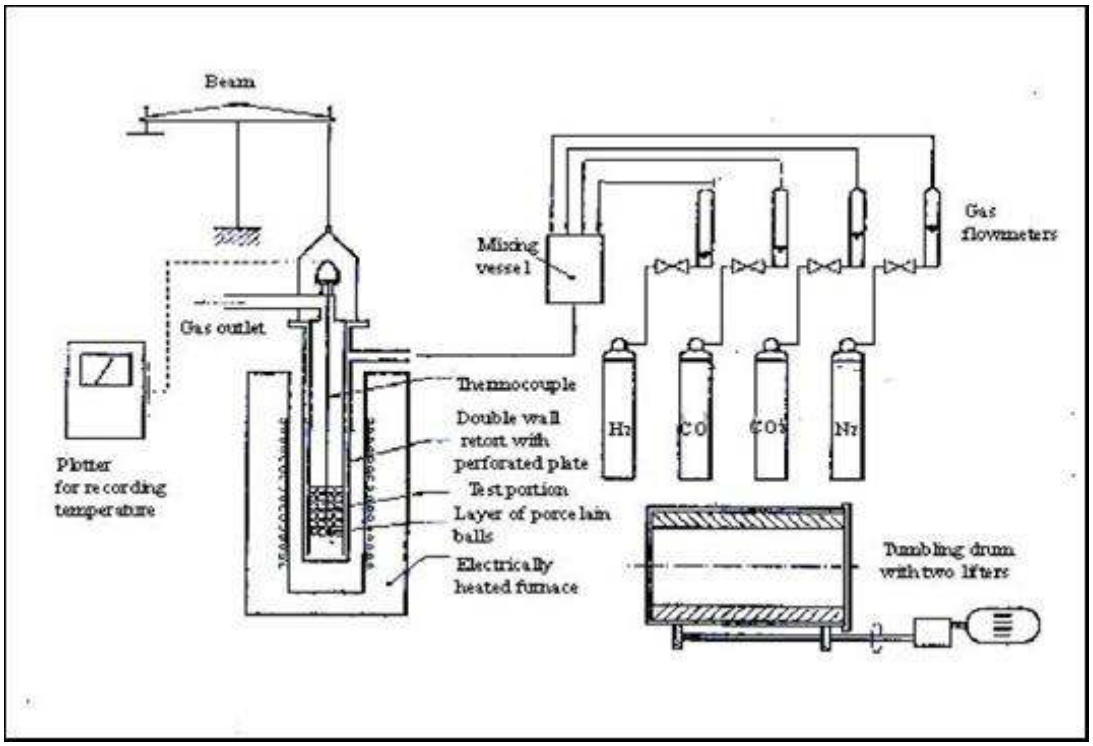


Figure 6.7 Arrangement of a test unit for I.S.O. 4696-1

Place a double layer bed of porcelain pellets, having a size range of 10.0 mm to 12.5 mm, on the perforated plate. Place the test portion on the porcelain pellets in the reduction tube, made of nonscalling, heat resistant metal to withstand temperatures of higher than 600 ºC with a diameter of 75 mm ± 1 mm, and level the surface. Place the thermocouple in the centre of the test portion. Close the top of the reduction tube. Then insert the reduction tube into the furnace, having a heating capacity sufficient to maintain the entire test portion and the gas entering the bed 500 ºC, and attach it to the weighing device of appropriate capacity and accuracy, to 0.1 g, ensuring that there is no contact with the furnace or heating elements (ISO 4696-1, 2007).

Replace the air in the tube with inert gas. Heat the test portion and, while heating, pass a flow of inert gas through the test portion at a flow rate of approximately 20 l/min. Continue the heating, while passing inert gas, until the test portion reaches the test temperature of 500 ºC ± 5 ºC.

Introduce the reducing gas at a flow rate of 20 l/min ± 1 l/min to replace the inert gas and to reduce the test portion. Continue the reduction with the reducing gas for 1 h. Figure 6.8 demonstrates reduction degradation index of static.

Figure 6.8 RDI testing device.

After 1 h reduction, stop the flow of the reducing gas and cool the test portion to a temperature below 100 ºC in the reduction tube under a flow of inert gas.

Remove the test portion carefully from the reduction tube, determine the mass (m0) and place it in the tumbler drum. Fasten the lid tightly and rotate the drum for a total of 300 revolutions at a rate of 30 rev/min ± 1 rev/min.

Remove all material from the drum, determine the mass and hand sieve with care on 6.30 mm, 3.15 mm and 500 µm sieves. Determine and record the mass of each fraction retained on the 6.30 mm (m_1), 3.15 mm (m_2) and 500 µm (m_3) sieves. Material lost during tumbling and sieving shall be considered to be less than 500 µm.

The reduction-disintegration index RDI, expressed as a percentage by mass, is calculated from the following equations (ISO 4696-1, 2007):

$$RDI_{+63} = \frac{m_1}{m_0} * 100 \qquad (6.8)$$

$$RDI_{-3.15} = \frac{m_0 - (m_1 + m_2)}{m_0} * 100 \qquad (6.9)$$

$$RDI_{-0.5} = \frac{m_0 - (m_1 + m_2 + m_3)}{m_0} * 100 \qquad (6.10)$$

m_0 is the mass, in grams, of the test portion after reduction and before tumbling.

m_1 is the mass, in grams, of the oversize fraction retained on the 6.30 mm sieve.

m_2 is the mass, in grams, of the oversize fraction retained on the 3.15 mm sieve.

m_3 is the mass, in grams, of the oversize fraction retained on the 500 µm sieve.

6.3 Specifications

6.3.1 Sinter

a. Chemical Analyze: (Erdemir, 2002)

Fe: %58	SiO_2: %5	Al_2O_3: %1, 6	CaO: %10
MgO: %1, 2	Na_2O: %0, 03	K_2O: %0, 12	Mn: %1
S: %0, 02			

b. Underscreen Analyze: (Erdemir, 2002)

+ 50 mm		Max. ----
- 50 mm	+ 9, 52 mm	Min. 70
- 9, 52 mm	+ 6, 35 mm	Max. 20
- 6, 35 mm		Max. 10

c. Drum Test:

+ 6, 35 mm	Min. 75
- 0, 6 mm	Max. 10

d. Property of Low Temperature Breakdown: (Erdemir, 2002)

+ 6, 35 mm	10-15
- 6, 35 mm +3, 15 mm	Min. 30
- 3, 15 mm + 0, 5 mm	35-40
- 0, 5 mm	Max. 5

e. Property of Reduction Degradation: (Erdemir, 2002)

+ 6, 35 mm	20-25
- 6, 35 mm +3, 15 mm	Min. 40
- 3, 15 mm + 0, 5 mm	30-35
- 0, 5 mm	Max. 5

f. Property of Reduction under a Load: (Erdemir, 2002)

R_{40} : 1, 80

ΔP : 1

ΔH : 8-10

6.3.2 Lump Iron Ore

a. Chemical Analyze:

Fe: %64	SiO_2: %5	Al_2O_3: %1	CaO: %0, 5
MgO: %0, 5	Na_2O: %0, 05	K_2O: %0, 05	Mn: %1, 5
S: %0, 02			

b. Underscreen Analyze: (Erdemir, 2002)

+ 25, 4 mm		Max. 10
- 25, 4 mm	+ 9, 52 mm	Max. 80
- 9, 52 mm		Max. 10

c. Drum Test:

+ 6, 35 mm	Min. 80
- 0, 6 mm	Max. 10

d. Property of Low Temperature Breakdown: (Erdemir, 2002)

+ 6, 35 mm	Min. 67
- 0, 5 mm	Max. 7

e. Property of Reduction Degradation: (Erdemir, 2002)

+ 6, 35 mm	Min. 70
- 6, 35 mm +3, 15 mm	Max. 10
- 3, 15 mm + 0, 5 mm	Max. 5
- 0, 5 mm	Max. 10

f. Property of Reduction under a Load: (Erdemir, 2002)

R_{40} : 0, 80

ΔP : 10

ΔH : 10

6.3.3 Pellet

a. Chemical Analyze: (Erdemir, 2002)

Fe: %66	SiO_2: %5	Al_2O_3: %1	CaO: %2, 5
MgO: %1, 5	Na_2O: %0, 05	K_2O: %0, 05	Mn: %1
S: %0, 02			

b. Underscreen Analyze: (Erdemir, 2002)

+ 16 mm		Max. 5
- 16 mm	+ 9, 52 mm	Min. 85
- 9, 52 mm	+ 6, 35 mm	Max. 7
- 6, 35 mm		Max. 3

c. Drum Test:

+ 6, 5 mm	Min. 95
- 0, 6 mm	Max. 5

d. Property of Low Temperature Breakdown: (Erdemir, 2002)

+ 6, 35 mm	Min. 84
- 0, 5 mm	Max. 3

e. Property of Reduction Degradation: (Erdemir, 2002)

+ 6, 35 mm	Min. 90
- 6, 35 mm +3, 15 mm	2-3
- 3, 15 mm + 0, 5 mm	1-2
- 0, 5 mm	3-5

CHAPTER SEVEN
EXPERIMENTAL STUDIES

7.1 Experimental Works

Iskenderun Iron and Steel Works in the blast furnace are used in the production of liquid crude iron sintering plants produce their own. Also by doing necessary tests to produce their sintering properties are determined.

This work; is production sintering technology in Isdemir and usage data, and also produced with these sinters in Isdemir was prepared by the results of various experiments. The effect of sinter production efficiency, mineralogical and mechanical properties of sinter product, and then reducibility are studied. Finally the sintering furnace is used in the production of liquid crude iron and the effects of productivity investigated.

In addition, the results of various experiments and analysis data were prepared with production knowledge of the sinter technology in Isdemir. Except that sinter blend is consisted of inevitable material, ferrous materials as flue dust, scale and flux constructive materials as limestone, dunite and BOF slag in different proportions performed with additional in sinter blend. The main subjects of study in this thesis are sinter applications at ISDEMIR, R & D Laboratory test work with MECC Corporation, statistic quality control of sinter samples and studies on sinter behaviors with mineralogical analysis.

7.2 Sintering applications at ISDEMIR

Sinter plant in the Isdemir; in sinter production; sinter dust ore, sinter return in under screen dimensions, coke dust and essential components such as flux material also flue dust, scale as well as other ferrous raw materials were used. These results were obtained experiment from production information and also, during sinter production and usage data of sinter technology at Isdemir. The sinter experiments and analysis were made with 40 different sinter productions.

7.2.1 General İnformation about at Sinter Plants in Isdemir

ISDEMIR Iron and Steel Plant, Turkey is located at Iskenderun, Turkey and at approximately 100 km east of Adana, Turkey. ISDEMIR Iron and Steel Plant was built in early 1970s of twentieth century. 3*75m^2 sinter machines were put into operation in 1976 and the fourth 75m^2 sinter machine was commissioned in 1984 when the annual skip sinter output was about 2.5 Mt, pig iron about 2.5 Mt and steel about 2 Mt. Type of installation is Dwight-Hoyd type sintering machine. Production conditions are given below.

Main Data of ISDEMIR Sinter Plant:

Number of strands:	4
Start-up date:	3 strands May 1976, 1 line May 1984
Capacity (total):	2.520.000 tons / year (skip sinter)
Maximum production:	2.467.133 tons (skip sinter) (in 1990)
Grate area:	75 m² * 4 = 300 m²
Width of grate:	2.5 m
Length of Sintering Area:	30 m
Grate speed:	2 – 4.5 m/min
Ignition temperature:	1.100 °C (average)
Cooler type:	Straight
Cooling area:	125 m²
Number of cooling fans:	8 (each strand)

Number of waste gas exhaust fans: 4 (one for each strand)

Exhaust fan capacity: 6.000 m³/min

Motor power exhaust fan: (each) 2.000 kW

Waste gas temperature: 100 – 120 °C

Suction at fan: (design) 1100 mm WG

At Sinter Machine wind boxes: 800 – 900 mm WG

The current production of Blast Furnace Sinter is close to 2.5 Mtpy.

Table 7.1 Capacity of proportioning equipment (Liuqing et al, 2005).

Material	No. per line	Discharge design (t/h)
Blended ore fines system	6	230
Limestone system	2	45
Dunite system	2	30
Coke breeze	3	30
Return Fines	2	230

7.2.2 Construction scale and product scheme

One set of 300 m^2 sinter machine shall be built with annual skip sinter output of 2.52 Mt. The sintering system shall be running continuously with 3 shifts per day and 8 hours per shift. The sinter machine shall work 330 days per year, i.e. 7920 hours, with availability of 90.4%. The product shall be cold sinter with temperature no greater than 100°C subject to sizing and suitable for blast furnace, size ranging from 5 to 50mm, content of <5mm no greater than 5% , content of >50mm upto 10% and tumbling strength (+6.3mm) ≥81%.

7.2.3 Technological Process

The technological process of the sinter plant shall include the whole technological process from incoming raw material to finished product output including preparation of flux and fuel, proportioning, primary mixing, secondary mixing, sintering, cooling, sizing and finished product output, referring to technological flow sheet.

7.2.4 Workshops

The sinter plant shall consist of the workshops and facilities, such as flux crushing and screening system, fuel crushing and screening system, proportioning building, primary mixing building, secondary mixing building, sintering building, main exhaust fan building, feed end ESP building, cold sinter screening building, power distribution room, discharge end ESP building and transfer station, referring to the general plan layout. Among the systems mentioned above, the flux crushing and screening system, fuel crushing system, proportioning and primary mixing system.

7.2.5 The Iron Bearing Raw Materials, Flux and Fuel Used in the Sinter Blend at Isdemir

7.2.5.1 Iron Bearing Material

The iron bearing raw material, flux, BF returns and fuel used in sinter plant. All these materials shall be proportioned as blended ore fines according to proportion in raw material stockyard and delivered to the proportioning building in sinter plant by belt conveyers. Iron ore in the sinter blend is composed of almost % 50 domestic and %50 foreign iron ore. Foreign iron ore is supplied from Brazil, Australia, and Canada etc. The domestic iron ore is supplied from Kayseri, Malatya and Sivas region etc. The typical chemical analysis of iron ore is given in table 7.2.

Table 7.2 The typical chemical analysis of iron ore.

Fe	58.94 %	FeO	9.14 %
SiO_2	4.96 %	Mn	0.51 %
Al_2O_3	1.65 %	P	0.04 %
S	0.181 %	K_2O	0.13 %
TiO_2	0.091 %	Cu	0.01 %
Zn	0.02 %	CaO	0.59 %
MgO	0.52 %	Fe / SiO_2	12.28 %

All iron bearing raw material including blended ore fines (import iron ore fines, domestic iron ore fines and mill scale), BF returns and varieties of dust shall be conveyed to the proportioning building by belt conveyers.

7.2.5.2 Coke Dust

The solid fuel used in the sintering process can be coke breeze, which can be conveyed to the fuel crushing building by belt conveyers. In Isdemir, the solid fuel in the sinter mix, 4% coke dust is added to the particle size of -3 mm. This coke dust, coke plants in integrated facilities is provided. Coke dust characteristics are given in the table 7.3.

Table 7.3. The typical chemical analysis and particle size distribution of coke breeze used in sinter blend.

Characteristic	%
Ash	15.40
Volatile	2.28
C	82.45
S	0.67
Moisture	14.0
+3.1 mm	18.35
+ 1 mm	56.79
-0.15 mm	10.72

7.2.5.3 Flux Material

The flux including dunite and limestone shall be conveyed to the flux crushing building by belt conveyers. Isdemir uses limestone, dunite and BOF as a flux material. Limestone and dunite are supplied of own Isdemir from limestone and dunite deposit. BOF slag is produced in the steel making plant and 4 % is added to the sinter blend. Slags in these steel making has typically contain 20–25% iron. In addition, the high content of 30–50% CaO, 3–10% MgO and MnO in steel slags can be used to substitute for a part of limestone, dolomite and manganese ore so as to reduce iron and steel making cost.

Limestone and other slags depending on the amount varying between 1.5 and 8% in the blend are involved. Dunite is not vary much from the 4% is added to the blend. BOF slag, the structure was calcined at up to 20% and slags where iron is used because it contains around 4% MnO.

The magnetic separation method is used for separating metallic iron and iron minerals from steel slag. Commonly used magnetic machines are the cross belt

magnetic separator, drum magnetic separator and magnetic pulley separator. In addition, the -1m fraction is added into sinter mix as a source of iron and fluxing materials. Based on results (Topkaya et al., 2004), the addition of -10 mm slag into the sinter mix up to 4% by weight of the iron-bearing burden has been decided by Isdemir officials to be used as a standard practice at Isdemir. According to a study in Isdemir 1.29 kg BOF slag was found to be equivalent to 1 kg slag. The average chemical composition and particle size distribution of limestone, dunite and BOF slag are given in table 7.4

Table 7.4 The average chemical analysis and particle size distribution of limestone, dunite and BOF slag used in sinter blend.

	% Limestone	Dunite	BOF slag
Fe	0.69	0.46	18-20
Fe_2O_3	0.17	-	10-14
SiO_2	1.10	32.37	4-9
Al_2O_3	0.18	0.96	2-3
CaO	51.90	9.11	40-50
MgO	1.82	32.26	2-4
S	0.043	0.065	0.2-0.4
K_2O	0.048	0.142	-
MnO	-	0.14	3-6
P_20_5	-	-	0.95 max
+3.1 mm	2.20	12.30	17.0
+1 mm	40.00	39.70	21.0
-0.15 mm	8.00	11.00	8.0

7.2.5.4 Flue Dust

Flue dust is accumulating in the output section of the dust in the blast furnace. Because of the iron and carbon containing mixture is added to the benefits. Around 1.5% is added to the blend. Chemical analysis of flue dust is given in table 7.5.

Table 7.5 The average chemical analysis of flue dust used in sinter blend.

	%
Fe	29.46
C	20.00
SiO_2	10.72
MnO	1.14
Al_2O_3	3.12
CaO	4.26
MgO	1.30
P	0.08
S	0.66
Na_2O	0.181
K_2O	0.16

7.2.5.5 Mill Scale

The scale consists of rolling mill and continuous casting. The iron containing up to 70% so, it is added in the sinter blend at a rate of 1.5-2%. Chemical composition is given in the table 7.6.

Table 7.6 The average chemical analysis of mill scale used in sinter blend.

Fe	68.01
FeO	53.84
SiO_2	0.45
Mn	1.26
P	0.01
S	0.03
Na_2O	0.16
K_2O	0.19
TiO_2	0.03
Cu	0.03
Zn	0.02

7.2.5.6 Return Sinter

Isdemir sinter plant, after the product obtained then sinter crushing and screening procedures are performed. Broken sinter, in the first +17 mm above the screen is sent to the blast furnace. Under screen -17 mm is eliminated two screens as sieve dimension 12mm and 7mm. A little of between -17+12mm particle size is sent to spread out on the grill as a bed material, the rest is sent to the blast furnace. Between -17+12 mm sent to blast furnace and under screen -7 mm came back dosage unit. Back in the sinter blend at Isdemir blend ratio is approximately 30-35%.

7.2.5.7 Water

The water required for the sinter blend ratio is 7-8%. Including taking into account the humidity in iron ore consist of sinter blend, during the preparation of sinter blend is added water in blend mixing average of 2 % ratio.

7.2.5.8 Product Scheme

The product shall be sized cold sinter for blast furnace with temperature no higher than 100 ^{o}C, <5mm content ranging 5~50mm no greater than 5%; > 50mm content ~10% and tumbling strength (+6.3mm)≥ 81%. For the chemical analysis of sinter, refer to Table 7.7. The typical sinter material balance was in table 7.8

Table 7.7 The average chemical analysis of typical sinter (Liuqing et al, 2005).

Composition	Fe	FeO	SiO_2	CaO	Al_2O_3	MgO	S	P	CaO/SiO_2
%	56	8	4.35	7.39	1.5	1.26	0.05	0.05	1.7

Table 7.8 The material balance of the typical sinter plant. (Liuqing et al, 2005).

Input		Output	
Description	%	Description	%
Blended ore fines	48.31	Sinter	55.44
BF returns	2.77	Loss	14.07
Solid fuel	3.83	Returns	22.17
Dolomite	1.73	Hearth layer	8.32
Limestone	5.54		
Water carried in by raw	4.35		
Newly added water	2.98		
Cold return fines and dust	22.17		
Hearth layer	8.32		
Total	100.00	Total	100.00

7.3 Sintering Applications at ISDEMIR

7.3.1 Examples of Sinter of Obtained with used Different Domestic and Foreign Iron Ore

The effect three different types of foreign iron ores and domestic iron ores usage in sinter making processes and effects of some sinter properties were examined. For this purpose preparation of sinter mix was done by considering the sintering conditions at İskenderun Iron and Steel Works (ISDEMIR). These are the effect of sinter production efficiency, mineralogical and mechanical properties of sinter product, and then reducibility effects, and finally the sintering furnace is used in the production of liquid crude iron and the effects of productivity investigated. The sinter experiments and analysis were made with 40 different sinter productions.

Table 7.9 Domestic and imported iron ore effect on the quality of sintering

Foreign Iron Ore(%)	Domestic Iron Ore(%)	Tromel (+6.35 mm) %	Recovery of Blending (TSS/TA)	Machine Productivity ($t.m^2/day$)	Machine Speed (m/min)	Basicity (CaO/SiO_2)	%Fe	Dust (-6mm) %	FeO	Al_2O_3	Coke Breeze %	Sinter Return Dust %
72.87	27.13	67.92	54.08	26.37	2.14	1.67	57.25	11.12		1.27	3.60	33.80
68.05	31.95	67.16	53.57	29.07	2.25	1.65	57.58	11.18		1.22	3.64	34.36
63.15	36.85	67.42	52.48	27.93	2.25	1.72	57.28	11.24		1.19	3.69	33.69
60.17	39.83	66.66	52.44	26.83	2.15	1.27	53.37	11.25	10.54	1.44	3.78	34.42
57.73	42.27	67.14	52.38	28.57	2.28	1.63	54.80	11.32		1.41	3.81	34.52
54.32	45.68	66.73	51.11	27.39	2.20	1.68	56.27	11.44		1.38	3.77	33.55
51.42	48.58	66.58	51.04	27.60	2.30	1.57	53.43	11.48		1.59	3.78	33.99
48.47	51.53	66.80	50.87	28.00	2.25	1.58	54.00	11.73		1.60	4.32	34.23
43.50	56.50	66.91	48.70	29.78	2.35	1.77	53.73	11.88		1.21	3.78	35.01
39.60	60.40	66.60	46.97	25.29	2.26	1.63	52.78	11.91	10.62	1.62	4.25	35.19
36.08	63.92	66.21	46.75	28.39	2.22	1.30	53.03	11.84	10.39	1.93	4.33	35.22
33.70	66.30	66.11	46.36	27.48	2.30	1.60	52.94	12.02		1.75	4.05	35.89

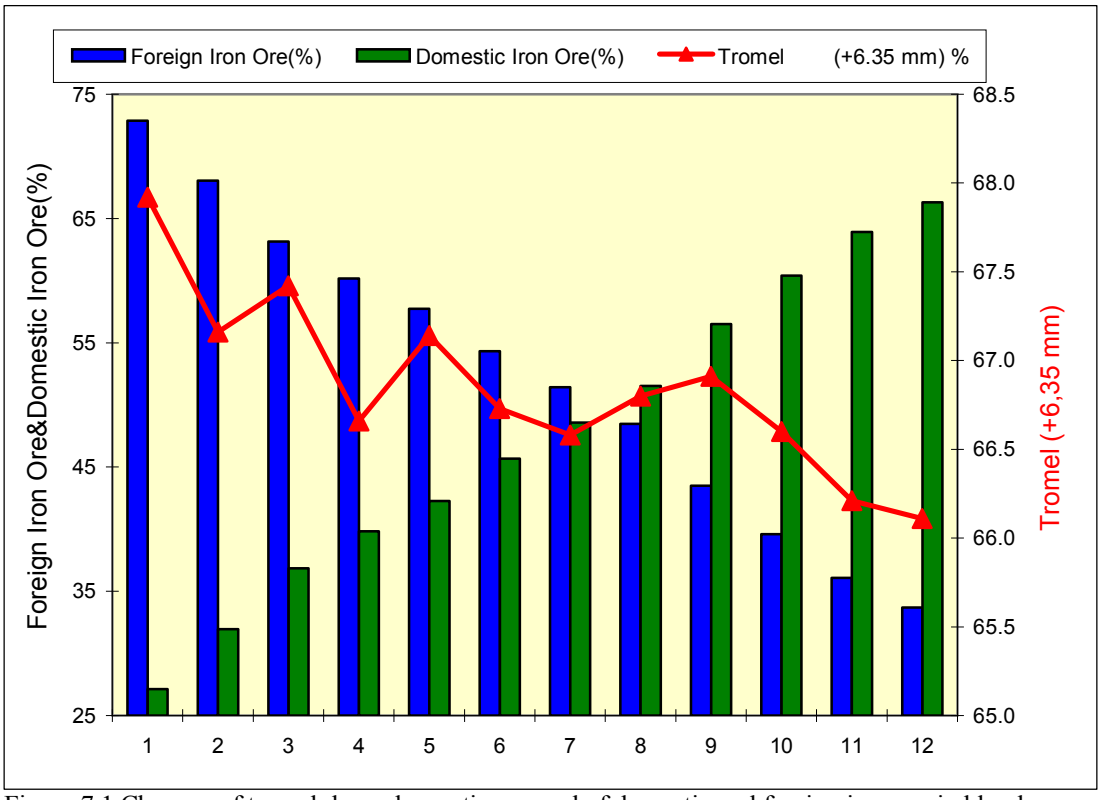


Figure 7.1 Changes of tromel depend on ratio on used of domestic and foreign iron ore in blend.

While domestic iron ore ratio in the blend increases, tumbler index value decreases that depend on imported and domestic iron ore usage rate in the sinter blend. Despite the fact that domestic iron ore dust rate is 27.13 %, tumbler index is 67.92%. Although domestic iron dust rate is 66.30 %, tumbler index is 66.11 %. Figure7.1 indicates this situation.

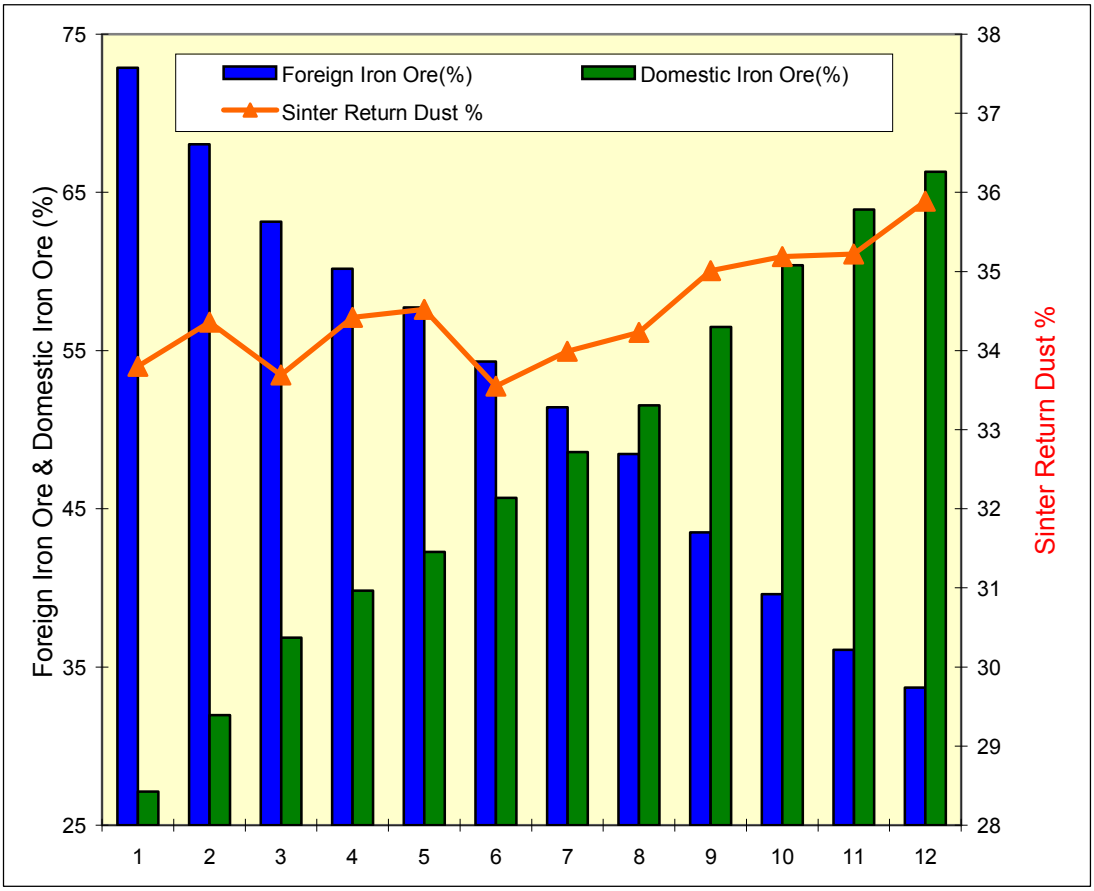


Figure 7.2 Changes of sinter return dust depend on ratio on used of domestic and foreign iron ore in blend.

While domestic iron ore ratio in the blend increases, sinter return dust rate increases that depend on imported and domestic iron ore used rate in sinter blend. Despite the fact that domestic iron ore dust rate is 27.13 % sinter return dust rate is 33.80 %. Although when the sinter return dust rate decreased to 35.89 %, domestic iron dust rate is 66.30 %. This was indicated in figure 7.2

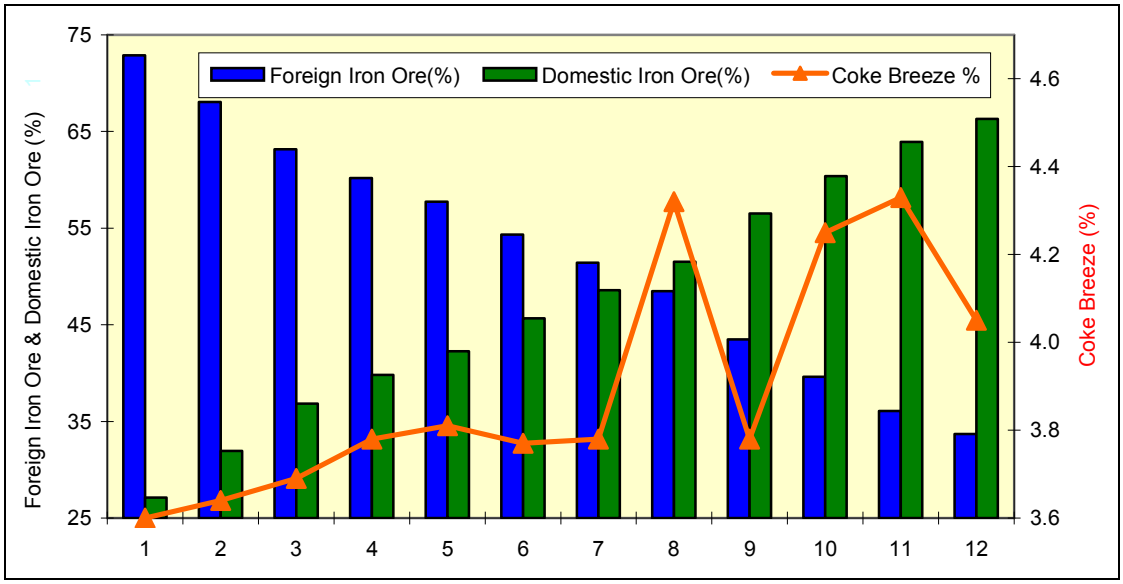


Figure 7.3 Changes of coke breeze depend on ratio on used of domestic and foreign iron ore in blend.

While domestic iron ore ratio in the blend increases, coke breeze dust rate increases that depend on imported and domestic iron ore used rate in sinter blend. Despite the fact that domestic iron ore dust rate is 27.13 % coke breeze rate is 3.60 %. Although when the coke brezee consumption increased to % 4.05, domestic iron dust rate is 66.30 %. Figure 7.3 indicates this situation.

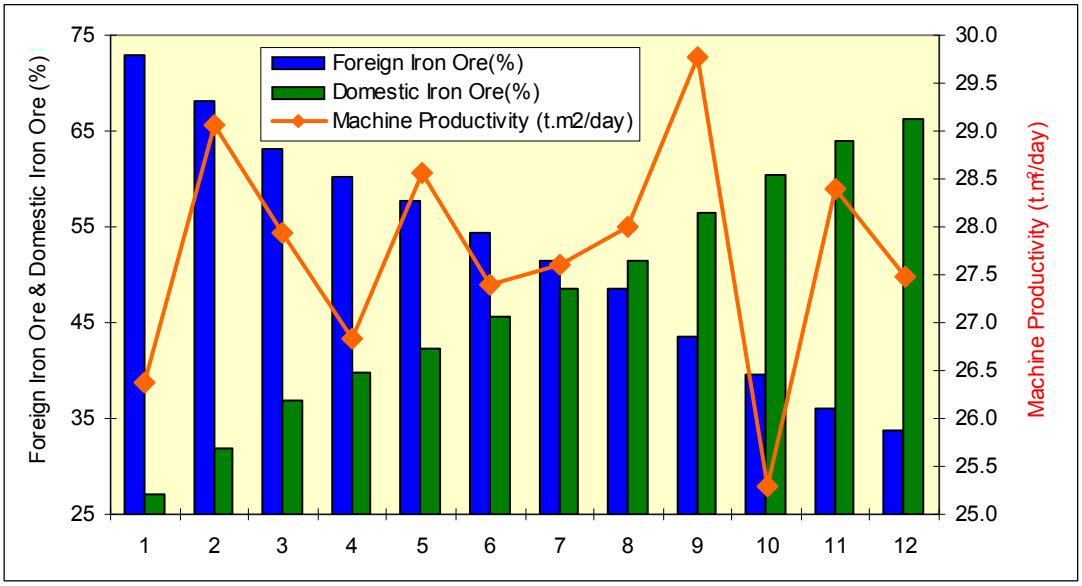


Figure 7.4 Changes of machine productivity depend on ratio on used of domestic and foreign iron ore in blend

Depending on the imported and domestic iron are usage rate is due to changes in machine efficiency. (wavy variables between 25.29 to 29.78 t.m^2/day). This situation was explained in figure 7.4

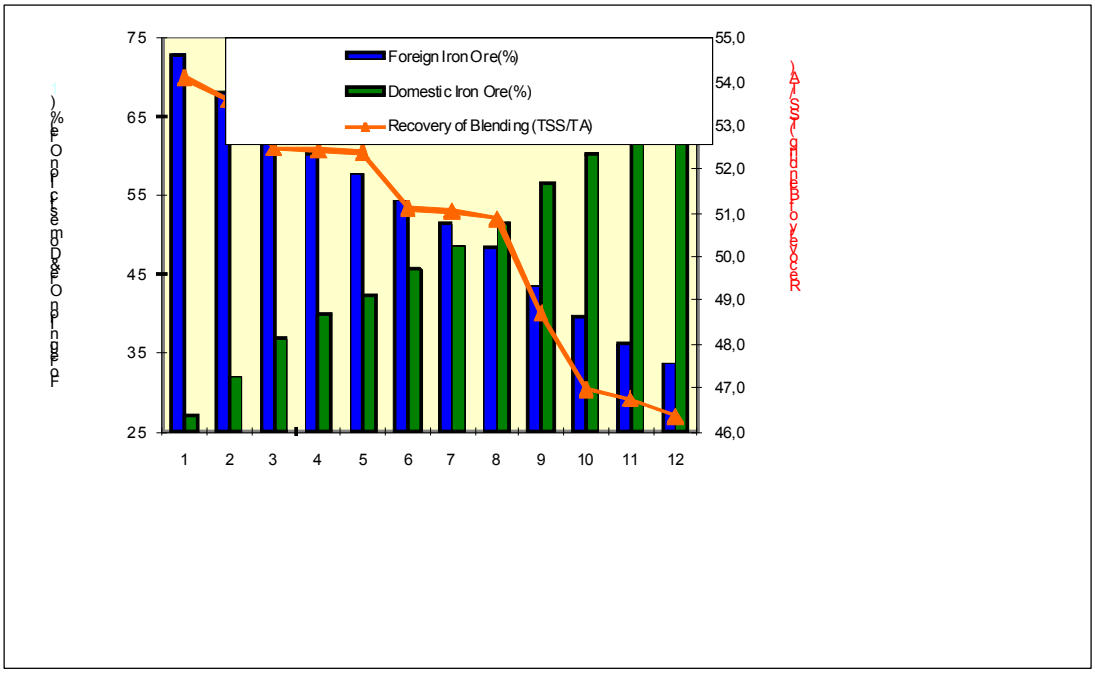


Figure 7.5 Canges of recovery of blending depend on ratio on used of domestic and foreign iron ore in blend

While domestic iron ore ratio in the blend increases, recovery of blending rate decreases are demonstrated that depend on imported and domestic iron ore used rate in sinter blend. Despite the fact that domestic iron ore dust rate is 27.13 %, recovery of blending rate is 54.08 %. Although when the recovery to blending rate decreased to 46.36 % domestic iron dust rate is 66.30 %. Figure7.5 indicates this situation.

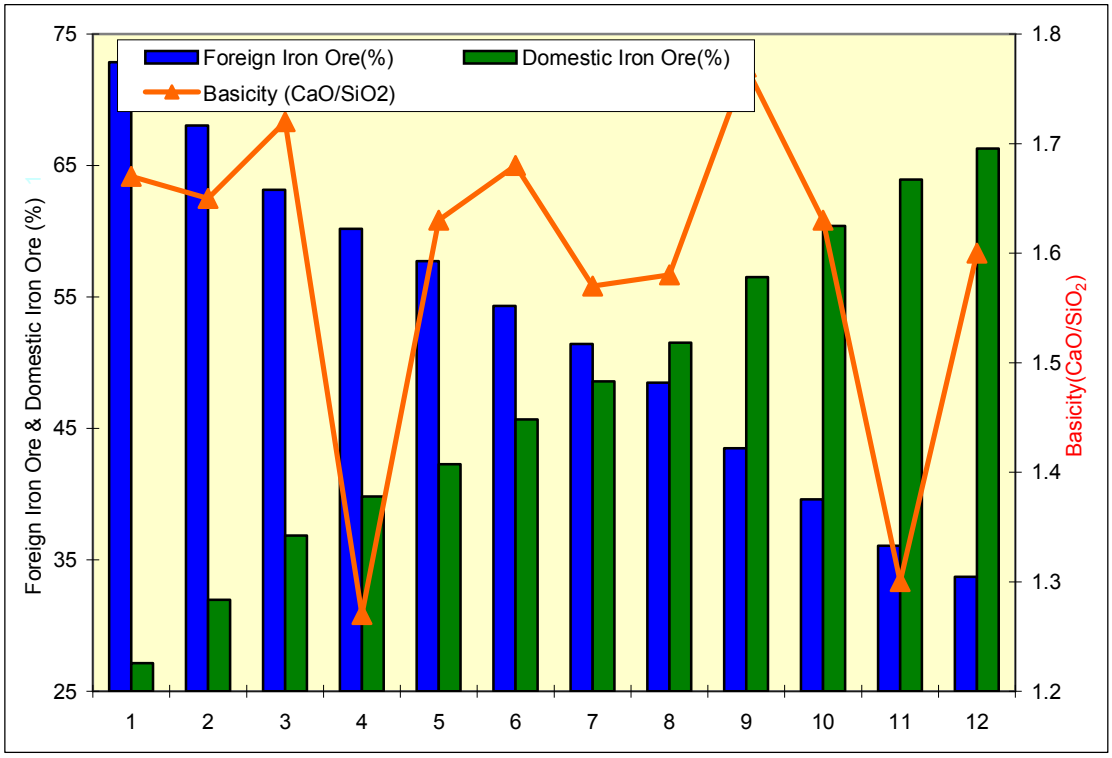


Figure 7.6 Changes of basicity depend on ratio on used of domestic and foreign iron ore in blend

Basicity values are changing that depends on both imported and domestic iron ore usage rate and SiO_2 grade in the blend. Wavy variables between 1.27 % to 1.77 %. This situation was explained in figure 7.6

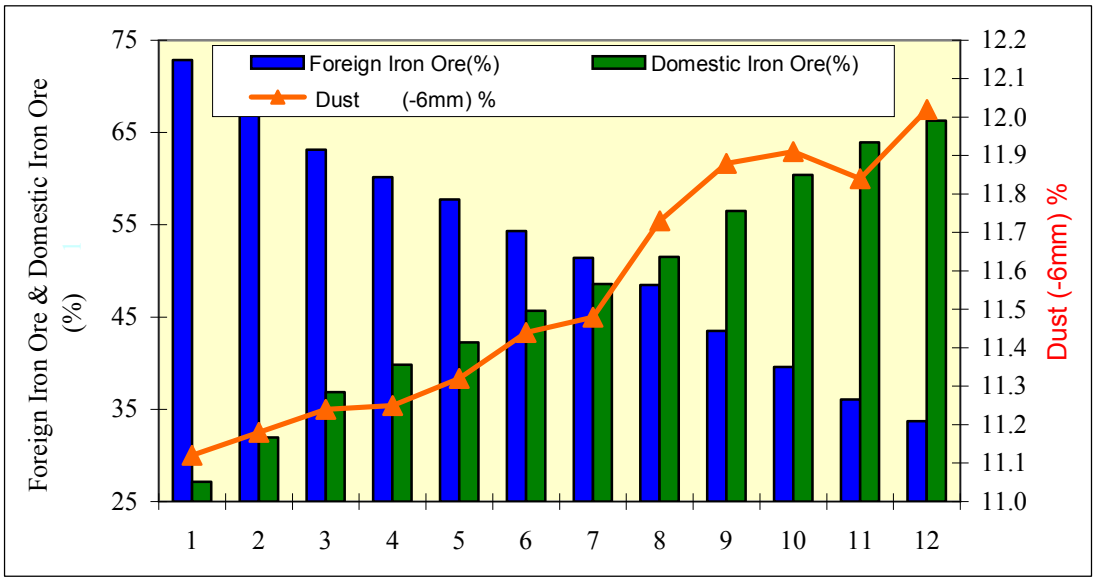


Figure 7.7 Changes of dust depend on ratio on used of domestic and foreign iron ore in blend

While domestic iron ore ratio in the blend increases, sinter dust (-6mm) increases are domstrated that depend on imported and domestic iron ore used rate in sinter

blend. Despite the fact that domestic iron ore dust rate is 27.13%, sinter dust is 11.12%. Although when the sinter rate increased to 12.02% domestic iron dust rate is 66.30%. This situation was explained in figure 7.7.

7.3.2 Examples of Sinter of Obtained with used Different Foreign Iron Ore in Blend

Iron ore (A), iron ore (B) and iron ore (C) were supplied from Turkey, whereas foreign iron ore (FIO1), foreign iron ore (FIO2) and foreign iron ore (FIO3) and foreign iron ore (FIO4) were used in sinter test program. These iron ores were uniformly mixed respectively to reduce fluctuation of chemical composition and obtain representative sample.

Fixed proportioning ratio among iron ore fines and relevant parameters applied in Test program are listed as follows;

1) Foreign iron ores: domestic iron ores = 50: 50

2) A: B: C: = 20:15:15 (based on weight %)

3) BF flue dust and iron bearing sludge account for 2.0 % and 1.5 % of raw mix respectively. According to the test program, sintering test was carried out under condition of basicity ($CaO/Si0_2$) 1.8. Moisture in raw mix is fixed at 7.0 %. Coke breeze consumption is fixed at 6 %. Different type of imported iron ores effect on the quality of sintering is given in table 7.10

Table 7.10 Different type of imported iron ores effects on the quality of sintering

Samle	Foreign Iron Ore %	Domestic Iron Ore %	Tromel (+6.35 mm) %	Recovery of Blending (TSS/TA)	Machine Productivity ($t.m^2/day$)	Machine Speed (m/min)	Basicity (CaO/SiO_2)	%Fe	Dust (-6mm) %	FeO %	Al_2O_3	Coke Breeze %	Sinter Return Dust %
FIO1	50.00	50.00	66.66	52.62	28.94	2.15	1.27	53.37	11.25	10.54	1.44	4.62	32.70
FIO2	50.00	50.00	66.56	52.46	27.2		1.30	55.52	11.37	10.88	1.87	4.44	35.38
FIO3	50.00	50.00	65.63	49.63	25.82	2.26	1.47	53.44	12.15	10.42	1.50	4.41	37.70
FIO4	50.00	50.00	67.9	53.28	29.36	2.14	1.67	57.3	11.41		1.27	3.74	31.45

Sinter was made using different types of imported iron ore. Imported ores are called as FIO1, FIO2, FIO3 and FIO4. Each of them is sintering characteristics is indicated below. This study used the same proportions of imported and domestic iron ores.

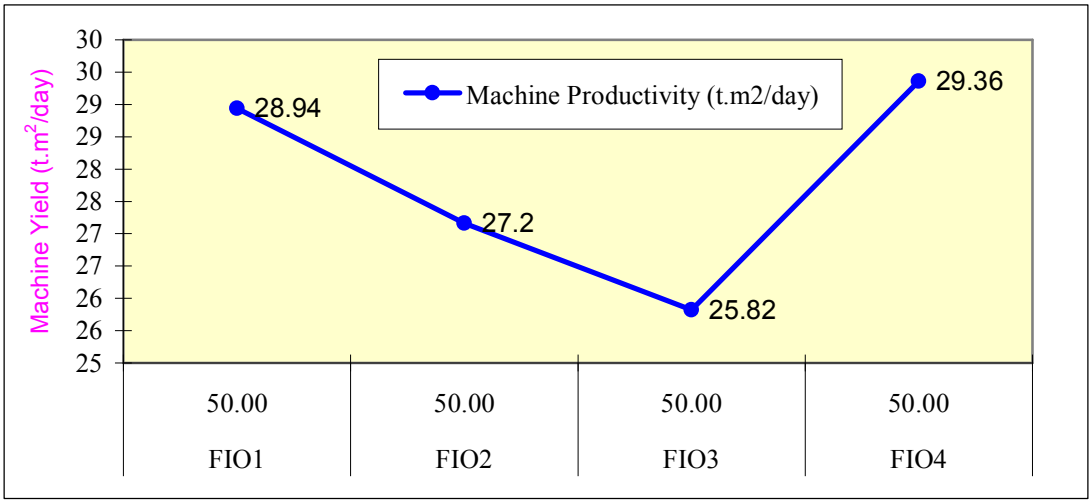


Figure 7.8 Changes ratio of machine yield at sinter, depending on using foreign iron ores in blend.

Changing the machine efficiency values depending on FIO1, FIO2, FIO3 and FIO4 imported ore utilization rates in blend. Figure 7.8 indicates for FIO1 (28.94 %) and FIO4 (29.36 %) of iron ore in the highest value reached and FIO2 (27.20 %) and FIO3 (25.82 %) iron ore used in the blend was decreased in the daily machine productivity.

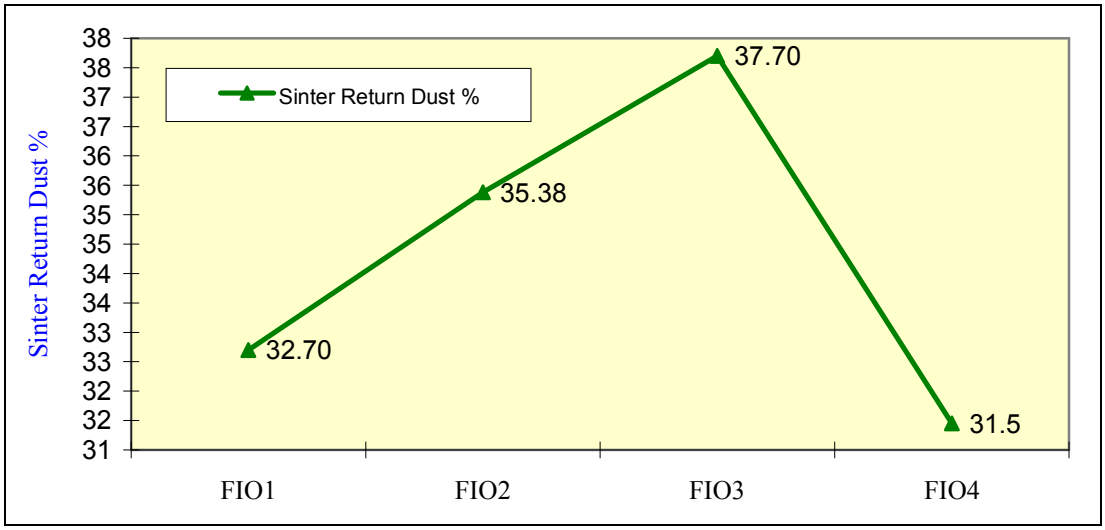


Figure 7.9 Changes ratio of sinter return dust at sinter, depending on using foreign iron ores in blend.

Depending on FIO1, FIO2, FIO3 and FIO4 iron ore type use in the blend that sinter return dust vary. While FIO2 iron ore used in the sinter, its sinter return dust is 35.38%. For FIO3 iron ore used its value is 37.70 %. This is the highest return value for sintering. On the other hand, FIO1 and FIO4 iron ore used in the sinter were obtained 32.70% and 31.50% values, respectively. This was indicated as figure 7.9.

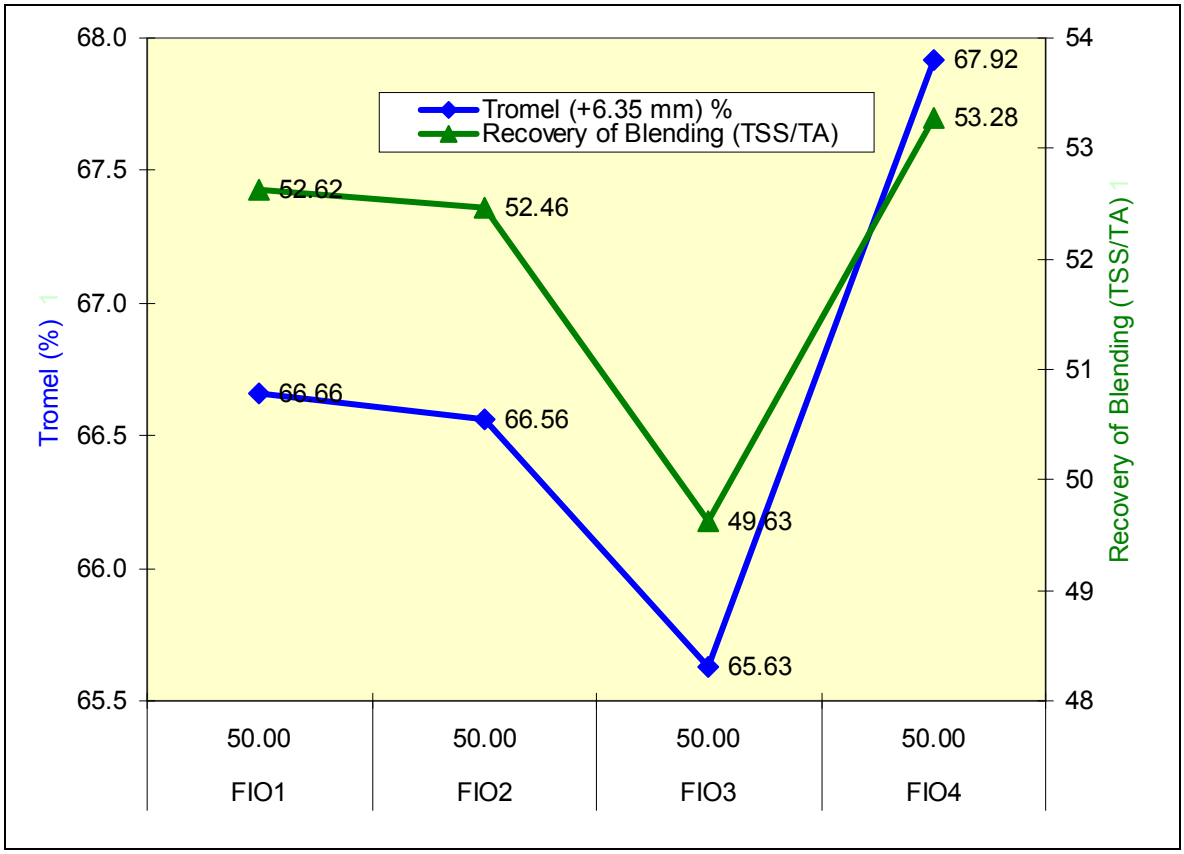


Figure 7.10 changes ratio of recovery of blending and tromel at sinter, depending on using foreign iron ores in blend.

Depending on FIO1, FIO2, FIO3 and FIO4 iron ore type use in the blend that sinter thumbler index and recovery of blending vary. FIO1 iron ore used that thumbler index value is %66.66 %, FIO2 used, its value is 66.56 % and FIO3 used, its value is 65.63 % and FIO4 ore used that was obtained the highest value as 67.90 .

According to the recovery of blending. FIO3 iron ore used its value the lowest 49.63 %. FIO4 iron ore used that was obtained the highest value as 53.28 %. Figure 7.10 indicates this situation.

7.4 R & D Laboratory test work with MECC Corporation

R & D Laboratory test work was done operating agreement between China Metallurgical Equipment Corporation (MECC) and the ISDEMIR. The test results are summarized as test report worked out in great detail. Coke breeze consumption, basicity test, moisture test of raw mix, reducibility test, mineralogical analysis of sinter was investigated in this study. Also, tumbler index of product sinter in hot pot sintering tests were compared with sinter plants of different countries.

7.4.1 Preparation of Raw Materials

Five kinds of iron ore fines, namely A, B, C, D and E, shall be uniformly mixed respectively in order to reduce fluctuation of chemical composition and obtain representative sample. These iron ore fines are mixed as per the proportioning ratio requirement. The homogenous sample after mixing shall be kept ready for test.

The iron bearing sludge is irregular lump shaped material with size of approx. 10~15mm and slightly hard with moisture content of 13.75%. If this lumpy material is directly put into sintering production, it will easily cause segregation and unburnt phenomenon, which may result in deterioration of sinter quality. To meet requirement of production and test, the lumpy iron bearing sludge shall be manually crushed for use in test. It's also required in industrial production to carry out treatment on lumpy iron bearing sludge. Iron bearing raw materials are identified as follows:

Iron ore A (tag name: I.O.A), Iron ore B (tag name: I.O.B), Iron ore C (tag name: I.O.C), Iron ore D (tag name: I.O.D), Iron ore E (tag name: I.O.E), BF flue dust and iron bearing sludge. Limestone fines and dunite fines used as flux material. Coke breeze used as fuel.

Among which Iron ore fines; E is identified as Brazil I.O,E namely the “foreign iron ore fines” mentioned in hot pot test program provided. Except that, ore fines A, ore fines B, ore fines C and ore fines D are supplied from Turkey.

7.4.2 Measurement of physical & chemical properties of raw materials

Measurement results of chemical composition analysis and physical property measuring of various raw materials The measurement results of chemical composition analysis and physical property of raw materials are shown in table 7.11

Table 7.11 Chemical composition and physical property of raw materials.

Name of raw materials			Chemical Composition (%)							Moisture	Bulk
	TFe	FeO	SiO_2	Al_20_3	CaO	MgO	S	P	Loss	(%)	(t/m^3)
I.O.A	39.56	36.18	4.71	1.47	1.37	3.05	0.150	0.018	28.00	1.25	1.54
I.O.B	54.69	0.43	6.36	0.86	1.06	0.90	0.090	0.030	12.02	6.50	2.22
I.O.C	49.27	0.35	10.17	3.44	1.26	0.68	0.040	0.054	12.36	9.00	1.88
I.O.D	59.83	15.47	5.73	1.79	1.02	2.06	0.640	0.026	3.28	2.60	2.63
I.O.E	65.16	0.35	3.65	1.37	0.56	0.09	0.017	0.013	1.94	3.00	2.85
B.F. flue dust	30.81	5.45	8.50	2.92	4.12	1.34	0.310	0.028	35.32	9.00	1.11
Iron bearing sludges	54.52	10.60	3.18	0.96	5.86	2.24	0.160	0.026	7.88	13.75	1.70
Limestone fines	0,36		1.30	0.27	51.45	2.24	0.007	0.012	43.10	0	1.83
Dunite fines	4.66	1.56	28.36	0.91	11.54	31.83	0.017	0.016	18.90	4.25	1.80
Coke breeze	Fe203: 17.32		47.01	23.03	2.49	2.91	0.610			9.50	0.77
In coke breeze: fixed carbon accounts for 79,24 %; ash content for 16,11 %; volatile for:4,65 %											

7.4.3 Mineralogical analysis of iron ore fines

The mineralogical analysis of iron ore fines indicates existing form of iron element in iron ore fines. The analysis results are shown in table 7.12

Table 7.12 Mineralogical analysis of iron ore fines.

I.O. samples	TFe	Distribution of iron in various minerals (%)						
		Magnetite	Siderite	Hematite	limonite	Pyrite	Pyrrhotine	Iron silicat
I.O.A	39.56	1.34	23.51	8.95	4.22	0.130	0.004	1.41
I.O.B	54.69	0.91	1.80	16.64	34.84	0.078	0.001	0.42
I.O.C	49.27	1.52	0.20	8.67	37.70	0.035	0.001	1.14
I.O.D	59.83	50.85	0.89	2.73	3.97	0.560	0.470	0.36
I.O.E	65.16	3.09	0.41	53.96	4.01	0.015	0.001	3.67

It's indicated in table 7.12 that I.O.A is combined with siderite, hematite and limonite; I.O.B and I.O.C are combined ore with hematite and limonite; while I.O.D is magnetite; I.O.E is hematite.

7.4.4 Measurement of particle size of raw materials

Measurement result of particle size of various raw materials are shown in table 7.13

Table 7.13 Particle sizes of raw materials.

Name of raw materials	+10 mm	-10+8 mm	-8+7 mm	-7+5 mm	-5+3 mm	-3+2 mm	-2+1 mm	-1 mm	-3 mm	-8 mm
I.O.A	3.5	4.5	2.0	9.0	7.0	9.0	16.0	49.0	74.0	92.0
I.O.B	25.0	9.5	6.0	9.0	10.0	5.5	9.5	25.5	40.5	65.5
I.O.C	0.0	7.0	15.1	25.0	27.0	11.0	11.5	3.0	25.5	93.0
I.O.D		0.0	1.0	3.5	9.0	5.5	13.0	68.0	86.5	100.0
I.O.E	0.0	3.0	7.0	9.0	12.0	6.5	10.5	52.0	69.0	97.0
Coke breeze		0.0	2.0	8.0	14.0	12.0	19.0	45.0	76.0	100.0
Limestone fines			0.0	2.5	22.0	14.0	19.5	42.0	75.5	100.0
Dunite fines			0.0	2.0	7.0	8.5	18.5	64.0	91.0	100.0
B.F flue dust						0.0	3.0	97.0	100.0	
Iron bearing sludge						0.0	6.0	94.0	100.0	

With reference to data listed above, make up fraction of I.O.A, B & C with grain size > 8mm is respectively 8%, 34.5% and 7%. In conclusion, grain size is relatively coarse, especially I.O.C (>8mm accounting for 34.5%), which can influence sinter strength. During actual industrial production, the coarse fraction shall be crushed up to -8mm.

7.4.5 Equipment for Sintering Test

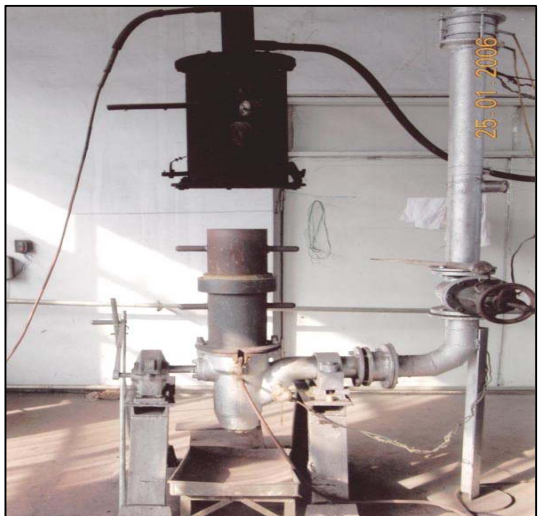
- Drum mixer: Φ280X600mm, 32 r/min;
- Sintering hot pot: Φ 150mm, 620mm in height
- Shatter index equipment: It is equipped with electric lifting device with opening door at bottom. The lifting height is 2m. The sinter falls down onto 10mm thick steel plate.
- Sieve set: 40mm, 25mm, 15mm, 12.5mm, 10mm, 6.3mm, 5mm;
- 1/5 ISO tumbler : Φ1000mm X 100mm, 25r/min, On inner wall of the tumbler, there are two 50 mm high lifting plates, 180° to each other.
- Exhaust fan: Model J6, 10kW, 960r/min, 2000 mmH_20 (Pa) 13m^3/min.
- Ignition device, thermal couple, de-dusting equipment and pipeline etc.

The major equipment is shown in color illustration as follows.

7.4.6 Methodology for sintering hot pot test

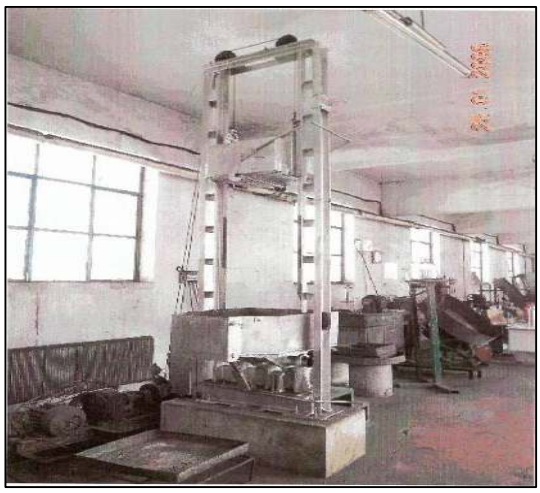
The process flow of sintering hot pot test is shown in process flow diagram. Equipment used for making sinter pot test is given in figure 7.11. Process flow diagrams for sintering hot pot is given in figure 7.12.

Ignition Furnace (Φ 200 mm)

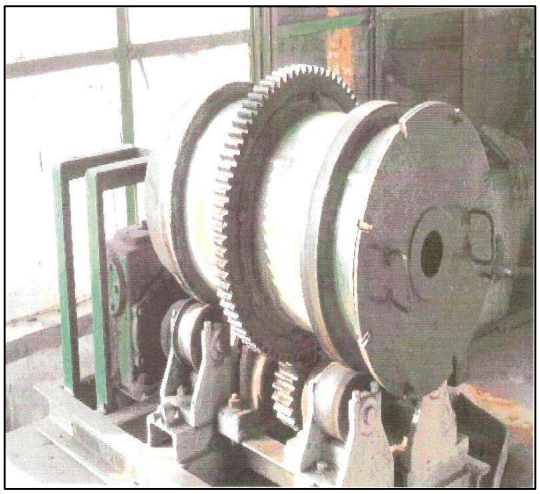


Hot Sintering Pot (Φ 150 mm 620 mm in height)

Hot sintering pot Φ 350mm



Shatter index equipment (Lift height 2m)



Drum mixer (Φ 280mm X 600mm, 32r/min)

ISO Tumbler drum (Φ 1000 mm X 100 mm 25 r/min)

Figure 7.11 Equipment used for making sinter pot test.

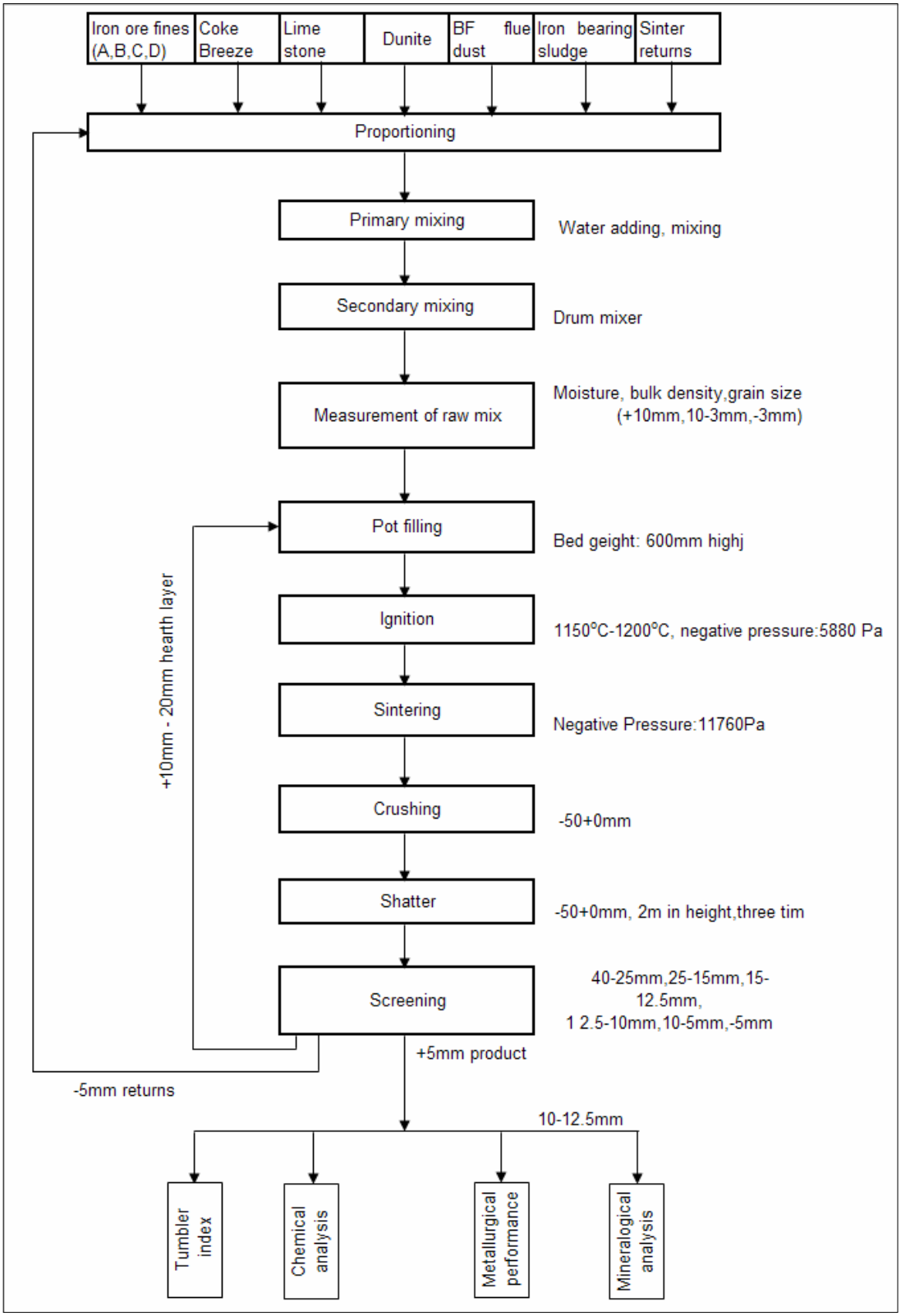


Figure 7.12 Process flow diagrams for sintering hot pot.

7.4.7 Methodology for sintering hot pot test

Proportioning is conducted as per test conditions. Under each test condition, 20 kg raw mix (dry basis) is required. A certain amount of water is added into raw mix for primary manual mixing. The raw mix after primary mixing is fed into Φ280X 600 mm mixing drum to start secondary mixing for 3 minutes with the speed revolution of 32r/m. Further mixing and balling is completed in secondary mixing process. Measurement of moisture content, bulk density and particle size (+10mm, 10~3mm, -3mm) of raw mix after secondary mixing are carried out. Raw mix after secondary mixing is distributed into sintering pot. The sintering pot is 620 mm in height with internal dia of 150 mm, which can be turned upside down. Grate plate with ϕ 6mm holes are provided at the bottom of sintering pot. Hearth layer with grain size of 10~20 mm is distributed onto grate plate by 20 mm in depth. Bed depth in sintering hot pot test is 620 mm. Ignition device starts firing after distribution of raw mix. Liquefied petroleum gas is used as ignition fuel. Ignition temperature is measured by thermal coupler. Assumed ignition temperature for test is 1150 °C ~1200 °C with ignition duration of 1 minute. The suction pressure at the time of ignition is 5880 Pa while suction pressure at the time of sintering process is 11760 Pa. Sintering time is considered from start of ignition till burn through point. Records are kept for sintering time, negative pressure variation and waste gas temperature. When sintering is finished, sinter cake is dumped out, and then put into shatter index test equipment, which is equipped with electric lifting device and opening door at bottom.

During transfer and loading/unloading, product sinter with poor strength is prone to be disintegrated and to generate sinter fines (<5mm) due to impact. If the sinter fines are fed into Blast Furnace, permeability of B.F. burden is deteriorated as well as the other negative effect takes place, therefore, shatter index test is carried out to measure anti disruptive strength. Sinter cake is put into loading box, which is lifted to a height of 2 m. Subsequently, it's freely dropped onto 10 mm thick steel plate under gravity, which is repeated for 3 times. Finally, sieve with 5 mm square hole is used to separate sinter with grain size of + 5mm as final product sinter from sinter with grain size of -5 mm as sinter returns Tumbler index test is made to final product using 1/5 ISO standard method.

Tumbler drum is of 1000 mm diameter and 100 mm in width with rotation speed of 25r/min. one batch of 3kg sinter with grain size of 10~40mm is put into tumbler drum for 200 revolutions. After that, the sinter is screened by sieve with square openings. The weight percentage of +6.3mm fraction is taken as tumbler index while the weight percentage of -0.5mm is taken as abrasion index. Chemical composition analysis, metallurgical perfo.rmance index and mineralogical analysis are carried out for final product sinter.

7.4.8 Content of test

Fixed proportioning ratio among iron ore fines and relevant parameters furnished in test program are listed as follows:

1) Foreign iron ore fines (not supplied from Turkey) : domestic iron ore fines (supplied from Turkey) = 65 : 35; namely I.O.E : I.O.F. (A+B + C+D) = 65 :35

2) A: B: C: D = 26: 40: 14:20

3) BF flue dust and iron bearing sludge account for 2.2% and 2.0% of raw mix respectively.

According to test program, sintering test is carried out under condition of three kinds of basicity (CaO/SiO_2), i.e., 1.0; 1.7 and 2.1. The other tests include coke breeze consumption and moisture content in raw mix also under condition of three kinds of basicity (CaO/SiO_2), i.e., 1.0; 1.7 and 2.1. Comparative test is carried out when addition of dunite fines is made in the raw mix based on the dunite fines furnished. Calculation results of proportioning ratio under different test conditions are shown in table 7.14

Table 7.14 Proportioning ratio under different test conditions % (on dry basis).

SI. No.	Basicity CaO/SiO_2	I.O.E	I.O.A	I.O.B	I.O.C	I.O.D	BFflue dust	Iron-bearing sludge	Limestone	Dunite	Coke breeze	Sinter returns
I	1.0	46.09	6.45	9.93	3.48	4.96	2.2	2.0	10.45	0	5.45	9.0
2	1.0	45.50	6.37	9.79	3.43	4.89	2.2	2.0	9.55	1.82	5.45	9.0
3	1.0	44.90	6.29	9.67	3.38	4.83	2.2	2.0	8.64	3.64	5.45	9.0
4	1.7	42.66	5.97	9.19	3.22	4.59	2.2	2.0	15.27	0	5.90	9.0
5	1.7	41.83	5.86	9.01	3.15	4.50	2.2	2.0	14.73	1.82	5.90	9.0
6	1.7	41.05	5.75	8.84	3.10	4.42	2.2	2.0	14.10	3.64	5.90	9.0
7	2.1	41.06	5.75	8.84	3.10	4.42	2.2	2.0	17.27	0	6.36	9.0
8	2.1	40.17	5.62	8.65	3.03	4.33	2.2	2.0	16.82	1.82	6.36	9.0
9	2.1	39.40	5.52	8.49	2.97	4.24	2.2	2.0	16.18	3.64	6.36	9.0

7.4.9 Test result and analysis

7.4.9.1 Coke breeze consumption test

Iron ore fines used in the test mainly consist of limonite. Therefore, there is large amount of crystal water, of which the decomposition temperature is far higher than that for evaporation of free water. To ensure sufficient heat energy and sintering temperature due to crystal water decomposition and all kinds of reactions during sintering process, adequate fuel consumption is required. Therefore, coke breeze consumption in the test is much more than that for sintering of magnetite in the test. Tests have been respectively carried out under condition of four kinds of coke breeze consumption, i.e., 5.00%, 5.45%, 5.90% and 6.36%. Appropriate fuel consumption for obtaining sinter with different basicity has been examined through tests. Moisture in raw mix is fixed as 7.0%. Test results are shown in table 7.15.

Table 7.15 Coke breeze consumption test results.

Basicity (CaO/SiO_2)	Coke breeze (%)	Vertical speed (mm/min)	Sinter cake yield (%)	Final sinter ratio (%)	Sinter returns ratio (%)	Productivity ($t/m^2.h$)	Tl +6.3 mm (%)	Abrasion index -0.5mm (%)	Chemical composition of product sinter (%)								Desulfurization efficiency (%)
									TFe	FeO	SiO_2	Al_2O_3	CaO	MgO	S	P	
1.0	5.00	31.7	79.5	68.9	10.6	1.78	61.0	10.0	58.44	13.85	6.90	1.75	7.01	0.87	0.012	0.014	91.5
1.0	5.45	32.0	79.5	69.3	10.2	1.80	63.3	8.5	58.81	14.68	7.18	1.68	7.03	0.88	0.012	0.014	91.3
1.0	5.90	32.8	79.7	69.5	10.2	1.80	63.8	8.8	58.85	16.65	7.09	1.97	7.05	0.88	0.012	0.014	91.3
1.0	6.36	33.0	79.5	70.0	9.5	1.83	64.0	7.3	58.88	17.55	7.18	2.07	7.23	0.85	0.012	0.014	91.0
1.7	5.00	34.2	77.4	68.0	9.4	1.76	65.0	9.8	56.28	12.05	5.80	2.05	9.78	0.92	0.012	0.014	88.0
1.7	5.45	34.8	77.7	68.8	8.9	1.79	66.0	8.3	56.38	12.33	5.62	2.11	9.60	0.99	0.012	0.014	88.1
1.7	5.90	34.8	77.8	69.0	8.8	1.88	67.7	7.0	56.71	12.60	6.63	1.92	11.04	0.85	0.013	0.014	88.1
1.7	6.36	34.8	77.6	69.2	8.4	1.89	67.8	7.0	56.53	14.63	6.05	2.19	10.09	0.94	0.013	0.014	88.0
2.1	5.00	34.4	76.8	67.1	9.7	1.89	66.5	7.0	54.76	10.96					0.012	0.013	86.2
2.1	5.45	34.8	76.8	67.4	9.4	1.89	67.7	7.0	54.97	11.13					0.013	0.015	86.8
2.1	5.90	34.8	76.7	67.8	8.9	1.99	69.0	6.8	55.19	11.20					0.012	0.015	87.0
2.1	6.36	34.8	76.6	68.0	8.6	2.01	72.0	5.8	55.46	11.28					0.012	0.015	86.1

The analysis result is described as below with reference to the content in table 7.15:

1) Influence of coke breeze consumption on strength of sinter:

With the increase of coke breeze consumption, the strength of sinter is obviously improved.

At basicity of 1.0, along with coke breeze consumption increased from 5.00% to 5.45%, 5.90% and 6.36%, the sinter tumble index improves gradually from 61.0% to 63.3%, 63.8% and 64.0%. When coke breeze consumption increased from 5.00% to

5.45%, the tumble index improves remarkably. As coke breeze consumption continues to increase, the extent of tumble index improvement is not the same ratio.

At basicity 1.7, along with coke breeze consumption increased from 5.00% to 5.45%, 5.90% and 6.36%, the sinter tumbler index improves gradually from 65.0% to 66.0%, 67.7% and 67.8%. When coke breeze consumption increased from 5.00% to 5.90%, the tumbler index improves remarkably. As coke breeze consumption continues to increase, the extent of tumbler index improvement is not so significant.

At basicity 2.1, along with coke breeze consumption increased from 5.00% to 5.45%, 5.90% and 6.36%, the sinter tumbler index improves significantly from 66.5% increased to 67.7%, 69.0% and 72.0%. The tumbler index at coke breeze consumption of 6.36% is optimum.

2) Effect of coke breeze consumption upon sinter cake yield and productivity:

Under condition of different kinds of basicity, coke breeze consumption has no obvious effect on sinter cake yield. However, increase in coke breeze consumption is conducive to improvement of productivity.

When basicity is 1.0, along with the increase of coke breeze consumption, productivity is improved from 1.78 to 1.83.

When basicity is 1.7, along with the increase of coke breeze consumption, the productivity is improved from 1.76 to 1.89.

When basicity is 2.1, along with the increase of coke breeze consumption, the productivity is improved from 1.89 to 2.01.

3) Effect of coke breeze consumption upon vertical sintering speed and sinter cake yield:

When basicity is 1.0, along with the increase of coke breeze consumption, the vertical sintering speed is improved slightly.

When basicity is 1.7 and 2.1, coke breeze consumption change has no remarkable effect on vertical sintering speed. But product sinter yield is increased, sinter returns ratio is decreased, productivity is therefore improved.

When basicity is fixed, the coke breeze consumption has no remarkable effect on sinter cake yield.

4) Relation between coke breeze consumption and basicity:

From tumbler indexes of sinter product with different coke breeze consumption under condition of various basicity, we can tell if basicity is higher, coke breeze consumption at the time of max. Tumbler index is also higher. It is because of more flux addition when basicity is increased, which causes change of mineralogical composition of sinter and subsequent improvement of sinter strength. However, the decomposition of limestone absorbs heat engery. When limestone consumption increases, the consumption of fuel must be appropriately increased to ensure provision of heat energy required by physical & chemical reactions during sintering process. That is why the more coke breeze consumption should be considered when the basicity is increased.

5) Effect of coke breeze consumption upon FeO content in product sinter:

The test results show that FeO content in product sinter is increased notably when coke breeze consumption is more. When basicity is 1.0, along with increase of coke breeze consumption from 5.00% up to 6.36%, FeO content in sinter product rises from 13.85% to 17.55%. When basicity is 1.7, along with increase of coke breeze consumption from 5.00% up to 6.36%, FeO content in sinter product rises from 12.05% to 14.63%.

When basicity is 2.1, along with increase of coke breeze consumption from 5.00% up to 6.36%, FeO content in sinter product rises from 10.96% to 11.28%. It is ascertained by comprehensive test indexes that the coke breeze consumption of 5.45%, 5.90% and 6.36% is optimum for sintering when sinter basicity is 1.0, 1.7 and 2.1 respectively.

7.4.9.2 Basicity test

In basicity tests, coke breeze consumption for various basicity of product sinter is guided by parameters concluded in coke breeze consumption tests. Moisture of raw rnix is fixed as 7.0%. All test result indexes are shown in table 7.16.

Table 7.16 Basicity test index.

Basicity CaO/SiO_2	Coke breeze (%)	Moisture in mixture (%)	Bulk density of mix. (t/m^3)	Vertical sintering speed (mm/min)	Sinter cake yield (%)	Final product ratio (%)	Return fines ratio (%)	Utilisation coefficient ($t/m^2.h$)	
								$t/(m^2.h)$	$t/(m^2.h)$
1.0	5.45	7.0	1.89	32.0	79.5	69.3	10.2	1.80	43.20
1.7	5.90	7.0	1.83	34.8	77.8	69.0	8.8	1.88	45.12
2.1	6.36	7.0	1.80	34.8	76.6	68.0	8.6	2.01	48.24

Tumbler index	Abrasion	Granulometry of product sinter (mm%)					
		+40mm	40-25mm	25-15mm	15-12.5mm	12.5-10mm	10-5mm
63.3	8.5	38.94	15.08	17.59	8.30	6.30	13.82
67.7	7.0	35.77	17.66	15.40	8.98	6.97	15.24
72.0	5.8	25.18	21.58	19.42	10.07	6.47	17.28

Chemical composition of product sinter (%)								Desulfurization
TFe	FeO	$Si0_2$	$A1_20_3$	CaO	MgO	S	P	
58.81	14.68	7.18	1.68	7.03	0.88	0.012	0.014	91.3
56.71	12.60	6.63	1.92	11.04	0.85	0.013	0.014	88.1
55.46	11.28	6.47	1.95	13.38	0.88	0.012	0.015	86.1

It's concluded from table 7.16 as follows:

1) Effect of basicity on sinter strength:

When basicity is higher, sinter strength is obviously increased. When basicity is 1.0, tumbler index is 63.3%. When basicity is 1.7, tumbler index is increased to 67.7%. When basicity is further up to 2.1, tumbler index is increased to 72.0 %.

2) Effect of basicity upon vertical sintering speed, final product ratio and productivity:

When basicity improves from 1.0 to 1.7, vertical sintering speed is increased slightly. When basicity is further upgraded upto 2.1, vertical sintering speed keeps unchanged. The increase of basicity has no significant influence on final product ratio.

When the basicity increases, the productivity improves slightly. With the basicity increases from 1.0 to 1.7 and 2.1, productivity improves from 1.88 to 2.01.

3) Effect of basicity on sinter cake yield:

With the increase of basicity, the sinter cake yield is obviously reduced. With the increase of basicity from 1.0 to 1.7 and 2.1, the sinter cake yield is reduced from 79.5 % to 77.8% and 76.6%.

4) Effect of basicity on coke breeze consumption:

With the increase of basicity, coke breeze consumption is more. This is the same reason as has been explained in coke consumption test for the relation between coke breeze consumption and basicity.

5) Effect of basicity on Fe content and FeO in sinter:

It's demonstrated from test data that along with the increase of basicity, Total Fe content in sinter is significantly reduced, so is FeO content. With the increase of basicity from 1.0 up to 1.7 and 2.1, TFe in product sinter is reduced from 58.81% to

56.71% and 55.46% as well as FeO content is decreased from 14.68% to 12.60% and 11.28% respectively.

6) Effect of basicity on desulphurization efficiency during sintering process:

With increase of basicity, desulphurization efficiency is slightly reduced. When the basicity is increased from 1.0 to 1.7, the desulphurization efficiency is reduced from 91.3% to 88.1%. When basicity is further increased to 2.1, desulphurization efficiency is reduced from 88.1% to 86.1%. Along with continuous increase of limestone addition, more absorption of heat energy is affected, which will cause sintering temperature lower. This is negative effect for desulphurization.

7.4.9.3 Moisture Test of Raw Mix

The moisture content in raw mix is one of the most significant factors influencing the sintering process. Optimum addition of water into mixture is favorable to homogeneity and balling of raw mix thus improving the permeability of sintering bed. The moisture in mixture is determined by the characteristics of raw material. Optimum moisture is obtained from test. The moisture test results of raw mix for different sinter basicity are shown in table 7.17.

Table 7.17 Moisture test index of raw mix.

Basicity CaO/SiO_2	Coke Breeze %	Moisture in raw mix	Bulk density of	Granulometry of mixture mm %			Vertical sintering speed	Sinter cake yield	Final product ratio	Returns ratio %	Productivity $t/m^2.h$	Tumbler index +6.3mm	Abrasion index -0.5mm
				+10	10-3	-3							
1.0	5.45	6.0	1.85	8	57	35	31.1	79.5	70.0	9.5	1.83	63.0	8.5
1.0	5.45	7.0	1.89	8	62	30	32.0	79.5	69.3	10.2	1.80	63.3	8.5
1.0	5.45	7.5	1.92	14	65	21	31.5	79.3	68.9	10.4	1.79	62.5	9.1
1.7	5.90	6.0	1.80	10	52	36	34.8	77.9	70.0	7.9	1.91	67.0	7.3
1.7	5.90	7.0	1.83	10	60	35	34.8	77.8	69.0	8.8,,	1.88	67.7	7.0
1.7	5.90	7.5	1.85	13	50	37	32.7	77.6	68.2	9.4	1.83	66.8	7.5
2.1	6.36	6.0	1.77	10	55	35	32.7	76.9	68.5	8.4	2.00	71.1	6.2
2.1	6.36	7.0	1.78	10	63	27	34.8	76.6	68.0	8.6	2.01	72.0	5.8
2.1	6.36	7.5	1.82	14	64	22	34.8	76.5	67.6	8.9	1.91	69.0	7.4

It can be seen from table 7.17:

1) Influence of moisture in raw mix upon particle size of raw mix:

From table 7.17, it can be seen that with increase of the moisture content from 6.0% to 7.0%, the quantity of + 10 mm fraction in the raw mix has no change but the quantity of 10~3mm fraction increases remarkably. When the moisture increases to 7.5%, the quantity of+10 mm fraction increases but that of 10~3mm fraction has not big change.

According to experiences, it is required for sintering of iron ore fines with fine grain size that the quantity of +10 mm fraction is less than 10% while that of-3mm fraction less than 15%. In this test, raw materials used are coarser in grain size, the +10 mm fraction should be still required as less than 10 %, but the requirement of percentage of -3 mm fraction may be relax to avoid poor sinfer strength caused by insufficient temperature of sintering bed due to faster vertical sintering speed.

2) Influence of moisture in raw mix on strength of sinter:

When the basicity is 1.0 and 1.7, the tumbler index is basically not changed with the increase of raw mix moisture from 6.0% to 7.0% and further to 7.5%.

When the basicity is 2.1, the tumbler index is basically not changed with the increase of raw mix moisture from 6.0% to 7.0%. But when raw mix moisture increases to 7.5%, the tumbler index of sinter decreases evidently from 72% to 69%. Because if moisture is increased, heat loss is more, subsequently, sintering temperature is decreased, thus sinter strength is reduced.

3) Influence of moisture in raw mix on sinter cake yield, final product ratio and productivity:

The test results demonstrate that when the basicity is fixed, the increase of raw mix moisture has no different influence on sinter cake yield, but the final product ratio and productivity decrease slightly.

With comprehensive consideration of all test indexes, it is decided that 7.0% moisture in raw mix is optimum for sintering less than three basicity.

7.4.9.4 Test of Sintering with Dunite Fines Addition:

Generally, when sintering the magnetite, it is usually to improve sinter quality through increasing the content of MgO in sinter. In this test, iron ore fines mainly consist of hematite and limonite with small quantity of magnetite, which has better reducibility of than magnetite. In consideration that sample has provided dunite fines instead of dolomite in an aim to find possibility to further improve the metallurgical performance of sinter, we carried out further comparative tests with 1.82% and 3.64% dunite fines added into raw mix for comparison with test results obtained from sintering without addition of dunite fines. The test results are shown in table 7.18.

Table 7.18 Sintering test result with dunite fines addition.

Basicity	Dunite	Coke fines	Moisture in raw mix	Bulk density of raw mix	Vertical sintering speed	Sinter cake yield	Final sinter ratio	Return ratio	Productivity	TI	Abrasion index	Chemical composition							
CaO/SiO_2	%	%	%	t/m^3	mm/min	%	%	%	$t/m^2.h$	+6.3 mm %	-0.5mm %	TFe	FeO	SiO_2	AI_2O_3	CaO	MgO	S	P
1.0	0,00	5,45	7.0	1,89	32,00	79.5	69.3	10,20	1,80	63.3	8,50	58.81	14.68	7,18	1,68	7,03	0.88	0,012	0.014
1.0	1,82	5,45	7.0	1,88	32,00	78.0	69.0	9,00	1,74	62.7	9,00		14.28	7,64		7,36	1,48	0,012	0.014
1.0	3,64	5,45	7.0	1,87	31,50	77.2	68.8	8,40	1,62	61.3	10,40	57.48	13.93	7,01	1,76	6,86	2,24	0,012	0.014
1,70	0,00	5,90	7.0	1,83	34,80	77.8	69.0	8,80	1,88	67.7	7,00	56.71	12,60	6,63	1,92	11,04	0,85	0,013	0.014
1,70	1,82	5,90	7.0	1,81	34,50	76.5	68.5	8,00	1,79	67.3	7,00	55.00	12,05	5,99		10,11	1,50	0,014	0.014
1,70	3,64	5,90	7.0	1,80	34,80	75.8	68.5	7,30	1,75	66.0	7,50	54.54	11,81	6,97	1,92	11,60	2,24	0,014	0.014
2,10	0,00	6,36	7.0	1,80	34,80	76.4	68.0	9,00	2,01	72.0	5,80	55.46	11,28					0,012	0.015
2,10	1,82	6,36	7.0	1,78	34,60	75.0	68.5	7,80	1,85	70.2	6,20	53.00	10,99	5,96	1,92	12,72	1,52	0,013	0.015
2,10	3,64	6,36	7.0	1,77	34,50	74.5	67.8	8,00	1,72	68.4	7,80	52.47	10,48	6,01	1,94	12,52	2,28	0,013	0.015

From table 7.18, it's concluded that:

1) Influence of dunite fines addition on strength of sinter:

Test data show that the strength of sinters with 3 kinds of basicity declines remarkably along with increase of dunite fines addition.

At the basicity of 1.0; the sinter without dunite fines addition has a tumbler index of 63.3%. As the dunite fines addition increases to 1.82% and 3.64%, the tumbler index declines to 62.7% and 61.3% respectively.

At the basicity of 1.7; the sinter without dunite fines addition has a tumbler index of 67.7%. As the dunite fines addition increases to 1.82% and 3.64%, the tumbler index declines to 67.3% and 66.0% respectively.

At the basicity of 2.1; the sinter without dunite fines addition has a tumbler index of 72.0%. As the dunite fines addition increases to 1.82% and 3.64%, the tumbler index declines to 70.2% and 68.4% respectively.

2) Influence of dunie fines addition on sinter cake yield, final product ratio and productivity:

At different basicity of sinter, the sinter cake yield, final product ratio and productivity decline along with increase of dunite fines addition.

3) Influence of dunite fines addition on TFe and FeO in sinter:

At different basicity of sinter, Both Fe and FeO content are decreased with increase of dunite fines addition.

7.4.9.5 Study on Test for Negative Pressures

According to sintering tests under different conditions, the vertical sintering speed is very fast because of good bed permeability. Under the premises that all kinds of technical index are kept unchanged, it's to be considered whether there's the possibility that the negative pressure is reduced. Therefore, tests at the basicity of 2.1 have been done using changed negative pressures while the other sintering conditions are kept unchanged. Test results are shown in table 7.19

Table 7.19 Sintering test results using changed negative pressures.

Basicity CaO/SiO_2	Ignition negative pressure Pa	Sintering negative pressure Pa	Vertical sintering speed mm/min	Sinter cake yield %	Productivity $t/m^2.h$	Tumble index +6.3mm %	Abrasion index -0.5mm %	Final sinter ratio %
2.1	5880	11760	34.8	76.6	2.01	72.0	5.8	68.0
2.1	4900	9800	29.8	76.3	1.70	70.3	8.0	68.9

From table 7.19, it's concluded as follows:

When sintering negative pressures are reduced, the tumbler index, vertical sintering speed and productivity are all decreased remarkably while the sinter cake yield and final sinter ratio keep unchanged. In consideration of good indexes for testing and production, the negative pressure for sintering should be selected as 11760 Pa.

7.4.9.6 Test of Metallurgical Properties of Sinter

To understand the reaction of sinter in blast furnace during iron making, tests using sinters having different basicity and different dunite fines additions have been done to know metallurgical property of sinter. The final sinter is screened to get the fraction > 12.5 mm, which is carefully crushed until all particles pass through 16.0 mm screen. The sample is then screened to remove the fraction > 12.5 mm and the fraction < 10.0 mm. The fraction 10.0 mm~l2.5 mm is mixed and uniformly selected as sample to be used for test.

7.4.9.7 Reducibility Test

The 500 gr. sample (surface is made even) is put into reducing tube on the porous plate, and the top of the tube is then sealed. The inert gas N_2 is given into reducing tube with a flow of 5 L/min under normal condition. The tube is put into reducing oven and hung in the center of electronic weigher. When the tube is put into reducing oven, the temperature inside the oven must be less than 200 °C. The oven is heated with a heating speed less than 10 ^oC/min. When the sample reaches 900 °C, the flow of N_2 is increased to 15 L/min and the temperature keeps unchanged for 30 minutes to get a constant weight of sample and a constant temperature of 900 °C + 10 °C. When the constant temperature of 900 °C ± 10 °C is achieved, the reducing gas (CO: N_2 = 30 %: 70 %) with a flow speed of 15 L/min replaces N_2. The reduction continues for 180

minutes. After completion of reduction, the sample is cooled to below 100 °C under protection of N_2. Then, the sample is taken out and the reducibility is calculated through weight reducing method.

7.4.9.8 Test for Low Temperature Disintegration Behavior During Reduction

The test is done through static reduction and cold tumbler. The 500 gr sample (surface is made even) is put into reaction tube on the porous plate, and the top of the tube is then sealed. The reaction tube is put into disintegration testing oven. The inert gas N_2 is given into reacting tube with a flow of 5 L/min under normal condition. The oven is heated with a heating speed less than ^{o}C/min. When the oven temperature reaches 500 ^{o}C, the flow rater of N_2 is increased to 15 L/min and the temperature keeps unchanged for 30 minutes to get a constant weight of sample and a constant temperature of 500°C+ 10°C. When the constant temperature of 500 °C ± 10 °C is achieved, the reducing gas (CO: CO_2: N_2 = 20: 20: 60) with a flow speed of 15 L/min replaces N_2. The reduction continues for 60 minutes. After completion of reduction, the sample is cooled to below 100°C under protection of N_2. Then, the sample is taken out and put into 130 mm (inner diameter) X 200 mm (length) tumbler, which rotates (30 r/min) for 300 revolutions. The sample is then taken out and screened to fractions of +6.3 mm, +3.15 mm and –0.5 mm, and the percentage of different fractions are calculated.

7.4.9.9 Test of Behavior of Sinter in the Reduction Softening Test Under Load

The device for testing is a vertical electric oven filled with a 48 mm (inner diameter) X, 65 mm (outer diameter) high purity graphite crucible where the test is carried out. The 200 gr sample totaling a bed height about 70 mm is divided into two layers, to which 10 gr coke are added. The grain size of both sample and coke is 10.0 mm ~ 12.5 mm. The heating mode is as follows:

Before reaching 900 °C, the heating speed is 10 °C/min; the temperature 900 °C keeps unchanged for 30 minutes; the protective gas is N_2 with a flow rate of 5 L/min. After reaching 900 °C, the heating speed is 5 °C/min; the reduction gas (CO=30% **±**

1.0%, N_2=70% ± 1.0%, $CO_2 \leq 0.5\% \pm 0.1\%$, $H_2 \leq 1.0 \pm 0.1\%$) is added until the test is finished. The loading for testing is 1.0 kg/cm^2.

The parameters such as change of temperature and shrinkage of bed layer are recorded.

7.4.9.10 Test Results and Their Analysis

The results of test for metallurgical properties of sinter are shown in table 7.20 where No.l sample represents sinter with a basicity of 1.0, No.2 sample sinter with a basicity of 1.7, No.3 sample sinter with a basicity of 2.1, No.4 sample sinter with a basicity of 1.0 and an dunite fines addition of 3.64%, No.5 sample sinter with a basicity of 1.7 and an dunite fines addition of 3.64%, and No.6 sample sinter with a basicity of 2.1 and an dunite fines addition of 3.64% (the sample numbering in table 7.38 is explained in the same meaning). Sinter indexes obtained under working mode according to above mentioned parameters are listed in table 7.21 and table 7.22.

From table 7.20, it's concluded as follows:

1) The reducibility of sinter increases with the increase of basicity of sinter. The reducibility of sinter increases remarkably from 60.43% to 72.1% and 76.34% while the basicity of sinter increases from 1.0 to 1.7 and 2.1.
2) The low temperature reduction degradation index RDI + 3.15 mm increases with increase of basicity of sinter. With the increase of sinter basicity from 1.0 to 1.7 and 2.1, the RDI+3.15 rnm. increases from 70.35% to 78.67% and 81.11%.
3) The reducibility and RDI+3.15mm of sinter with addition of dunite fines are increased.

Table 7.20 Indexes of metallurgical properties of sinter

S.N.	RDI under low temp.			RI %	loading softening melting dropping						
					4% deformation	10% deformation	40% deformation	50% deformation	Temp. Range for softening & melting	Temp. for beginning of dropping	ΔPmax kPa
	+6.3mm %	+3.15mm%	-0.5mm %		°C	°C	°C	°C	°C	°C	
1	45.35	70.35	6.17	60.43	1099	1152	1219	1271	172	1450	6.0
2	58.10	78.67	3.89	72.10	1133	1170	1285	1315	182	1465	5.7
3	61.83	8 1,11	4.55	76.34	1150	1180	1292	1324	174	1480	5.69
4	65.08	82.72	3.47	69.47	1168	1190	1305	1359	191	1518	5.0
5	65.58	83.99	3.23	75.84	1177	1212	1346	1389	212	1526	5.0
6	67.49	84.13	3.26	78.09	1 185	1232	1385	1441	256	1550	4.4

Table 7.21 Sinter indexes obtained under working mode according to above-mentioned parameters.

Basicity CaO/SiO_2	Proportioning ratio (dry) %										Bed depth (mm)	Moisture of raw mix (%)	Bulk density of raw mix (t/m^3)	Size texture of raw mix % (mm)			Mode of ignition		
	I.O.E	I.O.A	I.O.B	I.O.C	I.O.D	BF flue dust	Iron bearing sludge	Limestone fines	Coke fines	Sinter returns				+10	10-3	-3	Temp. °C	Time min	Neg. pressure
1.7	42.66	5.97	9.19	3.22	4.59	2.2	2.0	15.27	5.90	9.0	600	7.0	1.83	10	60	30	1150-1200	1.0	5880
2.1	41.06	5.75	8.84	3.10	4.42	2.2	2.0	17.27	6.36	9.0	600	7.0	1.80	10	63	27	1150-1200	1.0	5880

Table 7.22 Sinter indexes obtained under working mode according to above-mentioned parameters.

Negative pressure for sintering	Vertical sintering speed min/min	Sinter cake yield %	Final sinter ratio %	Sinter returns quantity %	Productivity		Size texture of sinter mm%						Chemical composition of product sinter %							
					$t/m^2.h$	$t/m^2.d$	+40	40-25	25-15	15-12.5	12.5-10	10-5	TFe	FeO	$Si0_2$	Al_2O_3	CaO	MgO	S	P
11760	34.8	77.8	69.0	8.8	1.88	45.12	35.77	17.66	15.40	8.98	6.97	15.24	56.71	12.60	6.63	1.92	11.04	0.92	0.013	0.014
11760	34.8	76.6	68.0	8.6	2.01	48.24	25.18	21.58	19.42	10.07	6.47	17.28	55.46	11.28	6.47	1.95	13.38	0.88	0.012	0.015

Desulfurization efficiency %	Tumbler strength			Low temp, reduction degradation index (RDI)			Reduction index (RI)	Properties of loading softening-melting dropping						
	Tumbler index +6.3mm%	Screening index +5.0mm%	Abrasion Index -0.5mm%	+6.3mm %	+3.15mm %	-0.5mm %		4% deformation "C	10% deformation °C	40% deformation "C	50% deformation "C	Temp, range for softening-melting °C	Beginning of dropping °C	A Pmax
88.1	67.7	74.7	7.0	58.10	78.67	3.89	72.10	1133	1170	1285	1315	182	1465	5. 7
86.1	72.0	76.9	5.8	61.83	81.11	4.55	76.34	1150	1180	1292	1324	174	1480	5.6

7.5 Statistic Quality Control of Sinter Samples

Statistical process control techniques are used for iron and steel plants in which data are measured for certain periods to increase physical, chemical and reducibility properties of sinter production. Thus, when some important advantages are provided in terms of control and applicability. In this study, static and dynamic test low temperature reduction disintegration and reduction under load of the samples taken out regularly from sinter plant of Isdemir were investigated to determine capability of the process. For each parameter, X-R control charts using moving range method were drawn. Thereby, deviations of lower and upper limit values were determined. It was observed that quantity of raw ores and grades should be fixed using suitable homogenization.

7.5.1 Statistic Quality Control of Sinter Samples

Statistics Process Control (SPC) techniques are used at the specified periods at the substance preparation foundation from which datum is obtained to increase the production's quality and output. So, some important easiness is provided in terms of time control and practicability. Statistic process control is a way that collected with the aim of minimizing faulty production by preventing production out of standard and providing the production activities being done conveniently to quality specification that determined in the beginning (Montgomery, 1997)

Statistic Process Control is seen as a kind of application to production of method of illustration.With the help of statistic process control the quality of the production can be under control and developed, then the production of bad outputs that don't fit the demanded standard statistic process control isn't an examination of it. The important difference between quality examination and statistic quality control is that statistic quality control's following the process of production and quality examination's control of the production. With quality examination the production that doesn't fit the standard or norm is isolated from other productions. As the production proportion that is out of standard rises, the loss of the manufacturer increases. The reason of this, these productions that are out of standard sold more cheaply or never sold. The aim of statistic quality control can be summarized as

presenting the proportion of faulty productions the lowest level by keeping process of production under control (Montgomery, 1997)

There are two reasons for the use of statistical methods in quality control:

a) Examining to mass or heap that need to be controlled at % 100 rates is generally impossible or not economic.
b) A measureable quality property of produced production change continuously and this change shows randomly construction on condition that staying between some limits.

7.5.2 X-R Tables

X-R tables are changeable tables. They consist of two lines graphic. X graphic consists of these sections.

- a line graphic
- a middle line (average)
- An upper control limit line
- A lower control limit line

As each point is an average of a few values that named as $\bar{X}$ (X average). Each value is evaluated as an only measurement. For whichever $\bar{X}$ table, (n) called model bigness is always the same. Middle line $\bar{\bar{X}}$ called big average or $\bar{\bar{X}}$. This is the average of averages. The lines of control limit, upper control limit of $\bar{X}$ is called (UCL $\bar{X}$) and lower control limit of $\bar{X}$ is called (LCL $\bar{X}$). These aren't related with specification. They are the natural limits of average value. X table that mentioned earlier is only, for only value is a changeable table. The principal reason for being used of averages at X- table; even 4 different measurement's average; it's providing of being guessed what process average or process aims.

7.5.3 Preparation of Control Charts

1) Model bigness is shown as “n”. At here, model bigness is the total number of measurements that taken one after another at one time. At here in the laboratory study if it’s thought that each analysis made 3 times as marked by repetition n=3 D_3=0 D_4=2,575 A_2=1,023 values are taken from table of control limit values.
2) For each model group the values of interval are calculated. Interval value is shown as the letter of ‘R’. It is the difference between the biggest value and the smallest value.
3) X graphic shows the change of averages, R graphic shows that this change’s being whether consistent or not from a unit to another.
4) If at the same times both X and R graphics are out of control, as a general rule R graphic is considered firstly. The reason that makes R graphic out of control can explain why X graphic is out of control.
5) At R table being a little changeability or not is evaluated as a demanded situation. But as it may not be always right, it may be sign of something that doesn’t go well. At working business enterprise it must be studied carefully that some changes of the process how much effect the changeability amount.
6) Control limits are natural limits of the process. If the undulations at the process increase (because of iron ore, coke breeze, basicity etc.) the control limits will reflect this and widen. The homogenity at the raw material at process and at business enterprise conditions as stability provided control limits are narrowed.

7.5.4 The Relation of Quality Control with Statistic

There are some matters in that using statistic ways, quality control is useful:

For being provided conditions the same, it determines necessary control means.

a) Determines the necessary control tools to ensure the conditions remain the same in the company.
b) With a certain degree of reliability of the results for the evaluation shall determine how many measurements need to be done.

c) To evaluate the most productive way specified number of measurement. It helps the experiments being prepared at the best way.
d) It encourages information being enrolled as smoothly and sensitively.
e) It produces the importance of determining of all conditions by taking into consideration that belongs to the time of collected information.

7.5.5 Material and Method

At this study statistic process control is made by being used sinter material's technological test results that are produced at İskenderun Iron and Steel Works. Limit values must be calculated separately for X and R control chart. While, X control charts are used the deviation from the mean value R control graphics are used to define the values of leaving from homogenity. At the drawing of control graphics 3 limit values are used. These are; lower limit values (LCL), upper control limit values (UCL) and middle value line. Being used parameters; R: change interval (range) $\bar{X}$: average of the data (CL), R: average of interval values and X is the division of total average to number of averages. When changing interval (range) is calculated as lower group number is little calculations are made by using the way of moving range.

7.5.6 X Diagrams

Control diagram which is about averages is used at observation of the changings that constitute at model's averages that are changing continuously and are measureable. Whatever the kind of variable is it is necessary that at the constitution of a control graphic always 3 principal values are calculated. Center line (CL), upper control limit (UCL), lower control limit (LCL). Figure 7.11 indicates X average control schema.

$\bar{X} = (X_1+X_2+X_3+\ldots\ldots+X_n)/n$

$UCL = \bar{X}+A2\ \bar{R}$

$LCL = \bar{X}- A2\ \bar{R}$

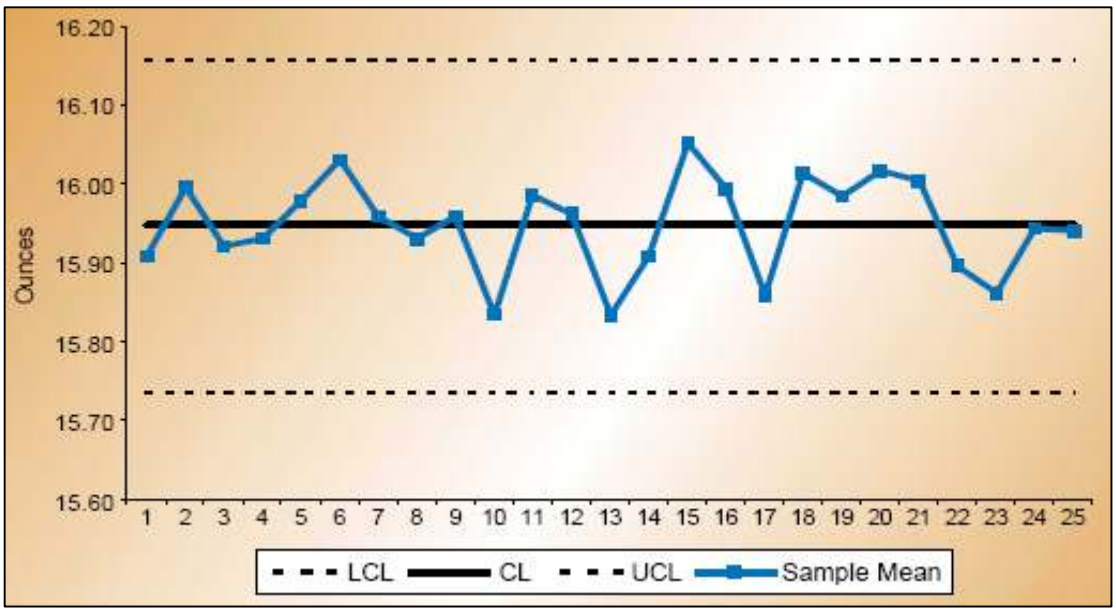


Figure 7.11 X average control schema (Montgomery, 1997).

7.5.7 R Diagrams

R diagrams are used with the aim of observing the changing that are at changing interval and belong to the models, the most widespread tool at studying of quality's scattering (Montgomery, 1997). Table 7.23 demonstrates coefficient values for control graphics. Table 7.24 gives information about R40 Dp and Dh values of sinter samples. Table 7.25 provides information about the physical and chemical analysis of sintered samples

$R= (R_1+R_2+R_3+\ldots\ldots+R_n)/n$

$CL= \bar{R}$

$UCL=D_4\ \bar{R}$

$LCL=D_3.\ \bar{R}$

Table 7.23 Coefficient values for control graphics (Control Limit Factors) (Montgomery, 1997).

Sample size (n)	A_2	D_3	D_4	d_2
2	**1,88**	**0**	**3,267**	**1,128**
3	**1,023**	**0**	**2,575**	**1,693**
4	**0,729**	**0**	**2,282**	**2,054**
5	**0,577**	**0**	**2,115**	**2,326**

Table 7.24 R_{40} Dp and Dh values of sinter samples.

Sample No:	R_{40} (Reduction Speed)	Dp (Pressure variation)	Dh (Height variation)
	gr-O_2/min.	kPa	%
Specifications	Min. 1,8	Max. 1	Min. 8-10
S 1	1,40	1,20	0,48
S 2	1,35	1,20	0,63
S 5	1,11	1,40	0,88
S 6	1,39	1,50	0,82
average	1,31	1,33	0,70

Table 7.25 Physical analysis of sinter samples.

Sample No:	Tumbler index + 6,35 mm	Tumbler index -0,600 mm	RDI+6,35 mm	RDI-0,500 mm	LTD + 6,35 mm	LTD - 0,500 mm
Specifications	min. 69		%20-25	5%	%10-15	14%
S1	68,60	3,54	33,94	3,79	33,04	9,33
S2	69,30	4,02	34,22	5,25	36,75	10,02
S3	64,76	6,28	30,52	5,33	62,47	4,52
S4	62,41	3,98	28,33	4,36	56,67	4,20
S5	67,24	4,19	50,21	4,38	57,67	7,49
S6	73,03	3,90	42,93	3,75	52,81	7,71
S7	70,72	4,24	44,15	5,32	-	-
Average	68,01	4,31	37,76	4,60	49,90	7,21

7.5.8 Process Description

In this study sinter material that's produced at ISDEMIR. Statistic quality control analyses are made for determined parameters by used its 7 representative models' technologic test results. If each datum is thought to be calculated repeatedly 3 times (n=3) according to table 7.23 D_3=0.00 D_4=2,575 A_2=1,023 values are used. For parameters that are measurement are made with foundation. Results that are at below are obtained.

If the points functioned to conrol graphic are getting long as being between control limits, it is assumed that process is under control and no struggle is needed. With this by not going out of the control limits, point's showing sistematic tendency

or going out of the control limits of at least one of the points sign out of control situation. At this situation being founded of the fault by interfering the production and being made of arrangements are necessary (Juran&Godfrey, 1998). At this study as quality property because of sinter's technologic test parameter value is taken into consideration average (X) and changing interval (R) graphics will be used. So these graphics conrol limits are calculated as given at Table 7.26

7.5.9 Made Tests on X-R Values, Graphics and Comments

Table 7.26 Reducibility values of X and R for R_{40}

Sample	X	R
S1	1,40	0,00
S2	1,35	0,05
S5	1,11	0,24
S6	1,39	0,28
$\bar{X}$	**1,31**	
$\bar{R}$		**0,14**
LCL	**1,17**	**0,00**
UCL	**1,46**	**0,37**

1,023	**A2**
0	**D3**
2,574	**D4**

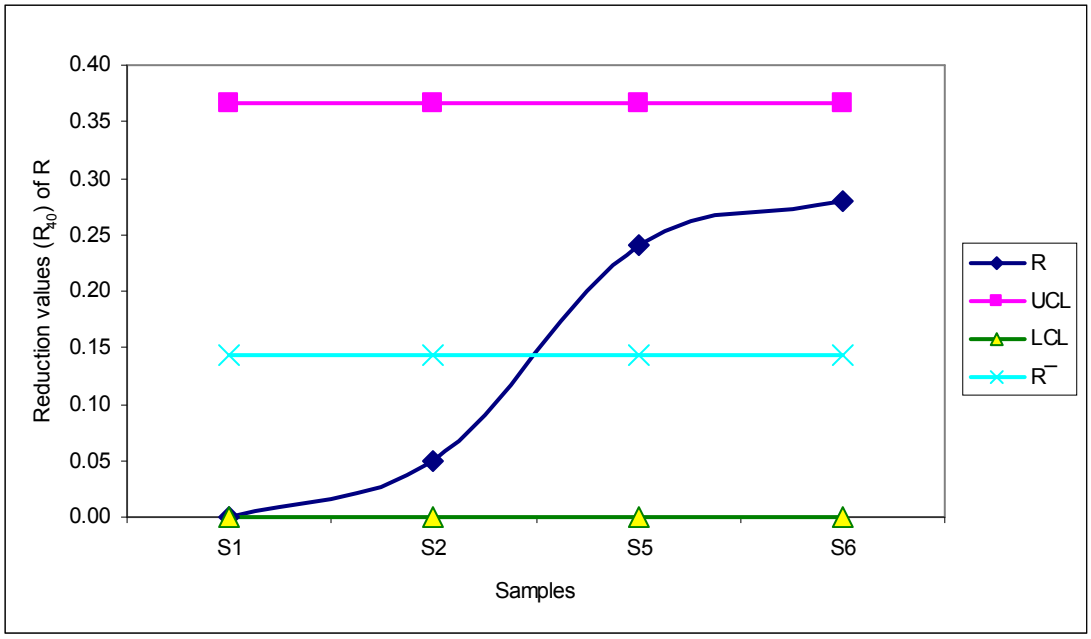


Figure 7.12 R diagrams of R40 values.

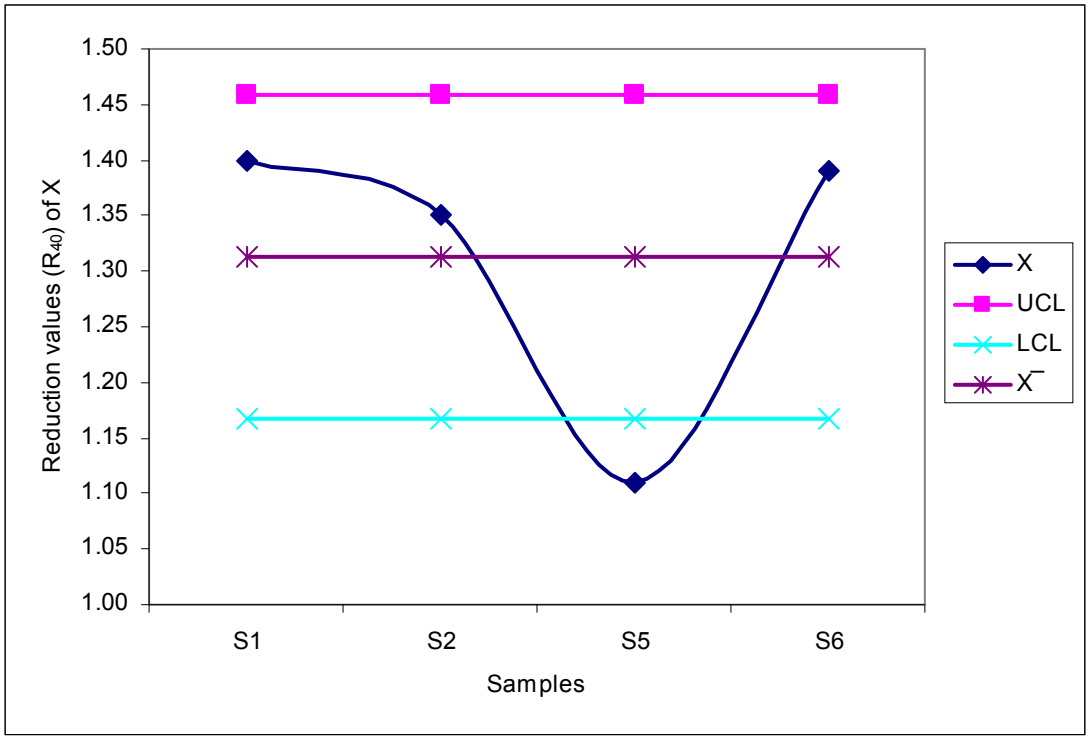


Figure 7.13 X diagram of R_{40} values.

At figure 7.13 as seen at X Diagram R_{40} values that belong to S1, S2, S6 sinter models are between under control to upper control limit values. But, at S5 test lower control decreases under limit values. At blast furnace because of being used of this sinter in the course of reduction the loss of oxygen is low, the time of function will be long and metallurgic coke consumption will increase, blast furnace output will be low. At the blending of this sinter it is seen that being used iron ores have low reduction and more magnetite. Blast furnace; can manage the sinter that reduction is low, at this process by using lump ore and pellet material that's reduction are high. As seen at R diagram the values are between under and upper control limit values. The R control graphic determines changing from unit to unit, and the values of leaving from homogeneity.

Table 7.27 Dp values of X and R.

Testler	X	R
Sinter 1	1,20	0,00
Sinter 2	1,20	0,00
Sinter 5	1,40	0,20
Sinter 6	1,50	0,10
$\bar{X}$	**1,33**	
$\bar{R}$		**0,08**
LCL	**1,25**	**0,00**
UCL	**1,40**	**0,19**

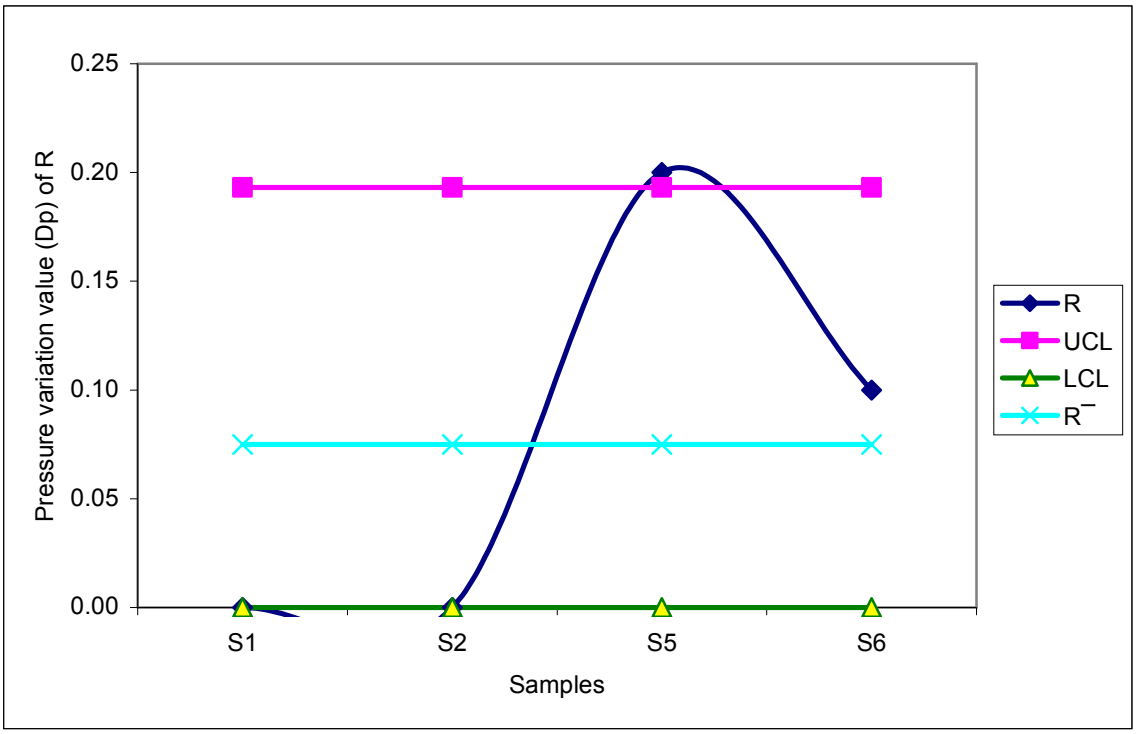


Figure 7.14 R diagram of Dp values.

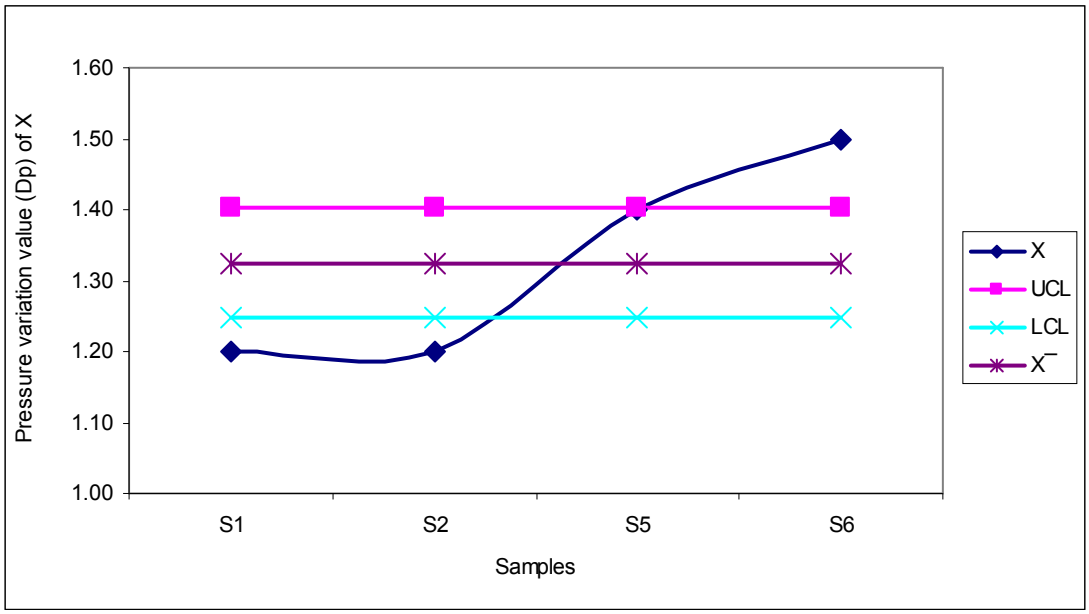


Figure 7.15 X diagram of Dp values.

As seen at figure 7.15 at, X diagram Dp values at test S1 and S2 are under the control limit. Dp values of S6 is upper the control limits. S1 and S2 sinters used at blast furnace because of high heat in the oven and gases the pressure difference is little.So blast furnace can use the lump ore and pelet materials that's pressure difference is high at its process. When using number of S6 sinter because of the pressure difference is high by functioning at adverse direction blast furnace process can be managed. But, it is always advantageous to give substance that has the same homogenity and characteristic property to the blast furnace. It takes the time to fit in varied process transition. As seen the R diagram S5 test value is above of upper limit value.

Tablo 7.28 Dh values of X and R.

Tests	X	R
Sinter 1	0,48	0,00
Sinter 2	0,63	0,15
Sinter 5	0,88	0,25
Sinter 6	0,82	0,06
$\bar{X}$	**0,70**	
$\bar{R}$		**0,12**
LCL	**0,59**	**0,00**
UCL	**0,82**	**0,30**

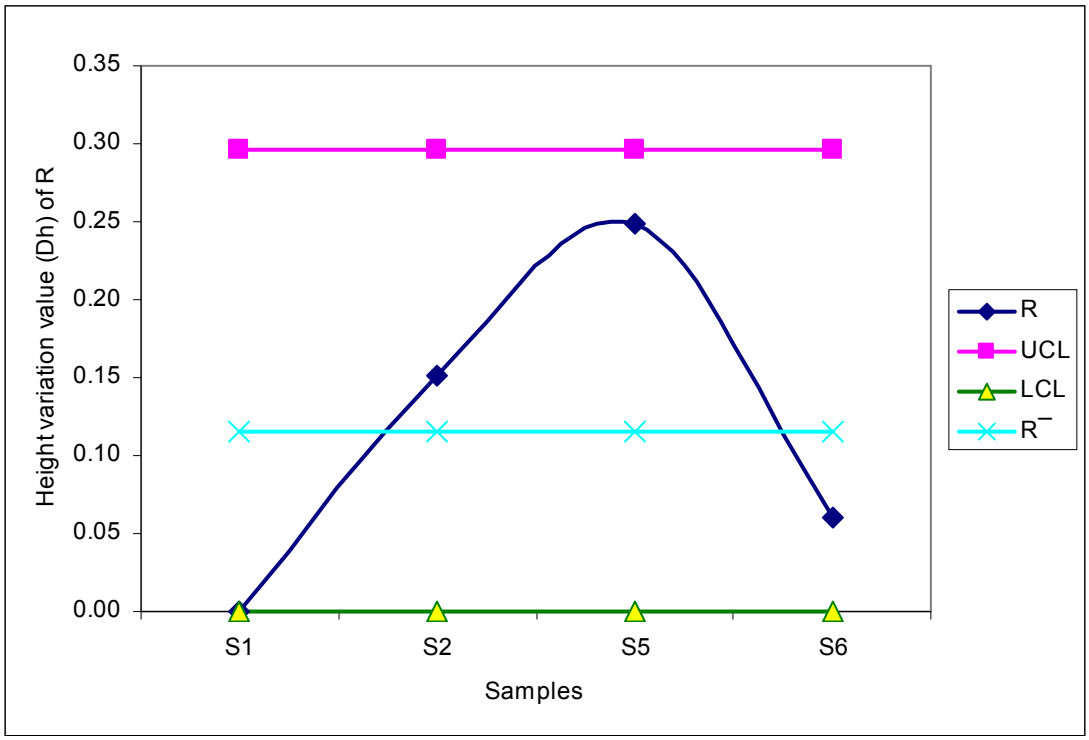


Figure 7.16 R diagram of Dh values.

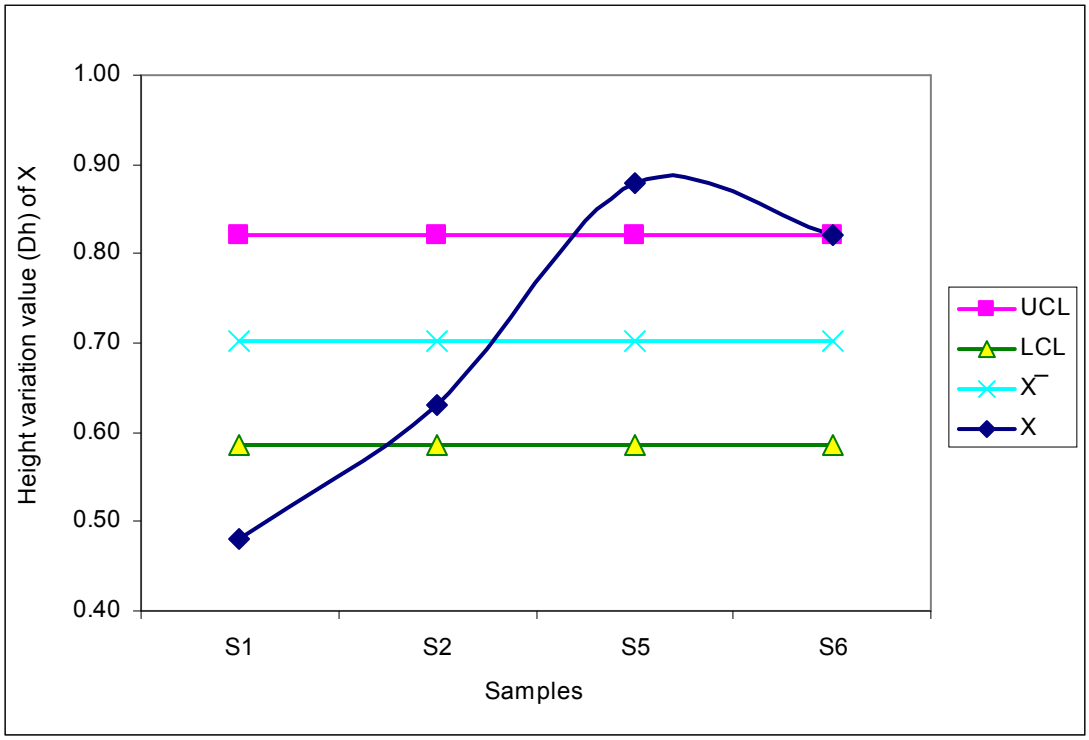


Figure 7.17 X diagram of Dh values.

As seen at figure 7.17, at X diagram at S1 and S5 sinter blending Dh values overflow out of upper and under control limit values. S1 sinter used at blast furnace softening and loss of height is little because of high heat and gases in the blast furnace and loss of oxygen compared to height of the sinter's first position.So blast furnace can use lump ore and pelet materials that's Dh value is high at its process. When using S6 sinter because of its Dh value is high blast furnace process can be managed by doing function at adverse direction. But it is always advantageous to give substance that has the same homogenity and characteristic property to the blast furnace. As seen at R diagram the values are between upper to under the control limit values.

Tables 7.29 RDI values of X and R for +6,35mm

Tests	X	R
Sinter 1	33,94	0,00
Sinter 2	34,22	0,28
Sinter 3	30,52	3,70
Sinter 4	28,33	2,19
Sinter 5	50,21	21,88
Sinter 6	42,93	7,28
Sinter 7	44,15	1,22
$\bar{X}$	**37,76**	
$\bar{R}$		**5,22**
LCL	**32,42**	**0,00**
UCL	**43,10**	**13,44**

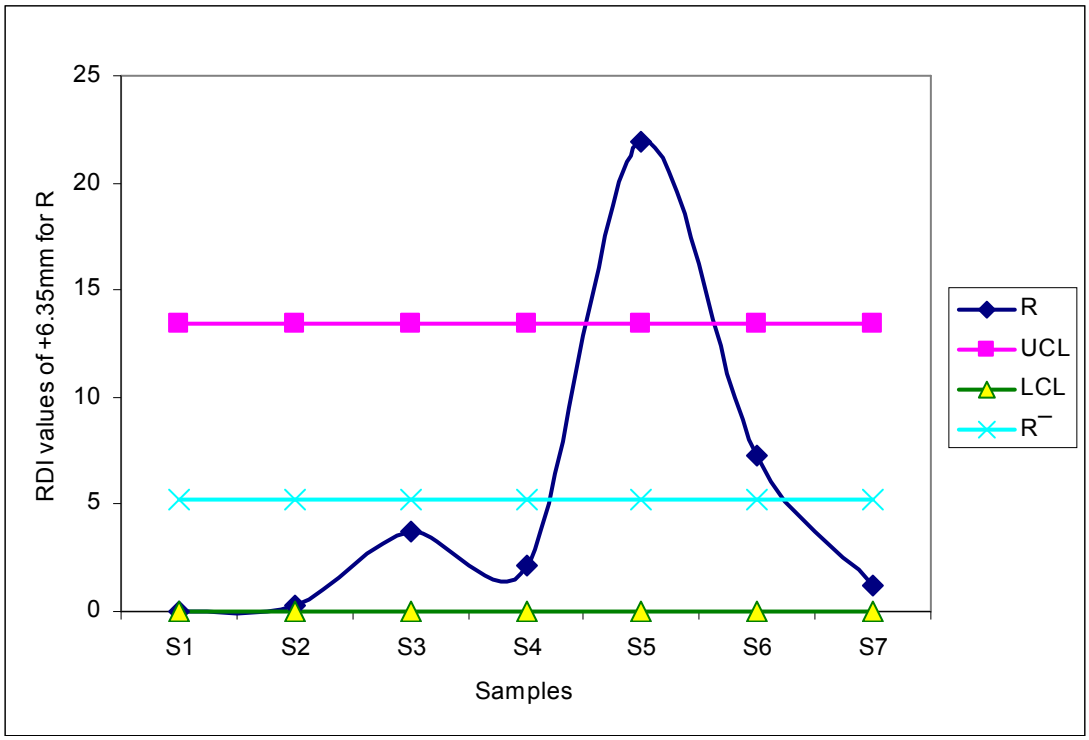


Figure 7.18 R diagram of RDI values for +6.35 mm

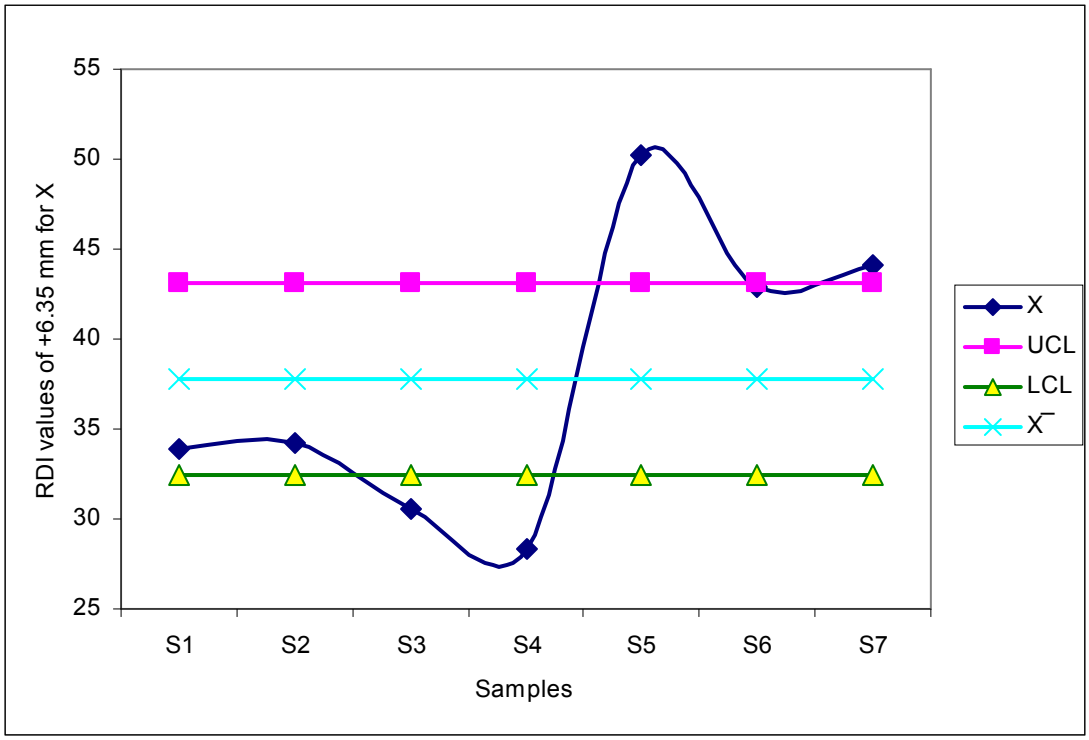


Figure 7.19 X diagram of RDI values for +6.35 mm

As seen at figure 7.19 at X diagram RDI values that belong to the sinter at S3, S4 and S5 test overflow out of upper and under limit values. At upper charge part in the S3 and S4 number sinter oven used at blast furnace (150-500 ^{o}C) with the effect of

high heat and gases encountering, because of at its structure's cracks piercing and physical changing is little . S6 sinter has adverse situation. As seen R diagram the values at S2, S4, S5 and S7 sinter overflow out of upper and under control limit values.

Table 7.30 RDI values of X and R for -0,5 mm.

Tests	X	R
Sinter 1	3,79	0,00
Sinter 2	5,25	1,46
Sinter 3	5,33	0,08
Sinter 4	4,36	0,97
Sinter 5	4,38	0,02
Sinter 6	3,75	0,63
Sinter 7	5,32	1,57
$\bar{X}$	**4,60**	
$\bar{R}$		**0,68**
LCL	**3,91**	**0,00**
UCL	**5,29**	**1,74**

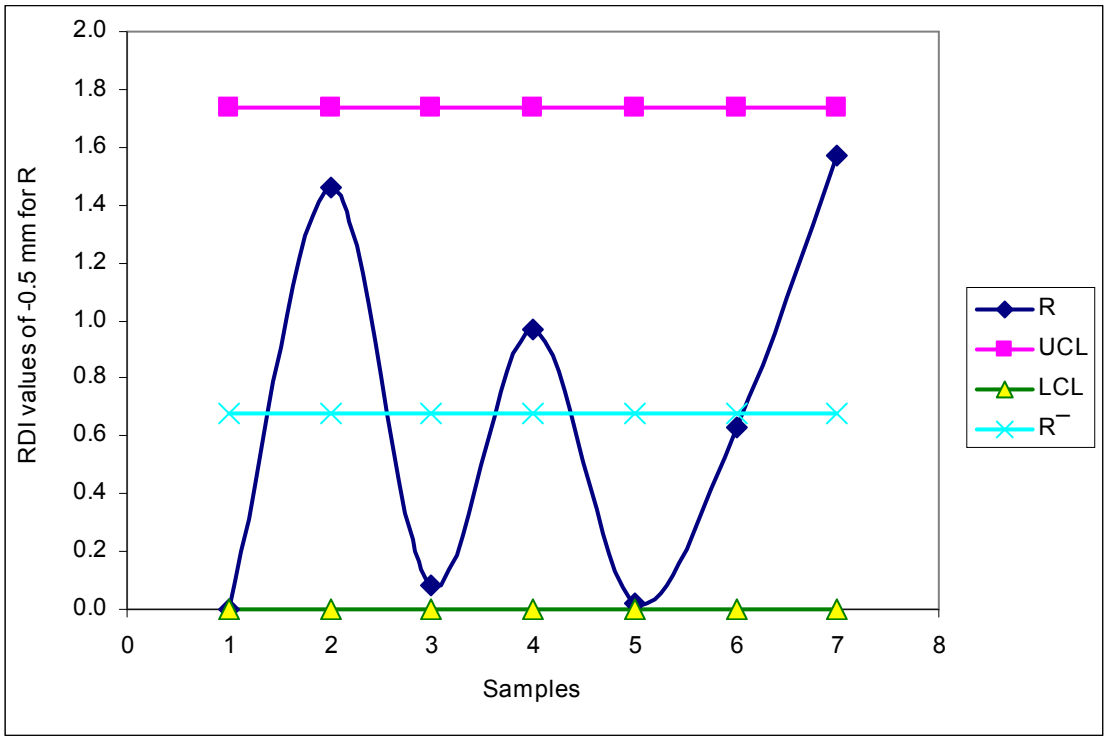


Figure 7.20. R diagram of RDI values for -0,5 mm .

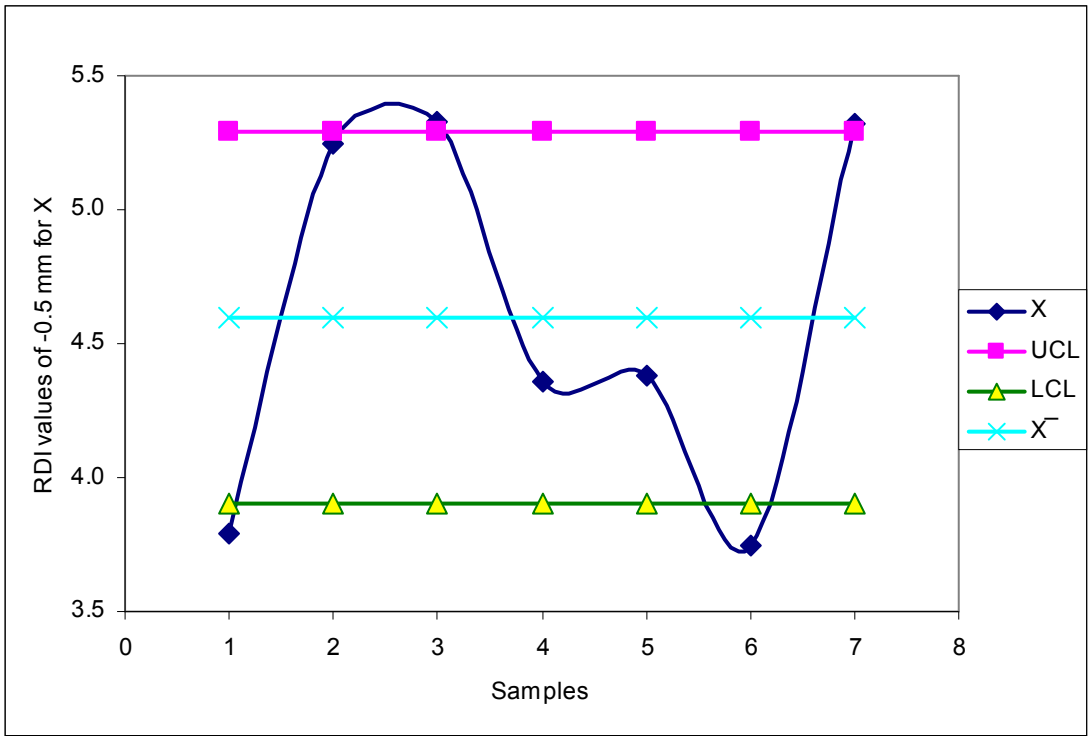


Figure 7.21. X diagram of RDI values for - 0,5 mm.

As seen at figure 7.21 at X diagram RDI values that belong to S1 and S6 sinter tests overflow out of the under control limit values. As S6 sinter's $RDI_{-0.5}$ value used at blast furnace is high the sinter is broken in the blast furnace at low heat in the course of reduction at that measure. This sinter in the course of reduction cause to change of the charge material's particle dimension dispersion that is loaded to the blast furnace. In the event of sinter's breaking index $RDI_{-0.5}$ is high, the gas permeability at the body of oven decreases, passing of the gases from the bed correctly gets difficult and dust amount in chimney gas increases. So it is demanded that breaking index $RDI_{-0.5}$ is low. Being low of the breakhing index of S1 number sinter is good for blast furnace working parameter. As seen at R diagram in figure 7.20 all values are between upper and under limit values.

Table 7..31 LTD values of X and R for +6,35 mm

Testler	X	R
Sinter 1	33,04	0,00
Sinter 2	36,75	3,71
Sinter 3	62,47	25,72
Sinter 4	56,67	5,80
Sinter 5	57,67	1,00
Sinter 6	52,81	4,86
$\bar{X}$	**49,90**	
$\bar{R}$		**6,85**
LCL	**42,90**	**0,00**
UCL	**56,91**	**17,63**

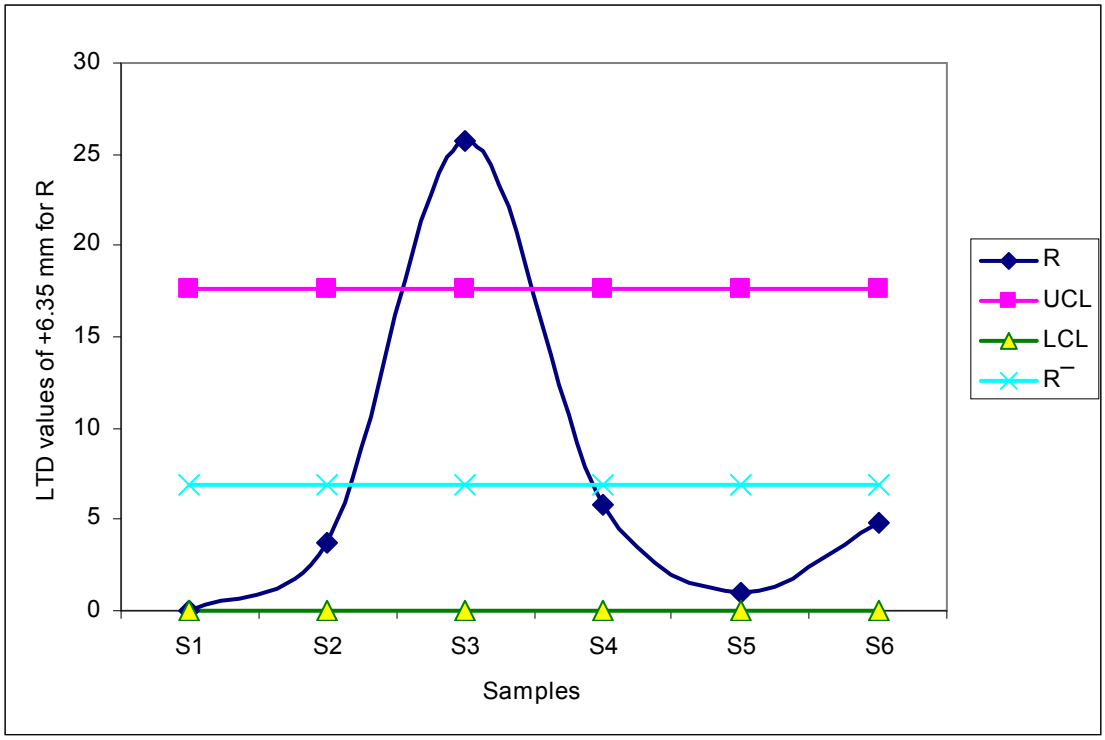


Figure 7.22 R diagram of LTD values for +6.35 mm

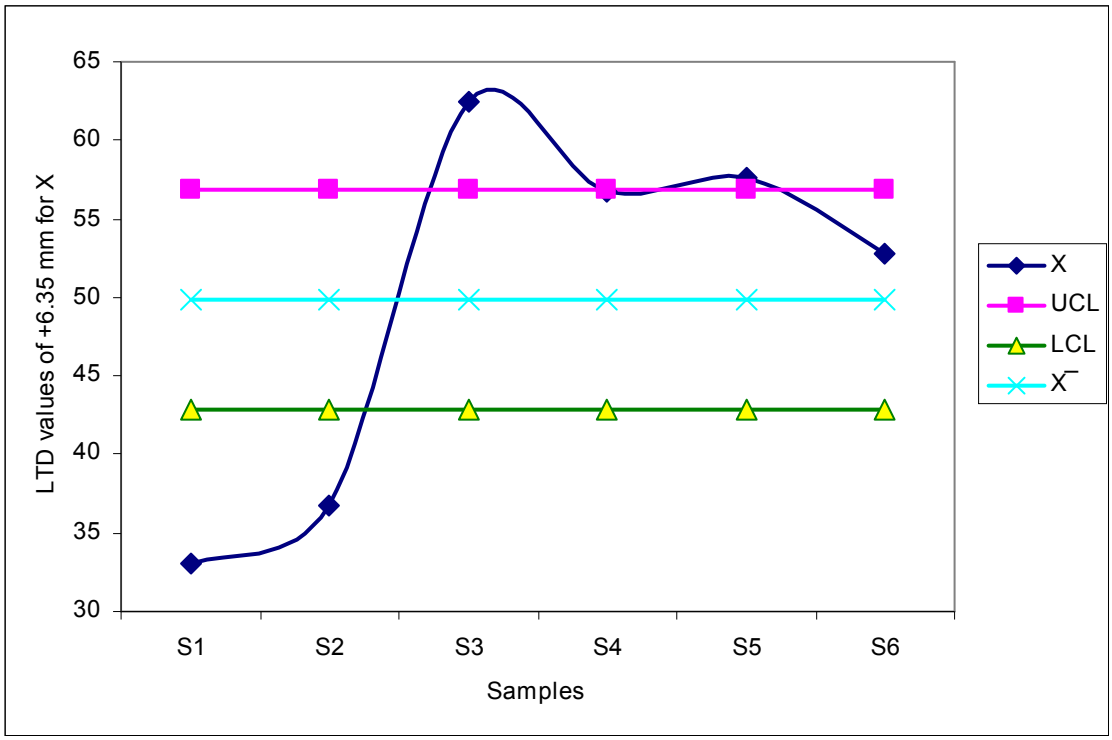


Figure 7.23 X diagram of LTD values for + 6.35 mm

As seen at figure 7.23 at X diagram $LTD_{+6.35}$ mm values belong to S1, S2 and S3 sinters overflow out of upper and under control limit values. S1 and S2 sinter used at blast furnace at upper charge part in oven (150-500 ^{o}C) with the effect of gases and high heat met firstly and because of its crack at in its construction, smashing and breaking is so much. At the S3 sinter there is an adverse situation. As seen at R diagram at S3 sinter values overflow out of upper control limit values.

Table 7.32 LTD values of X and R for -0,5 mm

Testler	X	R
Sinter 1	9,33	0,00
Sinter 2	10,02	0,69
Sinter 3	4,52	5,50
Sinter 4	4,20	0,32
Sinter 5	7,49	3,29
Sinter 6	7,71	0,22
$\bar{X}$	**7,21**	
$\bar{R}$		**1,67**
LCL	**5,50**	**0,00**
UCL	**8,92**	**4,30**

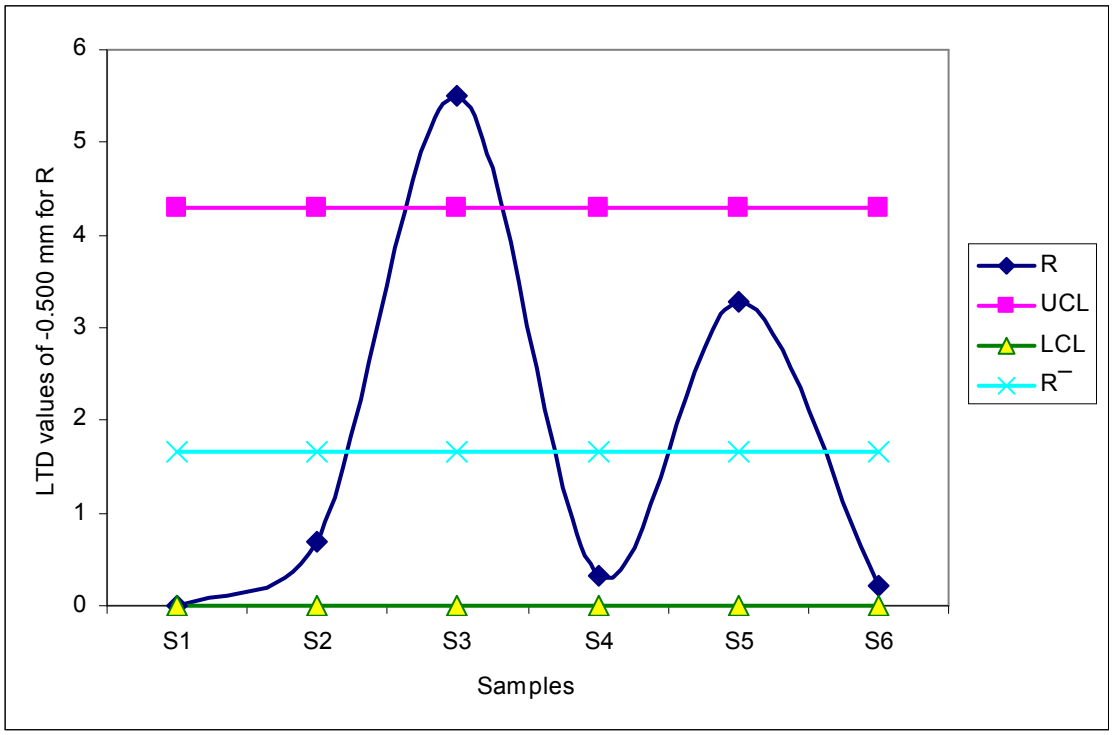


Figure 7.24 R diagram of LTD values for -0.5 mm

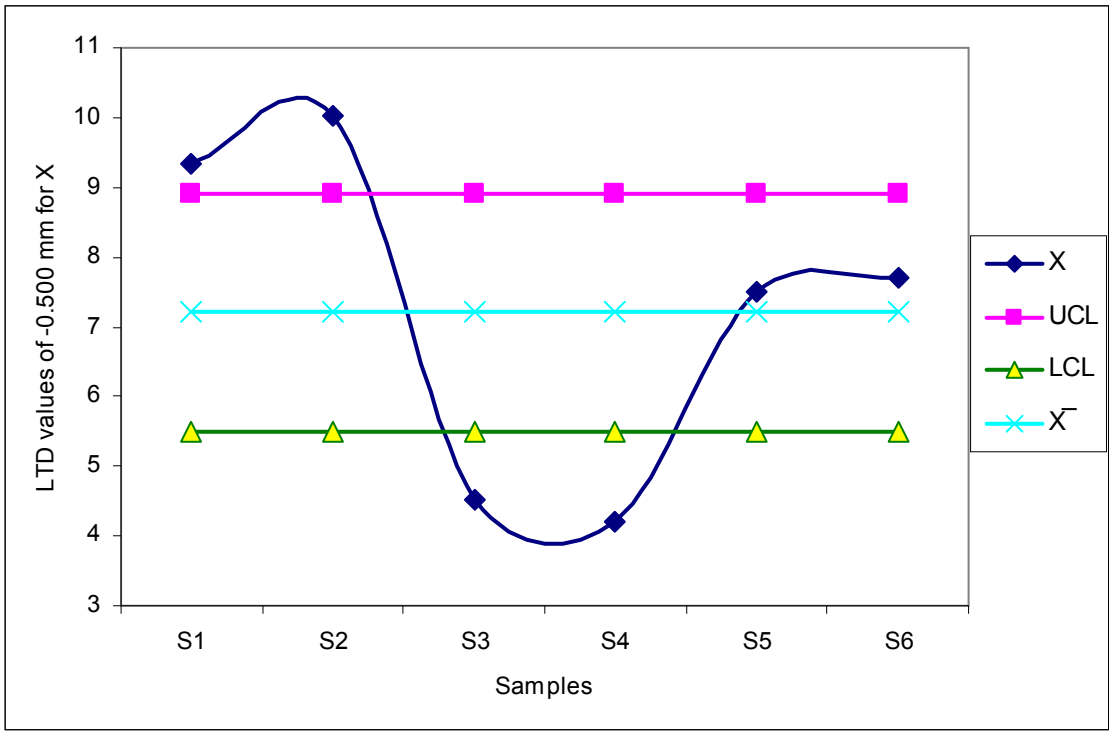


Figure 7.25 X diagram of LTD values for -0.5 mm

As seen at figure 7.25 at X diagram S1, S2 ,S3 and S4 $LTD_{-0.5}$ mm of sinter underscreen values overflow out of upper and under control limit values. In the event

of being used of S1 and S2 sinter at blast furnace as $LTD_{-0.5}$ value is high and because of consisting of iron oxide and impurities composition under the conditions such as reduction atmosphere at high heat and mechanic stretching start to be broken from their the weakest points.

In the course of reduction because of chemical reactions at low heat in the part of upper charge in the oven it becomes spoiled with consisting abrasion and pressure. As sinter's breaking index $LTD_{-0.5}$ is high the gas permeability decreases in the body of blast furnace. Passing of the gases from the bed correctly becomes difficult and dust amount in the chimney increases. So it is demanded that the value of sinter's breaking index $LTD_{-0.5}$ is low. Being low of the breaking index of S3 and S4 sinter are good for blast furnace working parameter. As seen at R diagram other test values except from S3 test are between upper and under control limit values.

Table 7.33 Tumbler index of X and R values for +6,35 mm

Samples	X	R
Sinter 1	68,60	0,00
Sinter 2	69,30	0,70
Sinter 3	64,76	4,54
Sinter 4	62,41	2,35
Sinter 5	67,24	4,83
Sinter 6	73,03	5,79
Sinter 7	70,72	2,31
$\bar{X}$	**68,01**	
$\bar{R}$		**2,93**
LCL	**65,01**	**0,00**
UCL	**71,01**	**7,55**

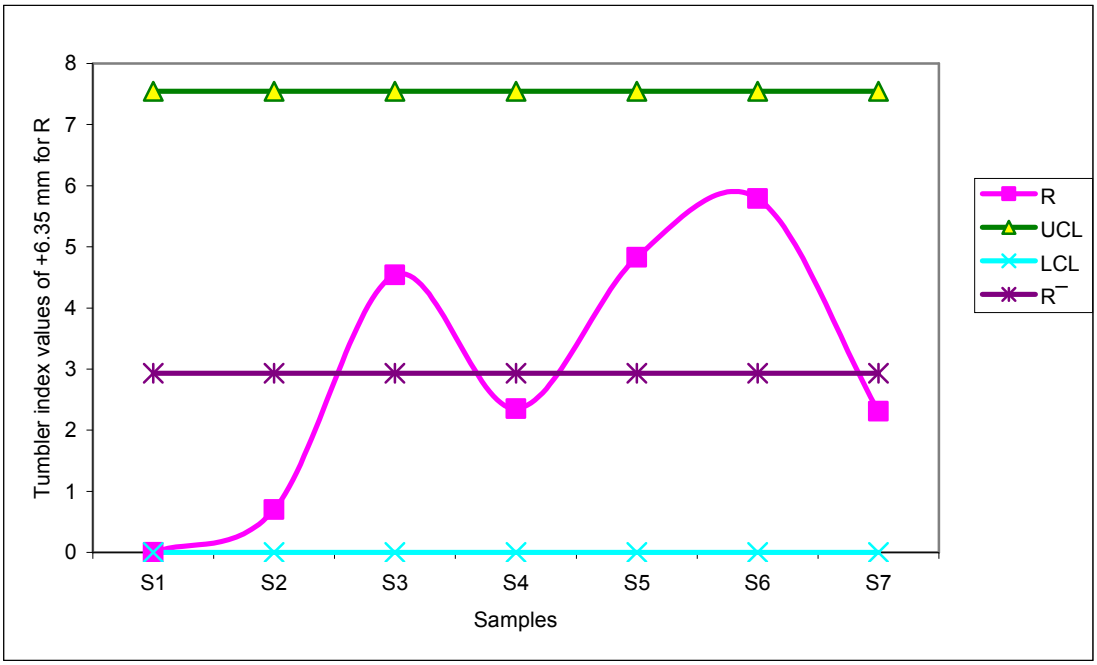


Figure 7.26 R diagram of Tumbler index values for +6.30 mm

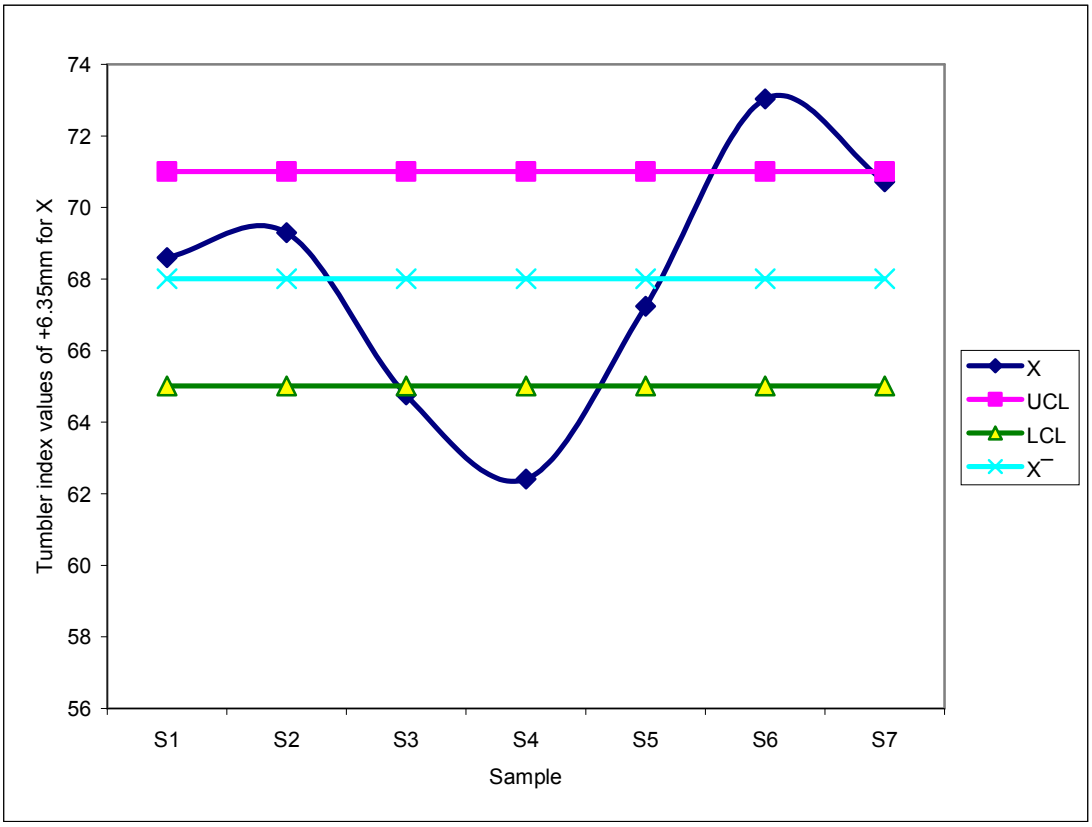


Figure 7.27 X diagram of Tumbler index values for +6.30 mm

As seen at figure 7.27 at X diagram S4 and S6 sinter test values of Tumbler index – 0.600 mm values overflow out of under control limit values. Abrasion and strength index of sinter must be high for used in blast furnace. R values between the limit values as shown in figure 7.26.

Table 7.34 Tumbler index of X and R values for -0,6 mm

Testler	X	R
Sinter 1	3,54	0,00
Sinter 2	4,02	0,48
Sinter 3	6,28	2,26
Sinter 4	3,98	2,30
Sinter 5	4,19	0,21
Sinter 6	3,90	0,29
Sinter 7	4,24	0,34
$\bar{X}$	**4,31**	
$\bar{R}$		**0,84**
LCL	**3,45**	**0,00**
UCL	**5,17**	**2,16**

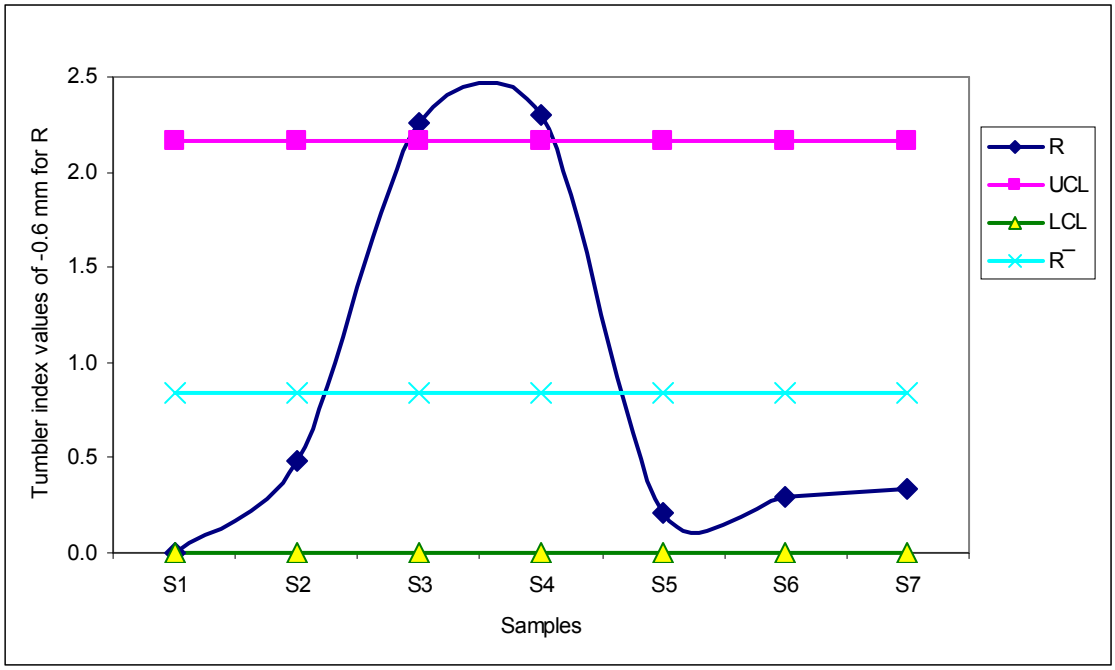


Figure 7.28 R diagram of tumbler index values for - 0.6 mm

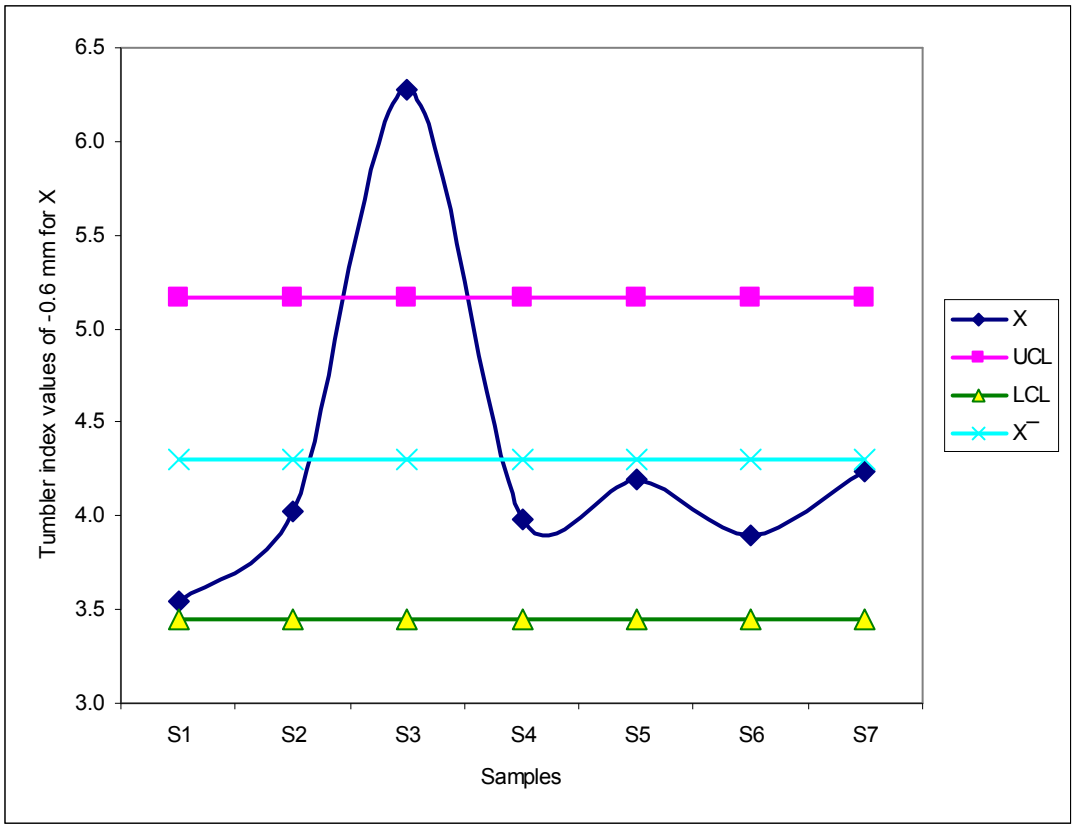


Figure 7.29 X diagram of tumbler index values for -0.6 mm

As seen at figure 7.29 at X diagram tumbler index – 0.6 mm values belong to S3 sinter test overflow out of upper control limit values. It isn't wanted that being moved of the iron iron ores or forming small particles with getting dust until being charged to the blast furnace. So substance that have tendency of abrasion and getting dust need to be screen again soon before loaded to the blast furnace. Both abrasion and getting dust indexes of sinters made from compact substances that have a lot of magnetite are more positive than sinters made from hematite, limonite, gothite etc. As seen at R diagram in figure 7.28 that S3 and S4 sinter test values is above of upper limit value.

7.5.10 Result

According to calculation made over; quality of production is economic if it is between UCL and LCL values. If the production made over UCL the quality is good but cost is so much. In this case production is wanted more than customer's desire quality as cost is much it increases that management's cost even though it is good in terms of quality. If it is made with LCL cost is cheap and the production isn't desired

quality. In this case the production can't be sold as it isn't desired quality and this is a danger for that management.

About the quality property of each production; showing change is normal. Changes consisting at properties about quality are separated from two parts as casual and specified changes; casual changes are changes that are found at nature of process. Generally this change that proportion is so small in total change is inescapable and is acceptable level. In the other side; specified changes originate from difference between substances that constitutes worker, machine and sinter blending is more important. The primary aims of quality control are factors that cause these changes, being detected and corrected. In the case of changes constitutes from only casual factors; process is under the control statistically in the case of constituting from specified factors; is out of control.

7.6 Studies on Sintering Behaviors with Mineralogical Analysis

7.6.1 The Reactions Occur of During Sintering and Phase Formation

In sinter blending, if the creator of ore ferrous mineral is hematite which suffered reduction and oxidation during the sintering. If the iron ore is composed of magnetite, during the sinter conversion process will be hematite. Figure 7.30 demonstrates the zone of sinter belt.

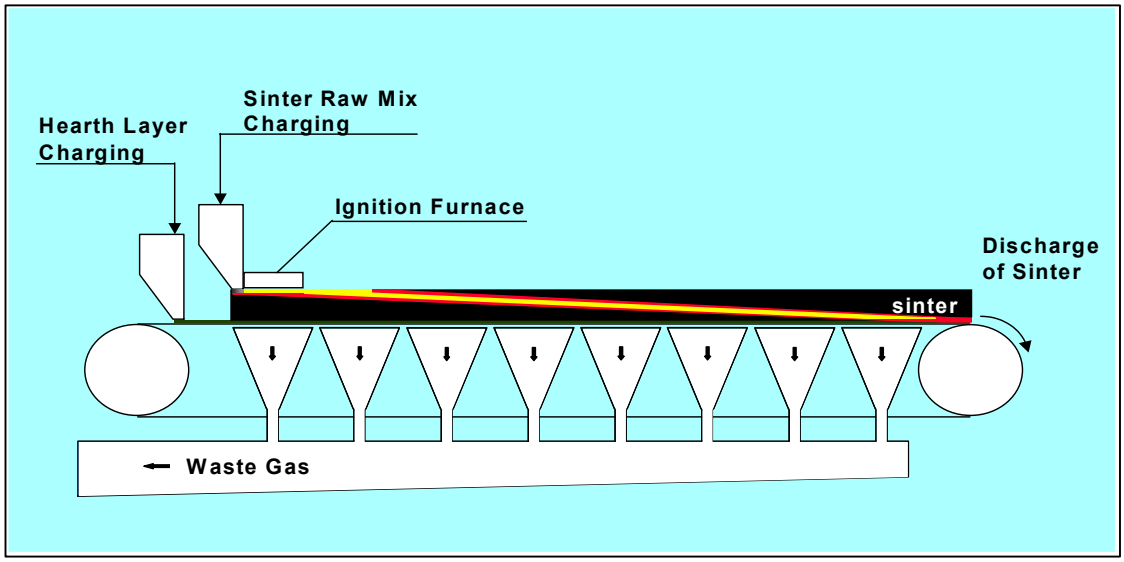


Figure 7.30 The zone of sinter belt (VAI, 2008).

According to this case, the main reaction only about in the sintering iron minerals;

$$Fe_3O_4 + 1 / 2\ O_2 = Fe_2O_3 \qquad (7.1)$$

Although this reaction is exothermic, it can not provide the heat which essential for the agglomeration. Therefore, to provide heat to the system, taking into account other characteristics of the sinter blend sufficient amounts of solid fuel (usually coke brezee) has been added. Sinter process in this case of magnetite to hematite oxidation processes with reaction;

$$C + O_2 = CO_2 \qquad (7.2)$$

In the sintering reactions are main reactions. Found in significant amounts in the

sinter blend of water, reaching temperatures above the vaporization temperature of steam in the region pass into the gas phase.

H_2O (s) = H_2O (g) (7.3)

The effect of the side of the heat consisting of burning coke, the smell of iron ore, the iron mineral can be found in together and / or entering further into SiO_2 sinter blends, $CaCO_3$, $MgCO_3$, FeS_2, minerals such as FeS and $A1_2O_3$ in the sintering process chemicals and / or phase changes have been taking place the product sinter. These can be summarized as the following reactions belong. Of these carbonate;

$CaCO_3 = CaO + CO_2$ (7.4)

$MgCO_3 = MgO + CO_2$ (7.5)

As will do calsine, the sulfur compounds found in the system,

$FeS_2 + 5 / 2O_2 = FeO + 2SO_2$ (7.6)

$FeS + 3 / 2O_2 = FeO + SO_2$ (7.7)

The oxidation of coke in the form of heat in the system provides additional heat, oxides such as SiO_2, Al_2O_3 which can be found in sinter blend don't change chemically on its own. These are just other materials in the system react with each other, and;

$SiO_2 + CaO = CaO.\ SiO_2$ (7.8)

$A12O_3 + SiO_2 + CaO = CaO.\ AI_2O_3.SiO_2$ (7.9)

$MgO + SiO_2 = MgO.SiO_2$ (7.10)

Compounds generated in the form of the system allow the slag formation. Those with the iron minerals,

$CaO + Fe_2O_3 = CaO.\ Fe_2O_3$ (7.11)

$MgO + Fe_2O_3 = MgO.Fe_2O_3$ (7.12)

can create slag which is the type of ferrite.

While sinter production of CO from burning coke is going down in the charge, it may cause forming the CO with Bouduard reaction by reacting with the C.

$CO_2 + C = 2CO$ (7.13)

In such a case,
It can be reduced to FeO and Fe levels by the following reactions;

$Fe_2O_3 + CO = Fe_3O_4 + CO_2$ (7.14)

$Fe_3O_4 + CO = 3FeO + CO_2$ (7.15)

$FeO + CO = Fe + CO$ (7.16)

If in this reactions, it can be reduced to the step of FeO and Fe. If a stable SiO_2 is available in the system.

$FeO + SiO_2 = Fe_2SiO_4$ (7.17)

Fayalite which is not wanted phase and can not be reduced in sinter may occur as in the reaction. Formed as a result of the reaction above to phases, where the product at the sinter structure that will affect the properties.

7.6.2 Preparation of Sintering Sample

SEM-EDS and optical microscopes and to be able to do in XRD analysis was prepared as shown in figure 7.31. Epoxy was mounting (Diameter is 30 mm and Height is 5 mm), SEM-EDS was inserted into the epoxy in order to analyze in optical microscope and in XRD device since sinter samples are very brittle. Blcak epoxy was used since it is harder and quick frozen.

Before put the samples of sinter into the epoxy, above and below of sample was grinded with grinder 600. The grinding was used carefully not to disrupt because of the sample is very brittle.

After placing the sample on the epoxy is added press machine was heated from room temperature up to 160 ^{o}C . Through this temperature, epoxy became fluid so it warped up the sinter completely and it was pressed so epoxy mold was occurred. Then, the pressed epoxy was cooled with the water channels around. All this process takes 12 min. Firstly the grinder 320, and then grinder 600 was used for experiment.

Figure 7.31 Sintered samples prepared for mineralogical analysis.

7.6.3 Mineralogical Structure

Chosen from within sintering blends, obtained from different ores imported epoxy powder was prepared at 7 sinter sample to the X-ray diffraction analysis and analyze the graph are shown in figure 7.32. Accordingly, the main mineral found in samples of wustite (FeO), magnetite (Fe+2Fe_2+3O4), mayenite ($Ca1_2Al_{11}O_{33}$), manganese oxide (Mn_3O_4), wollastonite (2M-$CaSiO_3$) to hematite (Fe_2O_3), calcium silicate (Ca_2SiO_4) and S2 , S4 and S5 allocated in the different samples as potassium sulfate ($K_2S_2O_4$) and sodium phosphate (Na_3PO_4) was encountered. Alumina manganese oxide ($MnAlO_4$) was also encountered in the sample 41.

Figure 7.32 Hardening of sintered samples with epoxy resin.

7.6.4 X-Ray Diffraction Analysis

Sinter samples obtained from different mineral sinter blend were taken to assign minerals. X-ray diffraction analysis of sinter sample results and analysis graphs is given as in the figure 7.33. The taken samples are analyzed in the department of D.E.U Metallurgical and Materials Eng. Laboratories with Rigaku D/Max-2200 model X-ray difraktometre. Scanning process was applied between 3º - 90º 2 min / degree of speed. Results were evaluated with Jade 7 software of device.

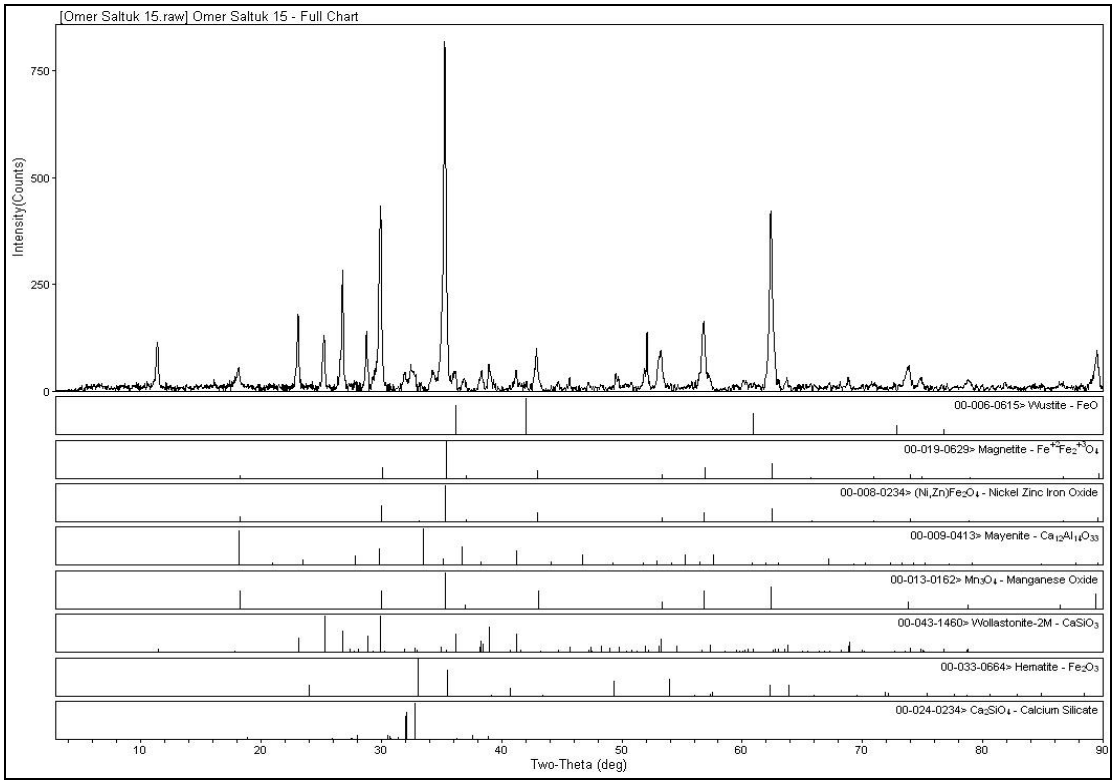


Figure 7.33 X-Ray diffraction analysis of sinter sample results and analysis graphs.

7.6.5 Scanning Electron Microscope (SEM) and EDS Analysis Method

Sinter samples obtained from different mineral sinter blend were taken to assign minerals in order to make the determination of the distribution of micro analysis and elemental analyses were subjected to scanning electron microscopy and EDS. SEM image and EDS analysis of sintered samples is given in figure 7.34. In the analysis, D.E.Ü Metallurgy and Materials Eng. Brand in the laboratories of the Department Jeol JSM-6060 model scanning electron microscope is used. The device is capable of magnification of 8x-300.000x. Magnifications of sinter samples were made 30X, 1.000x, 5.000x and 10.000X separately. 1.000x magnification SEM image analysis on the elements in phases over the images were obtained with the EDS analysis.

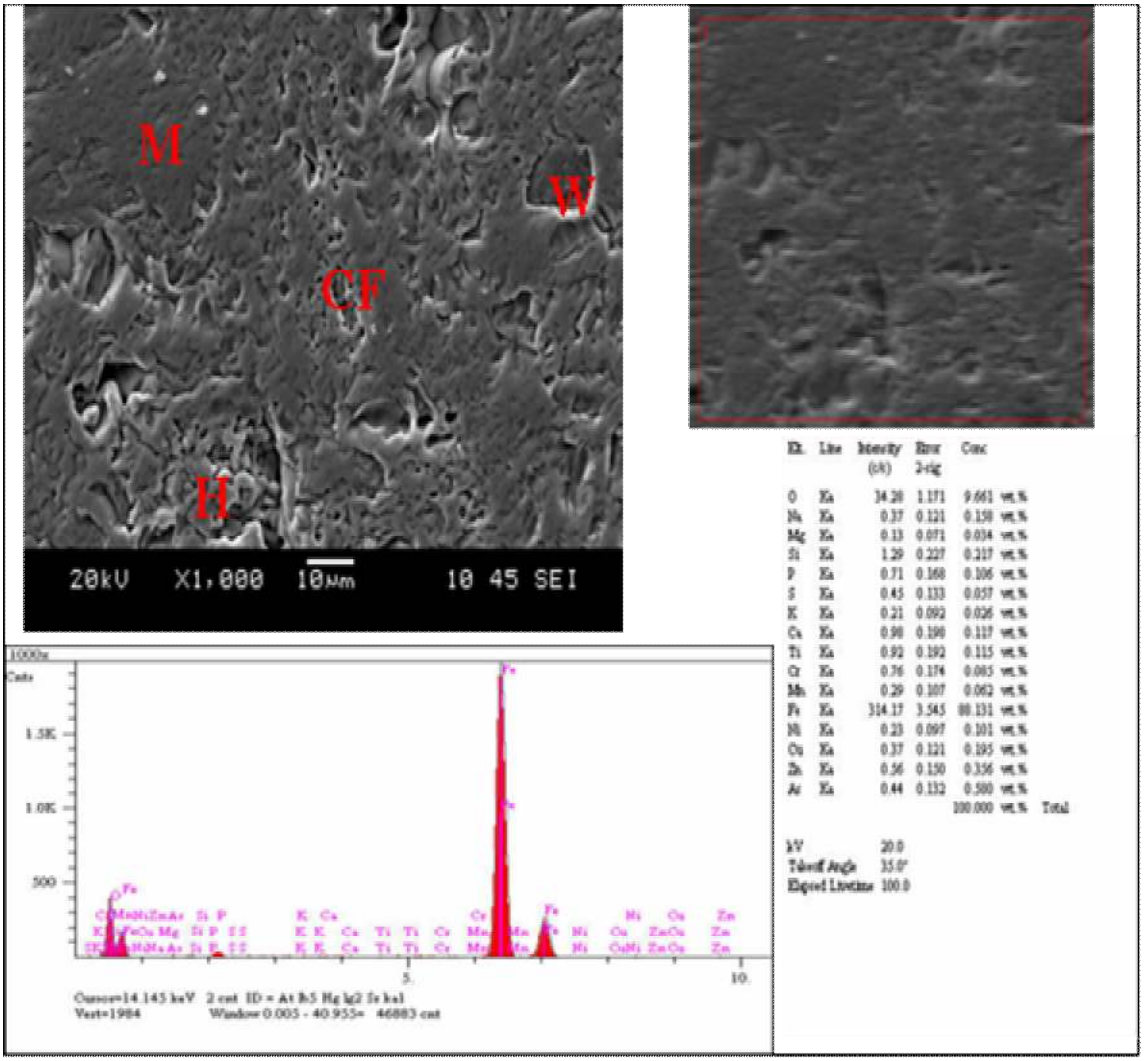


Figure 7.34 SEM image and EDS analysis of sintered samples.

7.6.6 Experimental Methods

Raw materials required blending prior to the sintering operation, initially; iron ore is crushed to a suitable size is put into the blending by a stacker. Some flux material, coke and return dust can also be added at dosage unit. In addition, collected dust and mill scale, flue dust and sludge from gas cleaning in steel making may be added to the ore blend at the mixing stage. At the start of the sintering operation the ore blend is transferred from the beds to storage bunkers at the start of the sinter plant. The ore blend and the coke breeze are weighed on conveyer belts and loaded into a mixing drum (as seen in figure 7.35). Here, they are blended completely and the mixture is moistening to enhance the lumpy, which improve the permeability of the sinter bed.

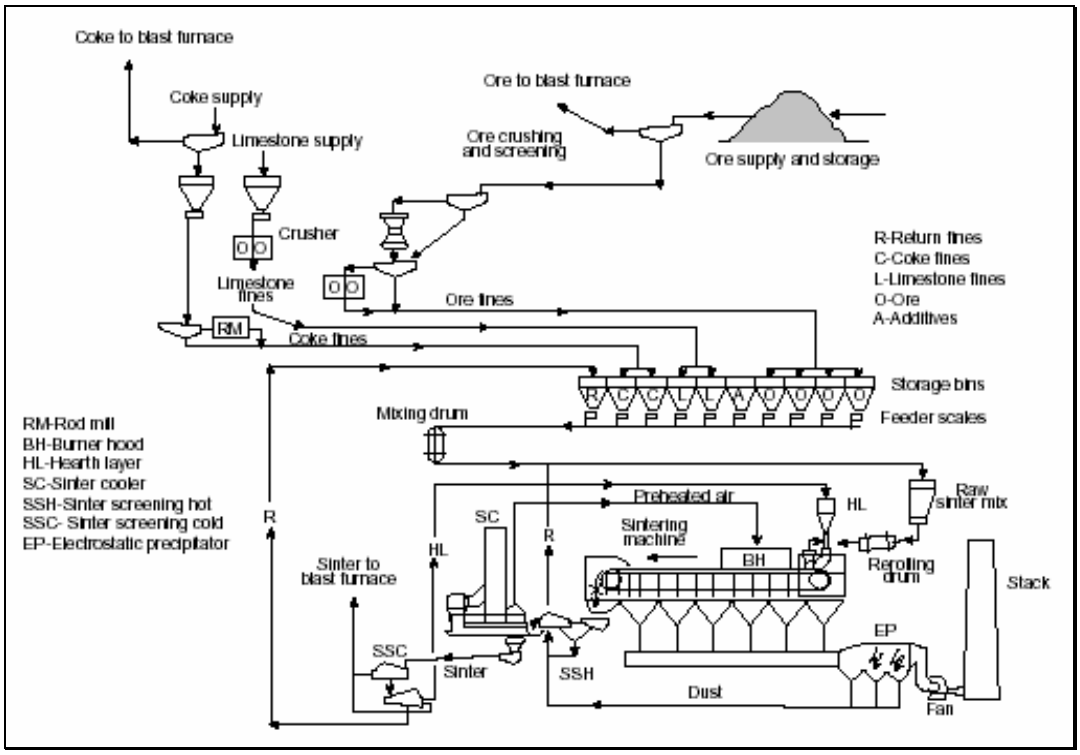


Figure 7.35. Iron ore sintering process flow diagram

Iron bearing raw materials: Iron ore (A), iron ore (B), iron ore (C) were supplied from Turkey, whereas iron ore (D), iron ore (E) and iron ore (F) were foreign iron ore fines that used in sinter test program. These iron ores were uniformly mixed respectively to reduce fluctuation of chemical composition and obtain representative sample. Chemical compositions and main iron mineral ores are given in table 7.35.

Table 7.35. Chemical and mineral composition of main minerals for iron ores

Raw materials	**Chemical Composition (%)**							**Mineral composition (%)**
	Fe	*$Si0_2$*	*$Al_2 0_3$*	*CaO*	*MgO*	*Zn*	*Ni*	
A	54.69	6.36	0.86	1.06	0.90	0.090	0.030	*Hematite, Geothite*
B	49.27	10.17	3.44	1.26	0.68	0.040	0.054	*Hematite, Limonite*
C	55.44	9.96	2.88	0,91	1.80	0.028	0.140	*Hematite, Magnetite*
D	65.16	3.65	1.37	0.56	0.09	0.017	0.013	*Hematite*
E	65.31	3.55	1.08	0.02	0.03	0.007	0.003	*Hematite*
F	63.22	3.28	2.05	0.01	0.02	0.006	0.002	*Hematite, Limonite*

Fixed proportioning ratio among iron ore fines and relevant parameters applied in test program are listed as follows;

1) Foreign iron ores (D, E and F): domestic iron ores (A, B and C) = 50: 50 (as seen in table 7.36)
2) A: B: C: = 20:15:15 (based on weight %)

3) BF flue dust and iron bearing sludge account for 2.0% and 1.5% of raw mix respectively. According to the test program, sintering test was carried out under condition of basicity (CaO/SiO_2) 1.8. Moisture in raw mix is fixed at 7.0%. coke breeze consumption is fixed at 6%.

Table 7.36. The sintering properties of blending ores.

Sample No	Blending of iron ores (mass %)				Productivity		RDI (%)	Machine Productivity (t/d m^2)
	Total domestic iron ore (A, B, C)	D	E	F	Moisture (Mass %)	Coke Breeze (kg/t sinter)		
S 1	50	50	-	-	6.4	58.54	28.40	30.78
S 2	50	-	50	-	6.1	59.21	26.60	28.39
S 3	50	-	-	50	7.1	60.48	25.20	27.64
S 4	50	33.3	33.3	33.3	5.4	57.62	30.20	32.08
S 5	100	-	-	-	7.3	61.35	24.90	25.96

Table 7.36 shows the sintering properties varied with iron ore type. The productivity of the sinter ranged from 25.96 to 32.08 t/d m^2; the RDI ranged from 24.90 to 30.20 %; the coke rate ranged from 57.62 to 61.35 kg/ton sinter, the proper moisture of raw mix, 5.4 to 7.3 mass %. The sinters made from the low alumina commercial iron ores (S1, S2, S4) presented the lower coke rate (57–59 kg/t sinter) and required the lower moisture (5.4–6.4 mass %). However, the RDI was not simply related to the alumina level of iron ores.

After dry mixing of the blend, sufficient amount of water was added, and wet mixing was done to obtain adequate ball. Then, sinter mix was placed in the sinter strand in which all sintering experiments were done during with this study. The baking of the sinter mix was done in a plant scale sinter. The baked sinter, called the sinter cake was then subjected to tumbling and abrasion test in sequence. Test results

were explained depending on sinter qualities such as sinter strength, return fine balance, sinter mineralogy. Tumbler index test was made to final product using 1/5 ISO standard method. Tumbler drum is of 1000 mm in diameter and 100 mm in width with rotation speed of 25 rpm. One batch of 3 kg sinter with grain size of 10-50 mm was put into tumbler drum rotating at 200 rpm and screen analysis was applied. The weight percentage of +6.35 mm fraction was taken as tumbler index as the weight percentage of -0.6 mm was taken as abrasion index (seen in figure 7.36). The final size was minus 100 microns and 100 gr of the sample was taken for chemical analysis. Table 7.37 shows chemical analysis of sinter samples.

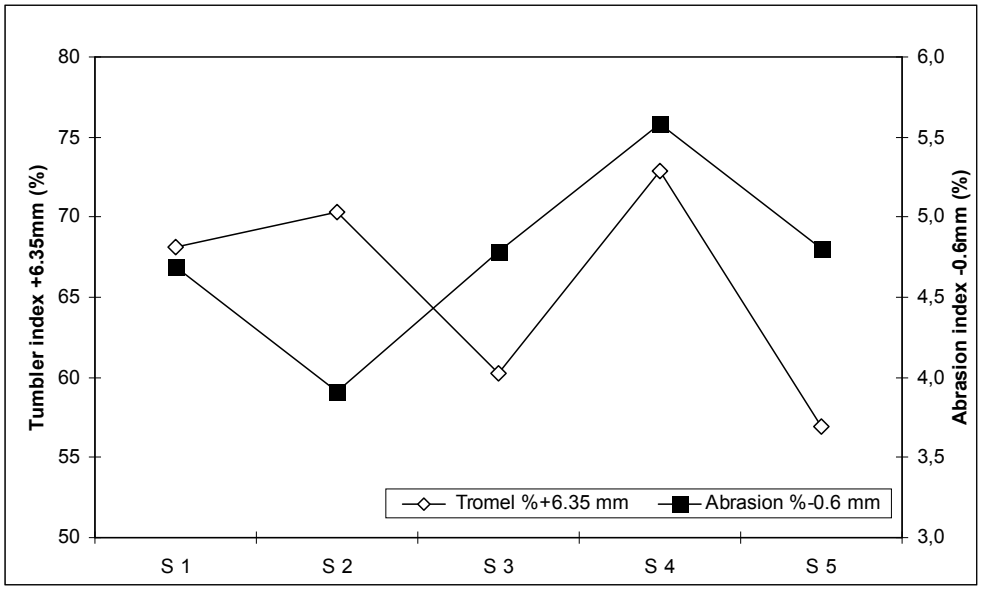


Figure 7.36. Changes of Tromel and Abrasion index depend on iron ore types.

Table 7.37 Chemical analysis of sinter samples.

Sample	Fe	SiO_2	CaO	MgO	Alkali $K_2O.Na_2O$	Al_2O_3	Ni	Zn	MnO
S 1	53.68	6.95	12.51	2.22	0.12	1.58	0.04	0.03	0.68
S 2	52.97	7.06	12.34	2.04	0.12	1.61	0.00	0.04	0.95
S 3	55.78	5.61	9.55	1.93	0.08	1.83	0.02	0.02	0.77
S 4	57.82	4.93	9.31	1.29	0.05	1.64	0.00	0.01	0.56
S 5	52.06	10.05	7.39	2.01	0.28	1.78	0.06	0.02	0.97

7.6.7 Macro and Micro Structural Analysis of the Sinter Samples

7.6.7.1 Macrostructure Analyses:

The sintered samples were collected from sintering plant of Isdemir. Various defects existing on the surface and inside of the sintered ore made easy to form crack propagates under the action of external forces and ambient media, which would finally lead to the breaking and cracking of the sintered ore. The distribution of crack formation in various sintered ore structures are shown in figure 7.37. The aim of the present work is to provide some reference rules for the macrostructure designing of the sintered ore so as to meet the requirement of BF production. Results of examinations, it is observed that calcium ferrite is a major factor influencing crack resistance of sintered ore. The finer the grain size of calcium ferrite, the greater is the crack resistance of sintered ore.

The crack lengths in various sintered ore structures were measured by optic microscope. The results of the distribution of crack lengths in various sintered ore structures are shown in figure 7.38. It is believed that the strength, reducibility and size distribution of sinter particles and the yield from a sinter strand are determined by the inherent strength of the bonding phases present and the structure of the pores (Loo & Leung, 2003; Kawaguchi & Usui, 2005).

Furthermore, different pore sizes seem to have different effects on sinter strength, reducibility and other properties; the high temperature reducing property is controlled by the micro pore sinter structure, while the sinter strength is determined by the macro pore sinter structure (Loo & Leung, 2003). It is therefore important to optimize the pore structure in sinter to improve its reducibility, while maintaining the cold strength, reduction degradation and load softening properties.

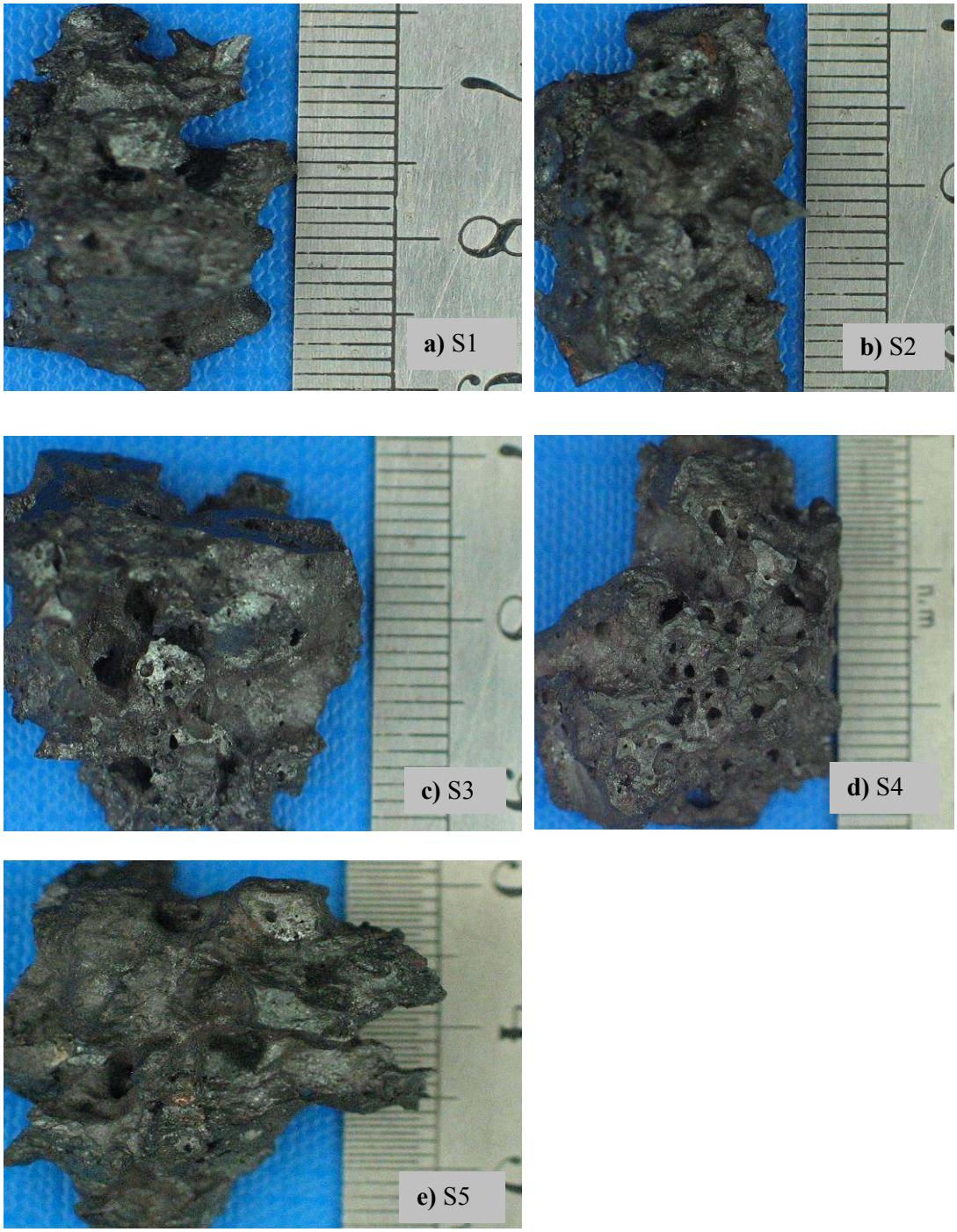


Figure 7.37 The crack formation of sintered granules with different iron ore. a)S1, b)S2, c)S3, d)S4, e) S5.

On the other hand, a number of other researchers suggested that the cracks resulting from a volumetric change accompanying the phase transformation of crystalline hematite to magnetite are mainly responsible for the reduction degradation of sinter (Sugawara, Shimizu & Kawazu, 1971)

Recent results suggested that it is possible to further improve the permeability of a sinter blend, and therefore control the pore size distribution of the sinter product, by adjusting the particle sizes of both limestone and coke breeze simultaneously

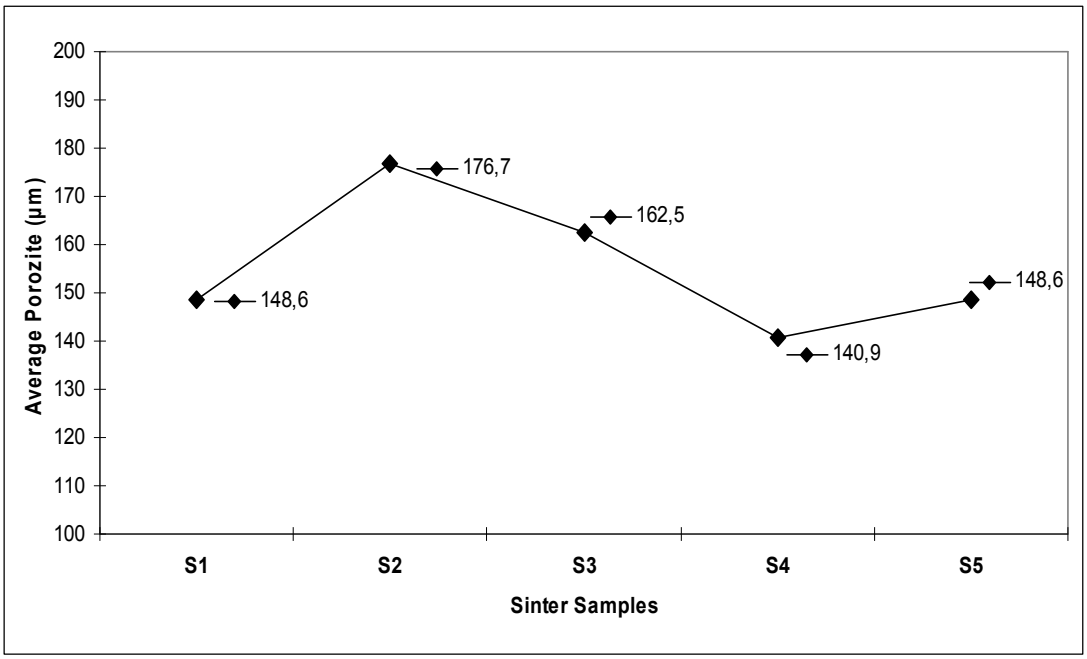


Figure 7.38 Crack distribution in various sintered ore structures.

7.6.7.2 Microstructure Analysis

After sintering, the specimens were mounted in epoxy resin and then vacuum impregnated. Generally, the specimens were polished to expose a plane section corresponding to 1 mm below and parallel to the original top surface. When a large ore particle was present, the specimens were polished to expose a plane section perpendicular to the top surface on a diameter of the specimen so that a cross section of the ore particle could be seen. These sections were polished by using silicon carbide paper to 1000 grit employing ethanol as a lubricant, and finally to ¼ micron by using diamond paste. During the whole process, the specimens were not exposed to water.

The mineralogical content and microcosmic texture of sinter were analyzed using optical microscope. The microcosmic textures of sinter are demonstrated on the micrographics a, b, c, d and e in figure 39.

Figure 7.39-a) shows typical aspects of sinter structure for a iron ore (F) ore mix, at 1250 ^{o}C sinter consists of fine texture of needle like calcium ferrites and hematite, porosity is irregular and disrupted; at 1320 ^{o}C hematite and magnetite crystals precipitate from slag; calcium ferrites are less abundant but are forming plates. There are a few intergrowths of fine magnetite crystal grains and hematite.

In the case of a iron ore (E) mix (seen in figure 7.39-b) the same general trends are observed. In addition, typical figures are obtained at low temperature: some kinds of quasi particles are formed; large amounts of needle like calcium ferrites are present at their periphery. The glass phases between hematite together with hematite form eutectic texture. Along edges of some hematite crystal grains, there is oxidation occurrence.

In the matter of a iron ore (D) mix (seen in figure 7.39-c) consists mainly of molten texture formed by magnetite and calcium ferrite or second birth hematite that are liquid phase filled by calcium ferrite. The crystal grains of some fine grain magnetite and hematite form interlacing texture, and the crystal grains of some magnetite are filled with calcium silicate. Few gangues are visible. There are a few intergrowths of fine magnetite crystal grains and hematite.

In the event of sinter made a foreign ore mix (seen in figure 7.39-d) consists mainly of molten texture formed by magnetite and calcium ferrite. A few needle shaped or branch shaped calcium ferrite and magnetite form interlacing textures, which occur along caverns holes. Magnetite amount is increased and also intergrowth of magnetite and hematite can be observed. Oxidation occurs at the edges or along caverns holes. At some places in the glass phase, separation out of magnetite fine crystal grains is observed.

In the case of sinter, made from domestic ore mix to contain high gangue ores (seen in figure 7.39-e), the same general trends apply. However, the reactions involving slag are promoted, due to the amount of gangue. The amount of calcium ferrites is rather high. At high temperatures, large plates are formed. Their chemistry is quite complex, with considerable substitutions, involving silica, alumina, lime an even magnesium oxide.

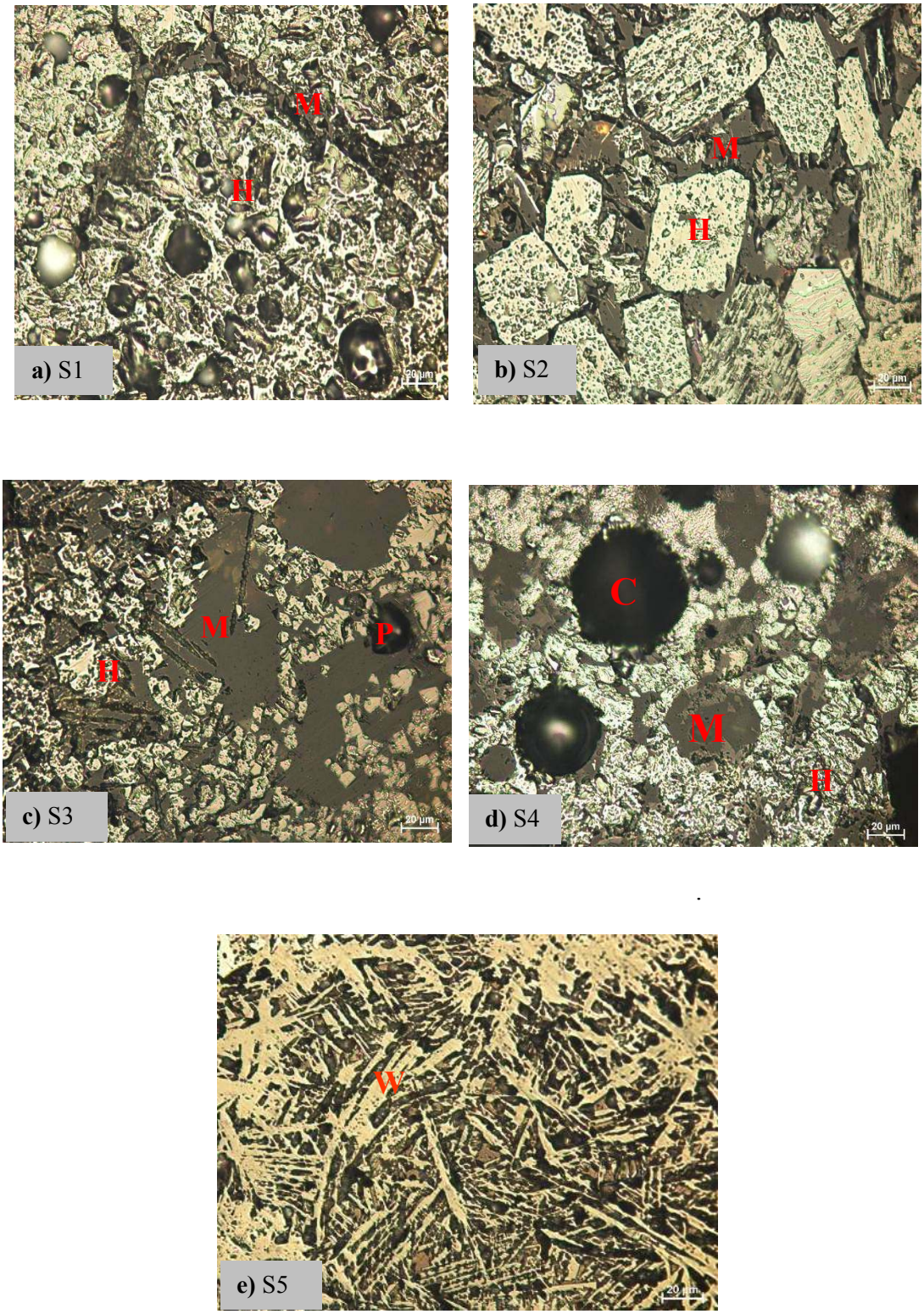


Figure 7.39 Microstructures of sinter samples a) S1, b) S2, c) S3, d) S4, e) S5 (H:Hematite, M:Magnetite, P: Pore, W: Widmanstaaten (x500).

Figure 7.40 shows comparison of SEM images of sinter surfaces. It was clearly shown that each ore had individual surface morphology. In particular, S2 sample consists mainly of molten texture formed by magnetite and calcium ferrite (as seen in figure 7.40-a). There are a few interlacing existences of crystal grains of magnetite and hematite and a few coarse grain porous hematite (as seen in figure 7.40-b). Thus, high capacity of water absorption of S2 was caused by readily intrusion of water thorough open pores from the ore surface. Granulation in high moisture content is necessary to enhance granulation ability.

Calcium ferrite melt was found to form within residual ores of sinter S4 (as seen in figure 7.40-b). There is intergrowth of magnetite and hematite. Oxidation is seen at edges or along caverns and holes. At some places between coarse grain hematite, there is glass phase and much separation out of magnetite fine crystal grains (as seen in figure 7.40-b).

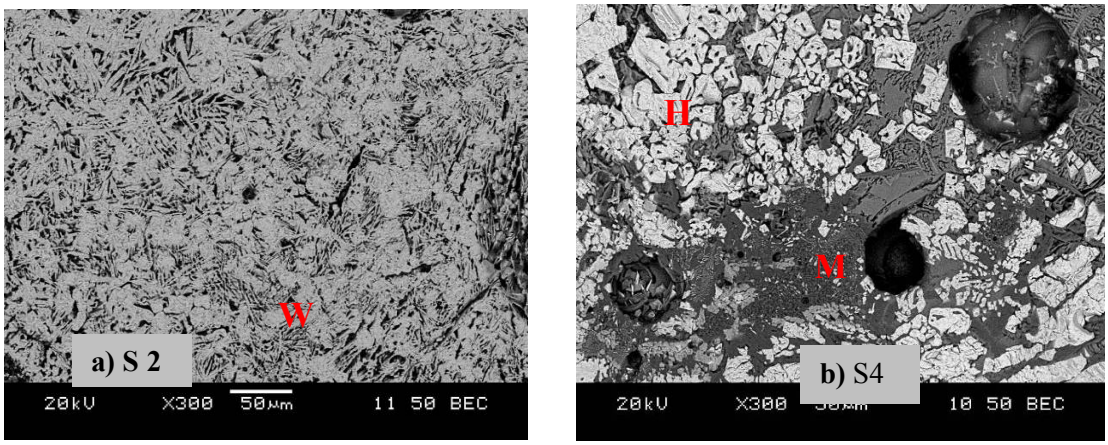


Figure 7.40 a) SEM images of sinter surfaces S2, b) S4 (H:Hematite, M:Magnetite, W:Widmanstaaten (X300).

7.6.7.3 X-Ray Diffraction Studies

Phase determinations of sinter samples were done with the help of its X-ray diffraction pattern given in figure 7.41 and with mineralogical investigation under an optical microscope. Sinter samples were mainly composed of FeO (wustite), Fe_3O_4 (magnetite), $CaSiO_3$ (wollastonite), and various $CaO\text{-}FeO\text{-}SiO_2$ solid solutions like $CaFeSiO_4$ (ferro-monticellite). Also Fe_3O_4 (magnetite), Fe_2O_3 (hematite), Mn_3O_4 (manganese oxide), Ca_2SiO_4 (dicalcium silicate), $K_2S_2O_4$ (potassium sulfate), Na_3PO_4

(sodium phosphate) and amorphous glassy phase were observed. The sinters were very heterogeneous, due to the limited diffusion possible during the short period at melt conditions and variations at the peak temperature reached. Secondary hematite known as skeletal rhombohedra hematite is the major cause of poor reduction degradation resistance of sinter (Hino & Kumano, 2003). This is based on frequent observations of cracks around the narrow neck regions of such hematite.

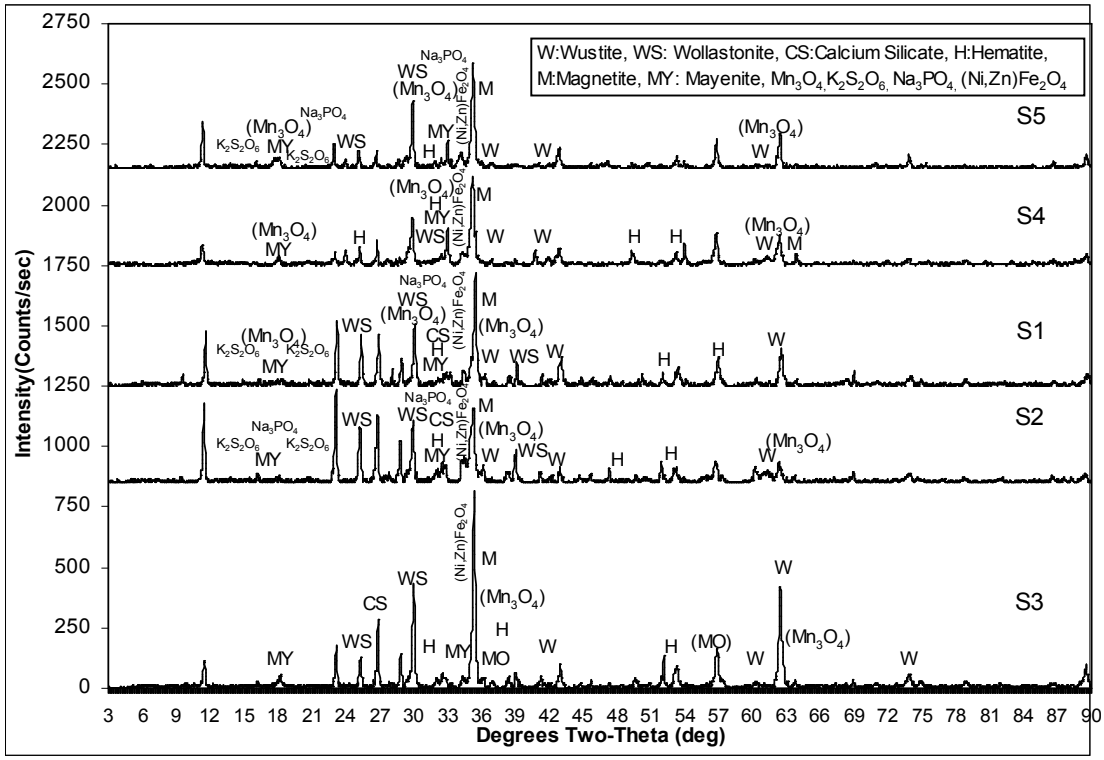


Figure 7.41 X-Ray diffraction studies of sinter samples.

7.6.7.4 Mineralogical Analysis of Sinter

The mineralogical content and microcosmic texture of sinter is analyzed using optical microscope. The results of test for microstructure properties of sinter are shown in table 7.38 where No.l sample represents sinter with a basicity of 1.0, No.2 sample sinter with a basicity of 1.7, No.3 sample sinter with a basicity of 2.1, No.4 sample sinter with a basicity of 1.0 and an dunite fines addition of 3.64%, No.5 sample sinter with a basicity of 1.7 and an dunite fines addition of 3.64%, and No.6 sample sinter with a basicity of 2.1 and an dunite fines addition of 3.64%.

Table 7.38 Mineralogical composition of sinter (%).

SI. No.	Hematite	Magnetite	Calcium ferrite	Calcium silicate	Vitreum	Gangue
1	16.75	43.25	17.3	10.8	9.3	1.3
2	14.38	44.63	20.0	12.5	6.3	0.5
3	11.75	45.13	25.0	12.2	5.3	0.3
4	16.20	44.50	15.7	12.8	8.2	1.3
5	15.80	46.50	16.0	11.7	7.2	1.0
6	15.30	47.70	18.0	11.3	6.2	0.3

The microstructure characteristics of sinter are as follows:

1) No. l sample consists mainly of molten texture formed by magnetite and calcium ferrite. The crystal grains of some fine magnetite and hematite form netlike interlacing texture. There is coarse grain porous hematite. At some borders of magnetite there is oxidation occurrence. Some magnetite is filled by liquid phase. Fine acicular calcium ferrite and calcium silicate separate out from the liquid phase. For details, please refer figure 7.42.

2) No. 2 sample consists mainly of molten texture formed by magnetite and calcium ferrite or of second birth hematite that are liquid phase filled by calcium ferrite. The crystal grains of some fine grain magnetite and hematite form netlike interlacing texture, and the crystal grains of some magnetite are filled with calcium silicate. Few gangues are visible. There are a few intergrowths of fine magnetite crystal grains and hematite. For details, please refer to photos in figure 7.43.

3) No. 3 sample consists mainly of molten texture formed by magnetite and calcium ferrite. Most calcium ferrite is of branched shape or tree branch shape. There are a few interlacing existences of crystal grains of magnetite, hematite and a few coarse grain porous hematites. Along edges of some magnetite crystal grains, there is oxidation occurrence. Few eutectic textures and gangues are visible. For details, please refer to photos in figure 7.44.

4) No. 4 sample consists mainly of molten texture formed by magnetite and calcium ferrite. A few of needle shape or branch shape calcium ferrite and magnetite form interlacing textures, which occur along caverns and holes. There is much porous magnetite. At some places, there is intergrowth of magnetite and hematite.

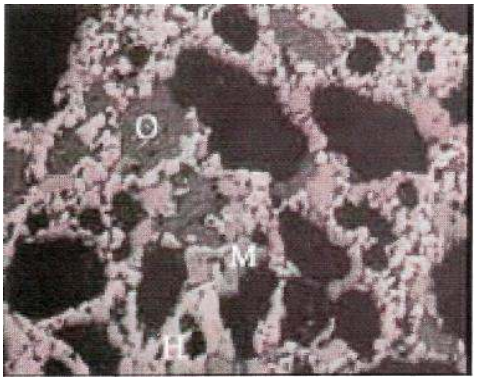
Oxidation is seen at edges or along caverns and holes. At some places between coarse grain hematite, there is growth of light blue color magnetite-hematite. For details, please refer to photos in figure 7.45.

5) No. 5 sample consists mainly of magnetite filled with calcium silicate and of magnetite granular texture. In the glass phase, there is much separation out of magnetite fine crystal grains. At some places, there is coarse-grain crude hematite and a few of intergrowth of fine granular magnetite and hematite. The gangues are seldom seen. For details, please refer to photos in figure 7.46.

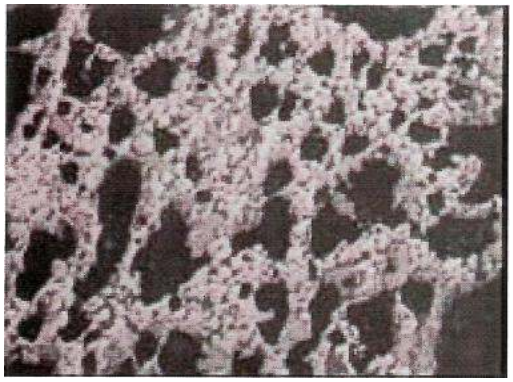
6) No. 6 sample consists mainly of molten texture formed by magnetite and calcium ferrite. The glass phases between magnetite together with magnetite form eutectic texture. In some glass phases there is separation out of calcium ferrite and calcium silicate. At some places, there are a few of granular textures and intergrowth of magnetite and hematite crystal grains. For details, please refer to photos in figure 7.47.

From table 7.38 and analysis on microscopic structure, under the condition that fuel quantity is fixed, basicity of product sinter has significant effect on mineralogical composition of product sinter. With increase of basicity, the calcium ferrite with relatively high strength in binding material is increased, thus tumbler index is obviously improved. For example, the No.3 test sample with basicity of 2.1, with basicity is increased from 1.0 to 1.7 and 2.1, calcium ferrite content in product sinter is increased from 17.3% to 20.0% and 25.0%, likewise, Tumbler index is improved from 63.3% to 67.7% and 72.0% (see table 7.16).

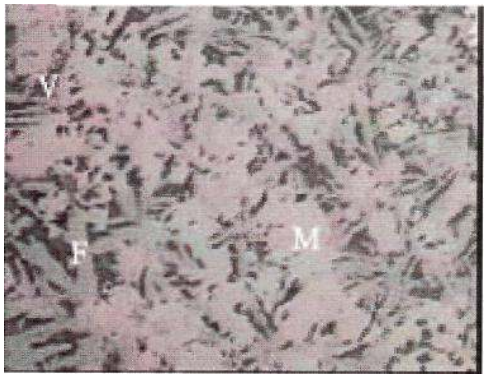
Legend:
M-Magnetite
H-Hematite
L-Bicalcium silicate
O-Eutectic crystal texture
V- Glass phase
P-Gangue
F- Calcium ferrite



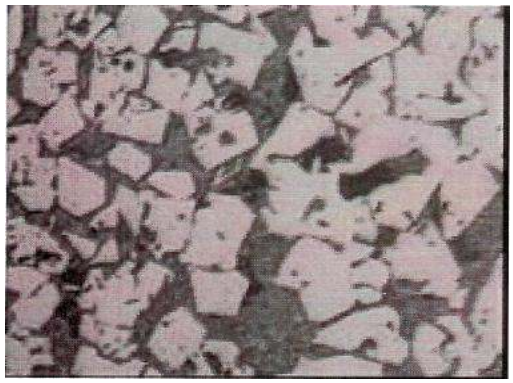
Net-shaped texture of fine magnetite-hematite crystal grains fixed by dunite



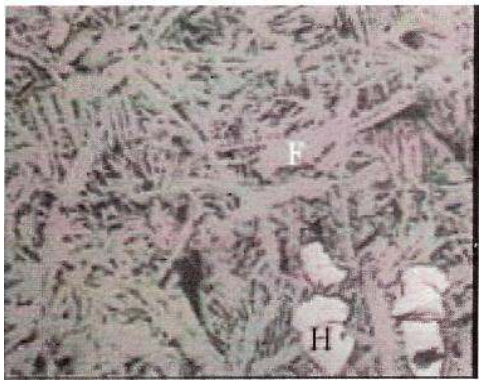
Net-shape texture formed by interlaced fine magnetite-hematite grain



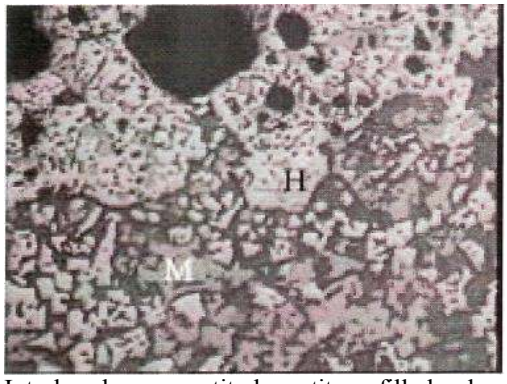
Molten texture formed by magnetite and calcium ferrite



Magnetite filled by bicalcium silicate and calcium ferrite

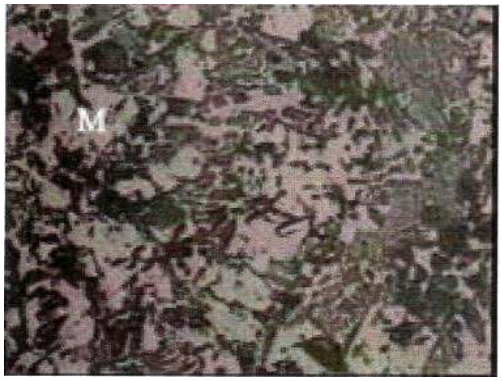


Bulk particles of hematite consolidated by interwoven texture

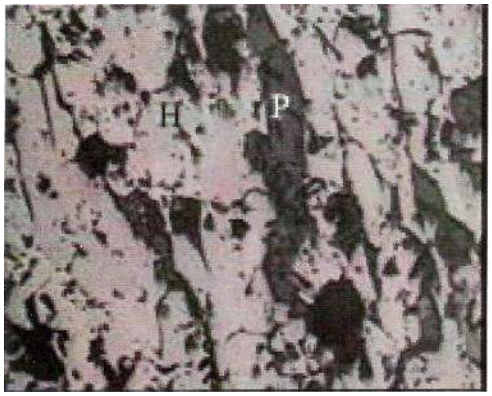


Interlaced magnetite-hematite filled by bicalcium silicate

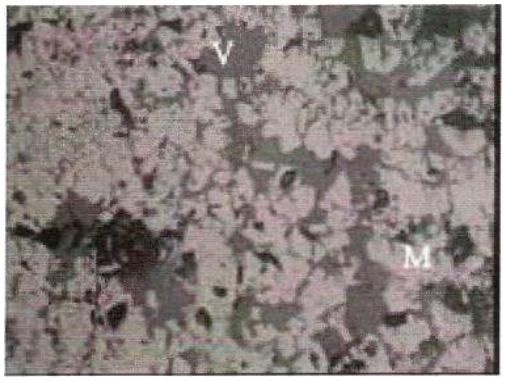
Figure 7.42 Microscope images of sintered samples (X 300).



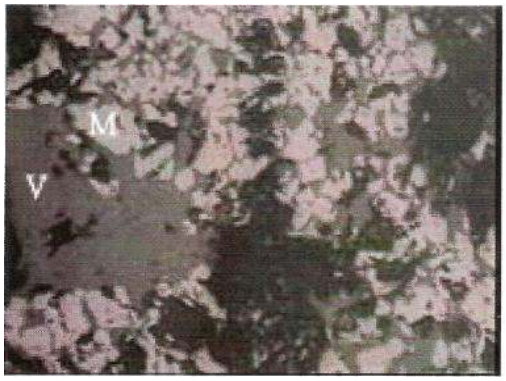
Magnetite and bicalcium silicate



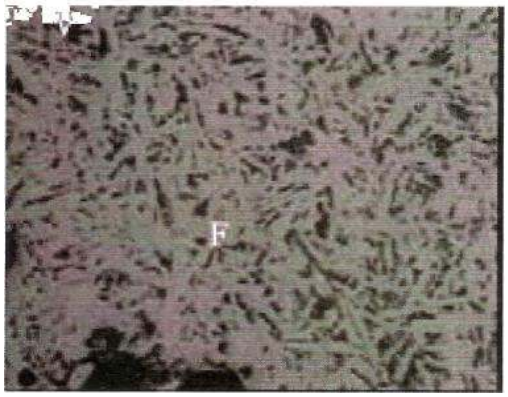
Hematite and gangue



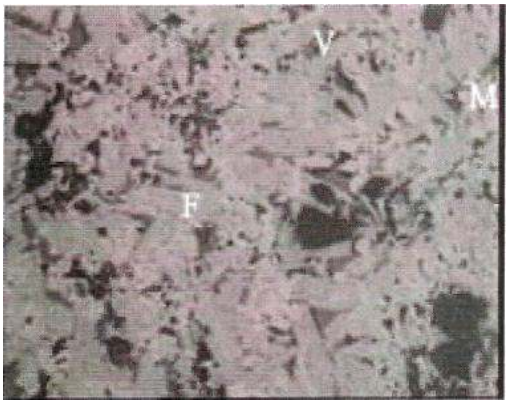
Grain shape texture



Magnetite with glass-phase and melilite

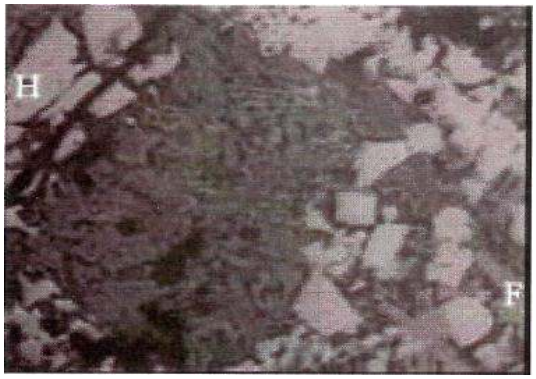


Interwoven molten texture of calcium ferrite

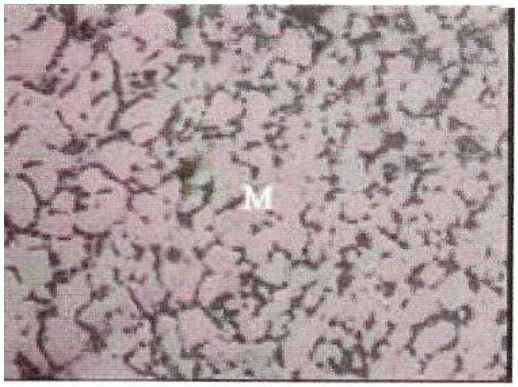


Molten texture of magnetite and calcium ferrite

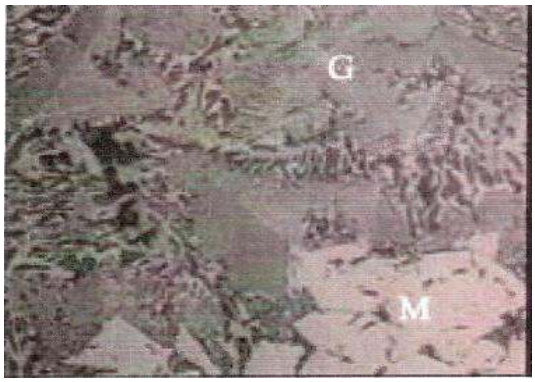
Figure 7.43 Microscope images of sintered samples (X 300).



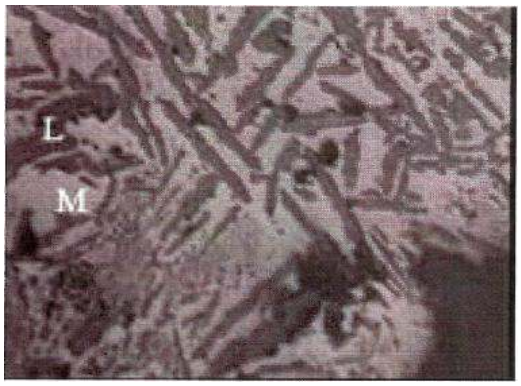
Eutectic crystal texture of calcium ferrite with bicalcium silicate



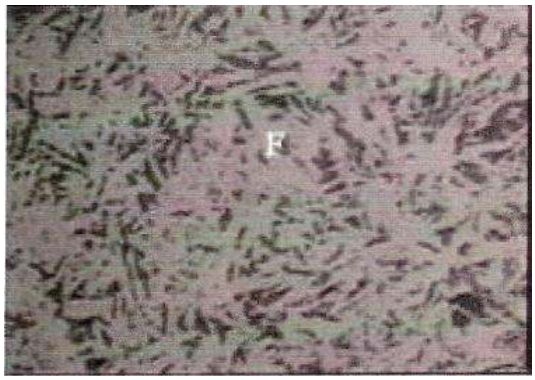
Bicalcium silicate



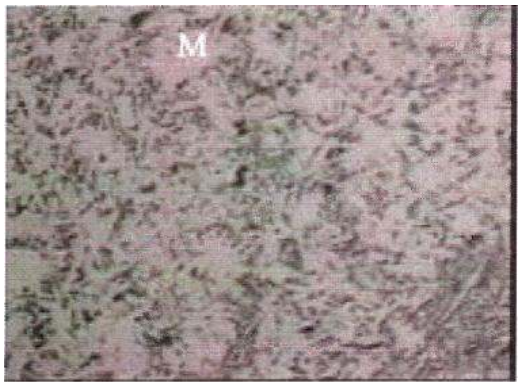
Eutectic crystal texture in liquid slag



Coarse needle shape bicalcium silicate

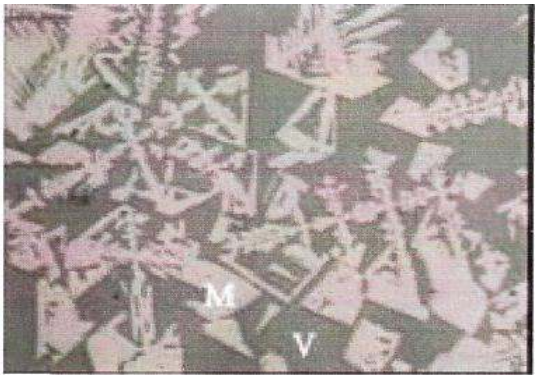


Interwoven texture of calcium ferrite



Interwoven molten calcium ferrite

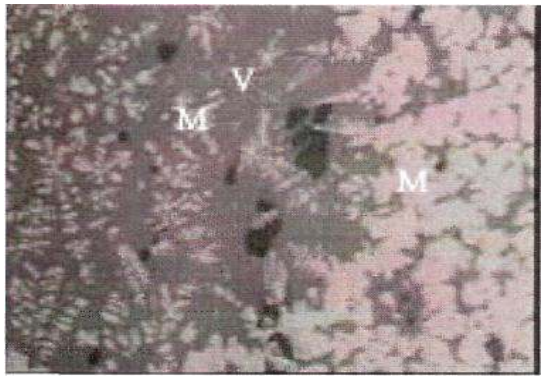
Figure 7.44 Microscope images of sintered samples (X 300).



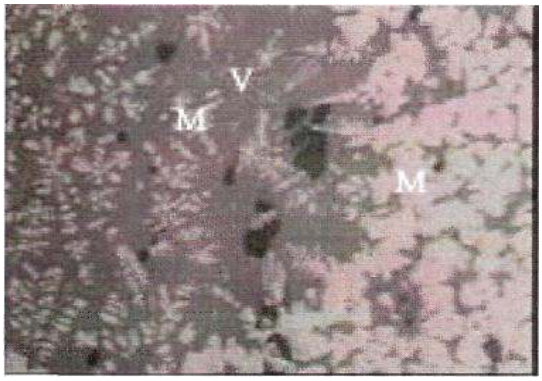
Fish bone shape magnetite separated out from glass phase

Dunite & magnetite early crystals separated out from glass phases between magnetite

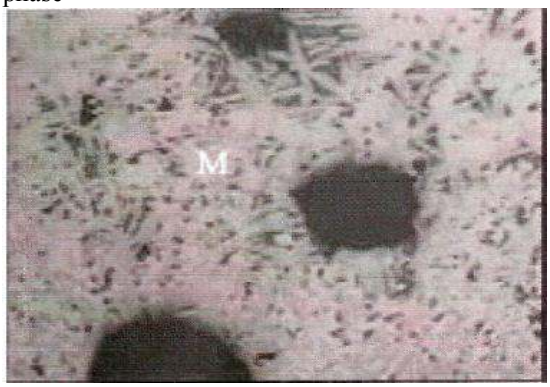
Early magnetic crystal separated from glass phase



Magnetic and hematite crystal separated from glass phase

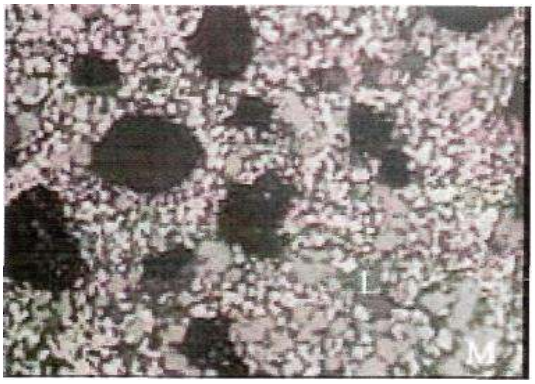


Magnetic and hematite crystal separated from glass phase

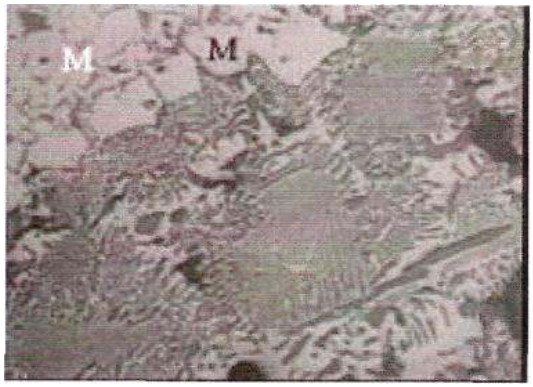


Interwoven molten texture

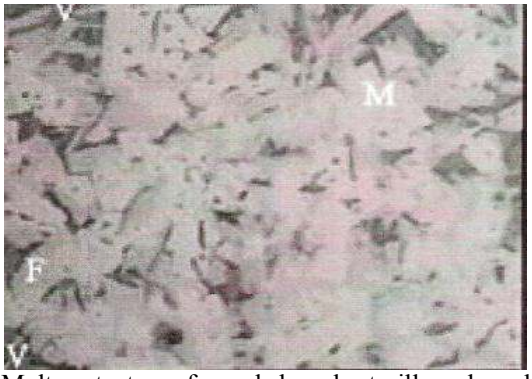
Figure 7.45 Microscope images of sintered samples (X 300).



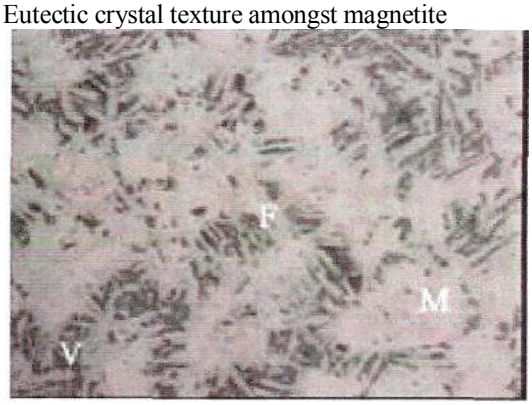
Interlacing texture enlarged 25 times



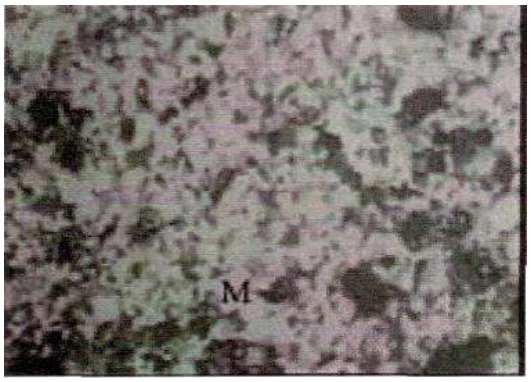
Eutectic crystal texture amongst magnetite



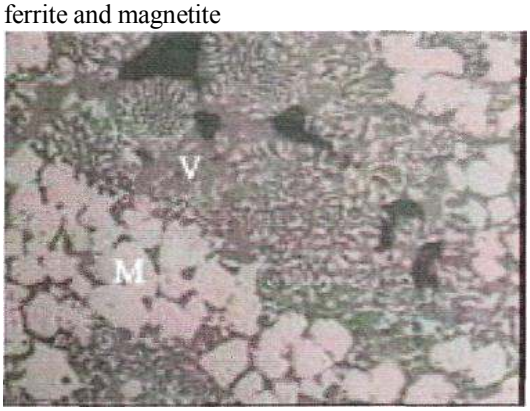
Molten texture formed by short-pillar shaped calcium ferrite and magnetite



Molten texture formed by needle shaped calcium ferrite and magnetite

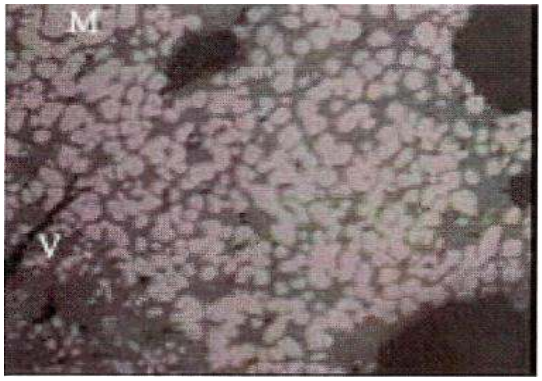


Interlacing texture enlarged by 25 times

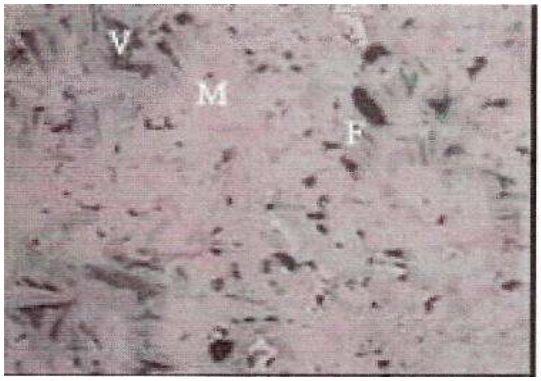


Early magnetic crystal separated out from glass phase

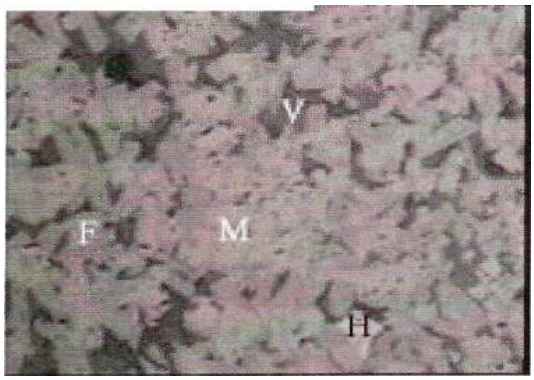
Figure 7.46 Microscope images of sintered samples (X 300).



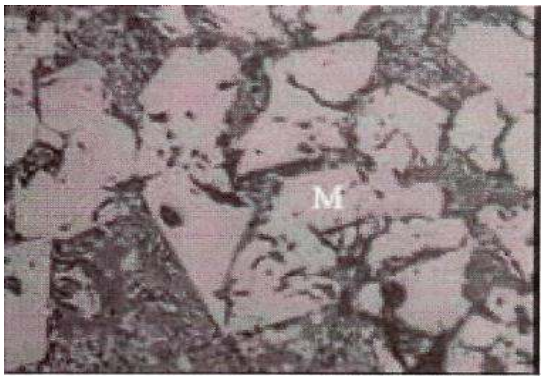
Magnetite and Wiistite (Fe O) consolidated by dunite



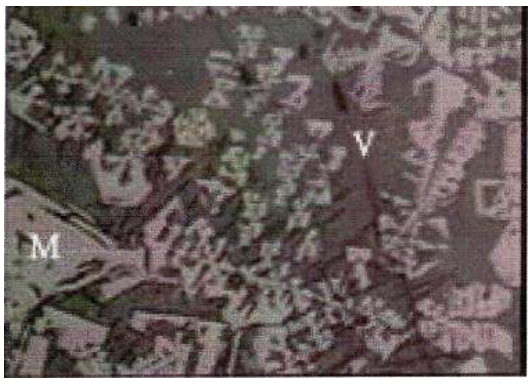
Molten texture



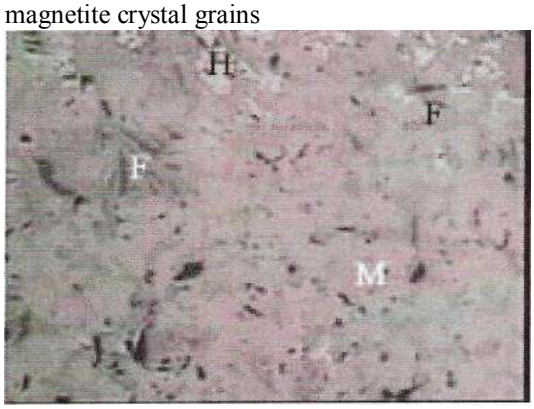
Molten texture formed by plate-shaped calcium ferrite and magnetite



Bicalicum silicate or magnetite early crystal separated out from glass phases between magnetite crystal grains



Fishbone shaped magnetite separated out from glass phase



Molten texture

Figure 7.47 Microscope images of sintered samples (X 300).

7.6.8 Mineral Formation During Sintering

The mineral formation processes shown in figure 7.48 is characterized by the reaction between calcium ferrite melt and gangue minerals containing SiO_2. On the basis of the Fe_2O_3-CaO-SiO_2 ternary phase diagram observed mineralogical changes are discussed in the following under the assumption that any other constituents such as $A1_2O_3$ and MgO have no effects on the changes (Phillips & Muan, 1959). This assumption is considered to be reasonable because their contents are small as shown in table 7.35.

At 1 400 °C the mixture of Fe_2O_3 % 90 and 10% CaO consists of magnetite and the melt at composition "ℓ" in figure. 7.49. In the following we consider that SiO_2 is added little by little into this system at 1 400 °C till the mean composition reach "N" (81% Fe_20_3, 9% CaO, 10% SiO_2) in figure. 7.49.

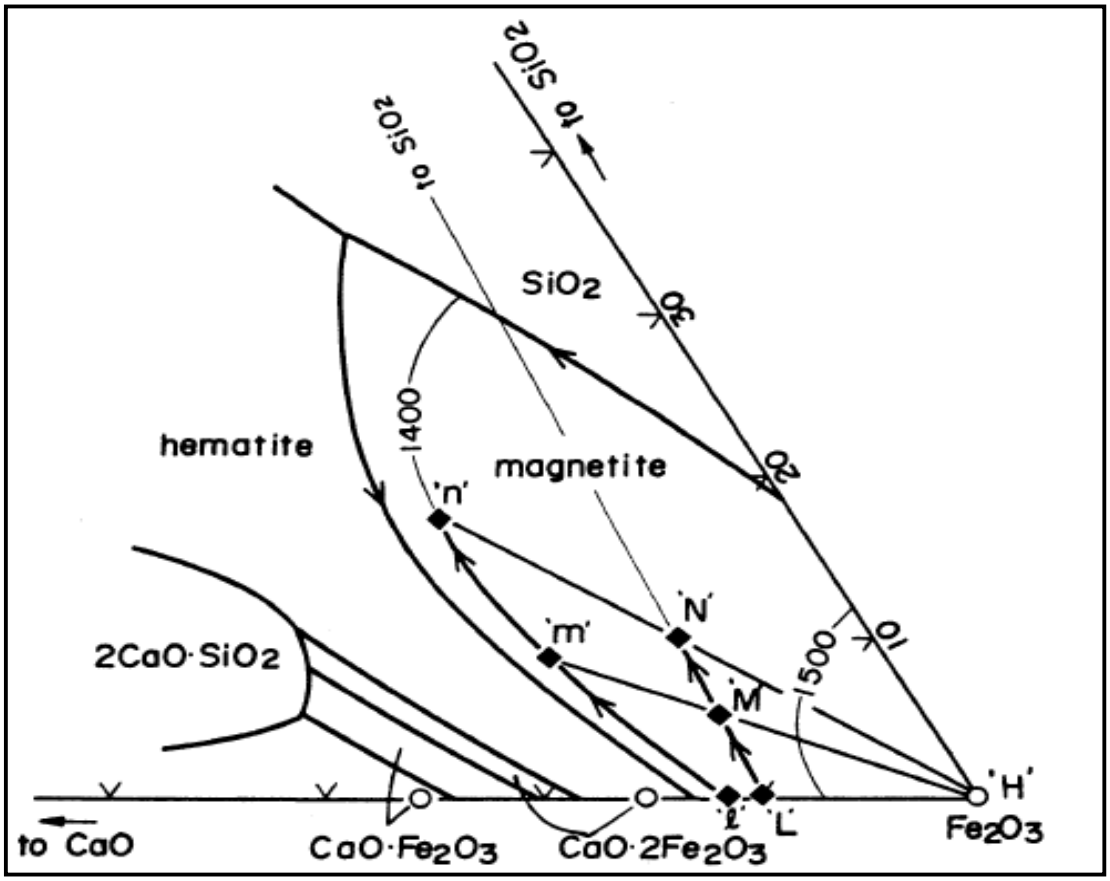


Figure 7.49 The assimilation path of quartz into the 90%Fe_2O_3 (Phillips & Muan,1959).

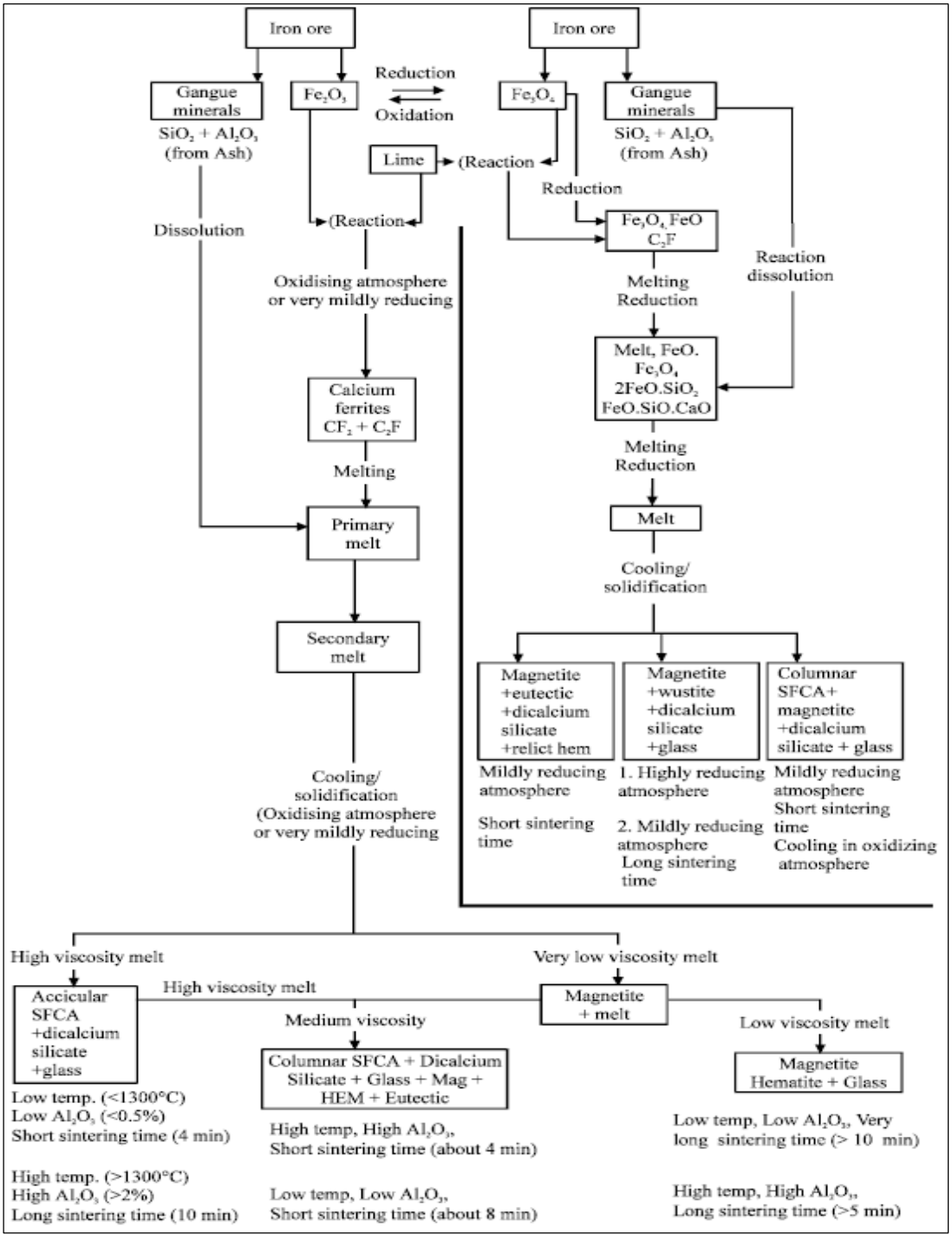


Figure 7.48 Schematic illustration of sintering mechanicm (Phillips & Muan,1959).

With the dissolution of SiO_2 into the melt, the composition of the melt would change along the liquid us surface of 1400° C or curve ℓ -*m*-*n* in figure. 7.49. At the same time, the mean composition of the system would change along the straight line L-M-N. The lever rule shows that the quantity of magnetite varies from the initial

value /L//H to the final value t nN/nH. This means that magnetite increases in quantity with the dissolution of SiO_2 into the calcium ferrite melt. If the dissolution takes place in the hematite phase, hematite will precipitate and its quantity will increase in the same way as mentioned above. In this way the mechanism for precipitation of iron oxides by reaction between calcium ferrite melt and gangue minerals containing SiO_2 can be understood on the basis of the phase diagram. The reason why this mechanism operates during heating is, after all, attributed to the fact that the formation of calcium ferrite occurs most easily (Okazaki, Hida & Sasaki, 1986).

Figure 7.50 shows that for a porous hematite ore (foreign iron ore D) and iron ore A,C (as seen in table 7.35) blends sinrtered in air at the maximum sintering temperature 1270^{o}C, SFCA increases when the porous hematite ore level increases from 40 to 60 %. So the porous hematite ore is beneficial for SFCA formation when it is blended with magnetite concentrates. There may be two types of melt formed bolow 1300 ^{o}C in air in the system $CaO\text{-}Fe_2O_3\text{-}SiO_2$, i.e., calcium ferritc silicate at low local basicity and calcium ferrites at high local basicity (Okazaki, Hida & Sasaki, 1986).

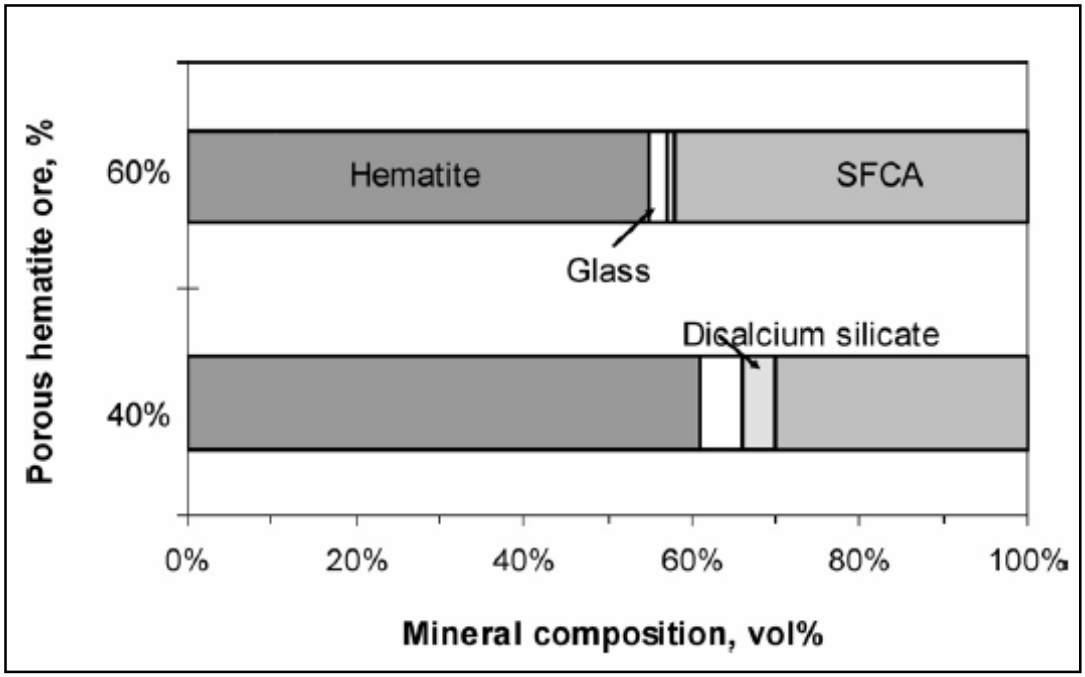


Figure 7.50 Mineral compositions for "Porous hematite ore in the blends sintering in air (Yang, 2004).

Figure 7.51 shows that porous hematite (foreign iron ore E) and hematite/goethite (Ore domestic iron ore A,C, e.g.) ores have igher ability to form SFCA, which is in line

with the assimilation studies that indicate that porous hematite and hematite/goethite ores are easier to react in sinter.

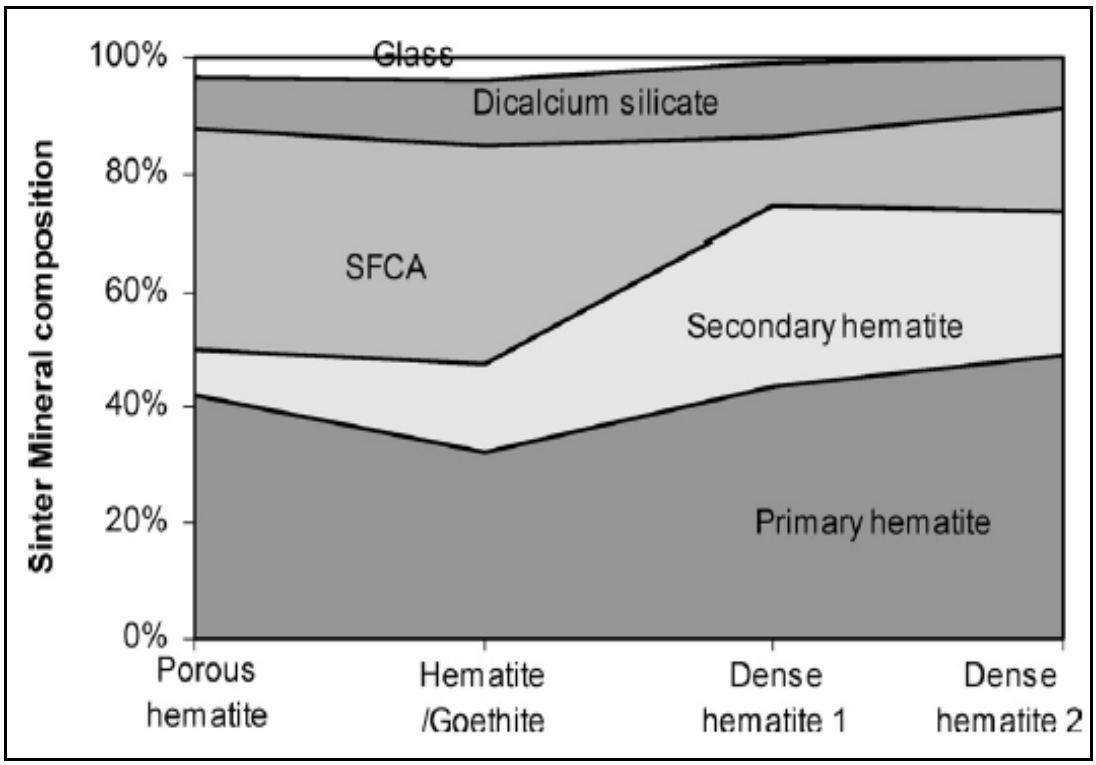


Figure 7.51 Mineral compositions for sinters of the same composition from different iron ores (Yang, 2004).

The studies showed that a small increase in the alumina content of sinter blends can have a significant adverse impact on the strength and reduction degradation characteristics of final sinter leading to deterioration in gas permeability in the upper part of the blast furnace (Hsieh, 1993; Fujimori, 1998). Finally, it increases the alumina loading into the blast furnace and therefore the furnace slag volume, leading to increased coke consumption. It could also alter the composition and properties of the primary melt formed in sinter located near the cohesive zone of the blast furnace (Matsuno, 1979). The trends of total hematite, reoxidized hematite, calcium ferrite and tumble strength which might affect the RDI were identical for all iron ores, but the trend of porosity varied with iron ore type (Matsuno &. Harada, 1981).

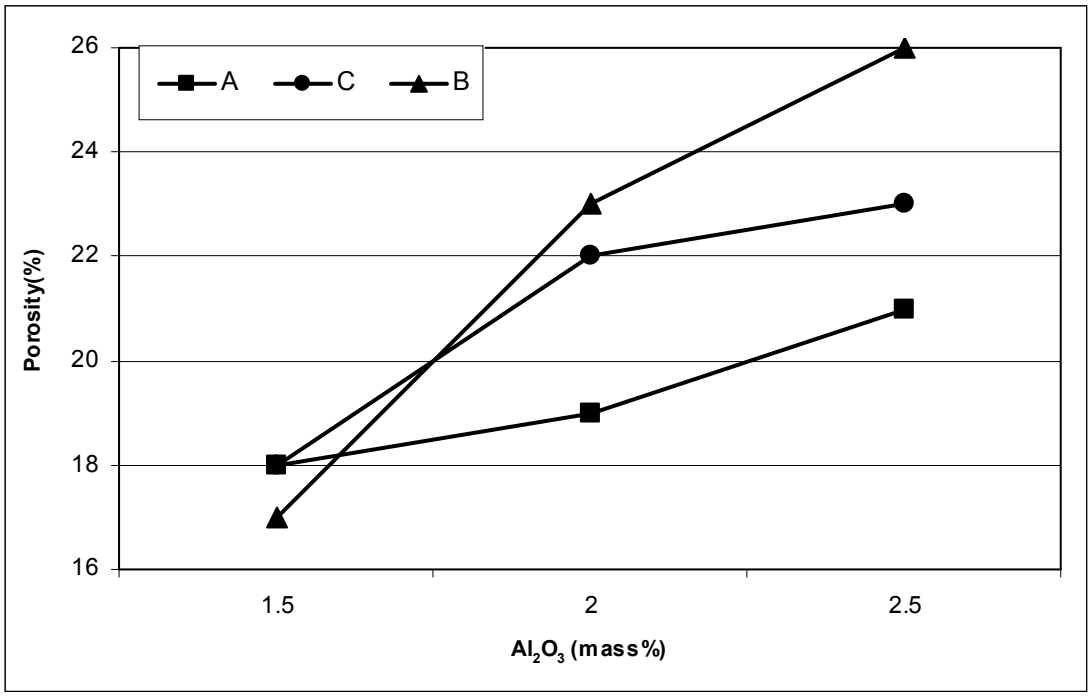


Figure 7.52 Effect of alumina content on the porosity of sinter made by individual iron ore.

Previous experimental work carried out in air showed that the amount of calcium ferrite increased with basicity and $A1_2O_3$ content. The same trends were found in this work. It was also recognized that MgO could retard the oxidation of magnetite to hematite in sinter a finding which is also confirmed in this work. In this study, iron ore B sintered with the lowest coke rate showed the largest increase in porosity; iron ore C sintered was less marked. For iron ore A sintered with a high coke rate, the porosity was lowest and the porosity was reduced slightly as the alumina content increased (seen in figure 7.52).

CHAPTER EIGHT
CONCLUCIONS

The government and the other private plants product of iron ores that is 6.2 million tones per a year in Turkey. Ermaden produces 3.000.000 tones/year of iron ores and the private sector produce 3.200.000 tones/year of iron ore. The total iron ore needed integrated steel work from Turkey that is 13.500.000 tones/year.But these steel works will need extra 7.300.000 tones/year iron ore, it is obtained from import. (Erünsal, 2005).

The world's raw iron production is increase from 28 million tones (beginning of 20 century) to 780 million tones (endings of this century) and it reaches the 1.414 million tones at in 2010 (List of countries by steel production, 2011). This was an increase of 15% compared to 2009, a new record for global crude steel production. Turkey last year, an increase of 15.2 percent, crude steel production of 29.1 million tonnes carried. (Steel Production, 2010).

Iron and steel industry plant from Turkey report to their scheme that include the increase of hot metal product from 5 - 5.5 million tone/year to 16-17 millions tones/year in the coming years. They wll need the 26-28 million tones/year iron ore to get the production of hot metal manufacturing. 25% of this need is only provided in domestic mine (Erünsal, 2005).

Today, in modern blast furnace ironmaking, the sinter plant no longer can be seen as a separate production unit, but must be fully integrated with the blast furnace to generate the ideal burden for optimized production and cost efficiency. Modern blast furnace operation at high levels of productivity and high coal injection rates is only possible using raw materials with consistent and uniform properties. As the main component used in the blast furnace burden, the production of high quality sinter is decisive for assuring high and stable blast furnace productivity with a simultaneously low consumption of reductants (Pammer, Stiasny, Wurm & Gould, 2002).

The iron ore steel industry that worked by blast furnace used sinter as a base feed. The sinter depends on some physical properties such as hot gas permeability at blast furnace, has low dust rate, low broken down index. Iron steel industry wants the quality chemical composition in point of including alkali, sulphur, nickel and iron grade. But the domestic materials are insufficient for chemical quality. So that the domestic ore and import ore that have acceptable ratio of quality are mixed and charged to blast furnace by sintering. Sinter charging to blast furnace is increase as a result of the foreign iron ore, lump iron ore and pellet costs are decreased.

ISDEMIR Sinter Plant is a facility that is running simultaneously and certain quality of sinter can be produced. Iron dust, coke breeze, dunite, and mill scale blend almost certain amounts involved. The amount of flue dust and sinter dust returned must be a little bit changed. The blending height uses of between 38 and 40 cm in ISDEMIR. Sinter production values of the impurity in the production are required to be done:

%Fe:	55 min
%FeO:	8 max.
%$Al2O_3$:	2 max.
%K_2O+Na_2O:	0,20 max.
%Ni:	0,02 max.
%Zn:	0.02 max.

This study is composed of production and usage information by the sinter technology in Isdemir. In addition, the results of various experiments and analysis data were prepared with production knowledge of the sinter technology in Isdemir. Except that sinter blend is consisted of inevitable material, ferrous materials as flue dust, scale and flux constructive materials as limestone, dunite and BOF slag in different proportions performed with additional in sinter blend. The main subjects of study in this thesis are given below:

a. Sinter applications at ISDEMIR.

b. R & D Laboratory test work with MECC corporation.

c. Statistic quality control of sinter samples.

d. Studies on improving sinter quality with mineralogical analysis.

A-) Sinter applications at ISDEMIR:

Example of sinter with obtained of different domestic and foreign iron ore used. These are the effect of sinter production efficiency, mineralogical and mechanical properties of sinter product, and then reducibility effects, and finally the sintering furnace is used in the production of liquid crude iron and the effects of productivity investigated (table 7.9). The sinter experiments and analysis were made with 40 different sinter productions. Results are given in figure from 7.1 to 7.7.

Example of sinter obtained with used different foreign iron ore in blend were uniformly mixed respectively to reduce fluctuation of chemical composition and obtain representative sample. Fixed proportioning ratio among iron ore fines and relevant parameters applied in test program are listed as follows foreign iron ores and domestic iron ores; 50 %: 50 % as given in table 7.10. Results are given in figure from 7.8 to 7.10.

B-) R & D Laboratory test work with MECC Corporation:

R & D Laboratory test work was done operating agreement between China Metallurgical Equipment Corporation (MECC) and the ISDEMIR. The test results are summarized in great detail. Coke breeze consumption, basicity test, moisture test of raw mix, reducibility test and mineralogical analysis of sinter was investigated in this study. Also, tumbler index of product sinter in hot pot sintering tests were compared with sinter plants of different countries.

Five kinds of iron ore fines, namely A, B, C, D and E, shall be uniformly mixed respectively in order to reduce fluctuation of chemical composition and obtain representative sample. The five kinds of iron ore s are mixed as per the proportioning

ratio requirement (given in table 7.11). The homogenous sample after mixing shall be kept ready for test. Test operating parameters are given below:

Foreign iron ore fines (not supplied from Turkey) : domestic iron ore fines (supplied from Turkey) = 65 : 35; namely I.O.F. E : I.O (A+B+C+D) = 65 :35

A: B: C: D = 26: 40: 14:20

BF flue dust and iron bearing sludge account for 2.2% and 2.0% of raw mix respectively.

Test results are summarized as follows:

1) High sinter strength: At basicity of 2.1 and 1.7, the tumbler index of sinter reaches 72.0% and 67.7% respectively. According to data statistics obtained from test and actual production, the tumbler index of sinter produced by sinter machine is higher than that of sinter produced in laboratory. Those laboratory tests were carried out for NINL, India and Hangang, China sinter plant project. Therefore, the sinter strengths obtained at basicity 1.7 and 2.1 by these tests can be seen as high. Comparison of tumbler index of product sinter in hot pot sintering test indicated in tble 8.1.

Tablo 8.1 Comparison of tumbler index of product sinter in hot pot sintering test. (Tao et al., 2005)

Description	Basicity	ISO Tumbler Index +6.3mm%
Product sinter for NINL, India	2.43	67.67
	1.8	60.33
	1.6	55.33
Product sinter for Hangang, China	1.6~2.5(Plan I)	66.70
	1.6~2.5(Plan II)	69.70
Product sinter for Turkey	2.1	72.0
	1.7	67.7
	1.0	63.3

2) Optimum bed permeability: The high vertical sintering speed means good permeability of bed. Although there is degradation of some materials during sintering process due to serious crack caused by crystal water separation out, no remarkable negative influence on vertical sintering speed has been seen. That is because of coarse size of raw material that ensures good permeability of sintering bed. However, for iron ore fines with coarser grain size, like I.O.B (+10mm accounting for 25.0%, -10+8mm accounting for 9.5%) and I.O.A & C (+8mm accounting for 8% and 7% respectively), it's advised that crushing is to be conducted to achieve grain size less than 8mm, which is conducive to sintering process. These results are indicated in table 7.13.

3) High productivity: When basicity is higher, sinter productivity is obviously increased. For basicity of sinter is 1.0, the productivity obtains 1.80 $t/m^2.h$ ($43.2t/m^2.d$). While basicity of sinter is 1.7, the productivity reaches 1.88 $t/ m^2.h$ ($45.12t/m^2.d$) and basicity of sinter is 2.1 the productivity increased to 2.01 $t/ m^2.h$ ($48.24t/m^2.d$) respectively. These results are demonstrated in table 7.16.

4) Sinter strength: When basicity is higher, sinter strength is obviously increased. As basicity is 1.0, tumbler index is 63.3%. When basicity is 1.7, tumbler index is increased to 67.7%. When basicity is further up to 2.1, tumbler index is increased to 72.0%. These results are exhibited in table 7.16.

5) Vertical sintering speed: When basicity improves from 1.0 to 1.7, vertical sintering speed is increased slightly. When basicity is further upgraded upto 2.1, vertical sintering speed keeps unchanged. These are displayed in table 7.16.

6) High desulfurization efficiency: When basicity improves from 1.0 to 2.1, vertical desulfurization efficiency is decreased slightly. At sinter basicity of 1.0, 1.7 and 2.1, the desulfurization efficiency reaches 91.3%, 88.1% and 86.1% respectively. These results are shown in table 7.16.

7) Coke breeze consumption: With the increase of basicity, coke breeze consumption is more. When basicity is 1.0, coke breeze is 5.45%. When basicity is 1.7, coke breeze increased to 5.90 %. When basicity is further up to 2.1, coke breeze is increased to 6.36%.

These results are implied in table 7.15.

8) FeO in sinter: It is demonstrated from test data that along with the increase of basicity, total Fe content in sinter is significantly reduced, so is FeO content. With the increase of basicity from 1.0 up to 1.7 and 2.1, Fe in product sinter is reduced from 58.81% to 56.71% and 55.46% as well as FeO content is decreased from 14.68% to 12.60% and 11.28% respectively. These results are revealed in table 7.16.

9) Dunite fines addition: we carried out further comparative tests with 1.82 % and 3.64 % dunite fines added into raw mix for comparison with test results obtained from sintering without addition of dunite fines. The test results are shown in table 7.18. In consideration that sinter samples have provided dolomite fines instead of dunite in an aim to find possibility to further improve the metallurgical performance of sinter.

10) Additional material particle size: The grain size of limestone fines and coke fines is coarse (-3 mm 76.0 %), it will be further crushed until grain size of -3 mm more than 90.0 %. These results present in table 7.13.

11) The metallurgical properties of sinter: The reducibility of sinter increases with the increase of basicity of sinter. The reducibility of sinter increases remarkably from 60.43% to 72.1% and 76.34% while the basicity of sinter increases from 1.0 to 1.7 and 2.1. With the increase of sinter basicity from 1.0 to 1.7 and 2.1, the RDI+3.15 rnm. increases from 70.35% to 78.67% and 81.11%. The tests results are listed in table 7.21 and table 7.22.

12) Dolomite and dunite used: Test for the sintering performance of raw mix with dunite addition have shown that only the metallurgical property indexes are increased while other parameters such as tumbler index, sinter cake yield, final sinter ratio and sinter grade generally declined when dunite fines are added into sintering raw mix.Therefore, the dunite fines addition is not recommended. The final decision could be made according to actual conditions in the practice.

The following parameters are recommended:

Basicity: 1.7 or 2.1

Coke fines consumption: Coke fines consumption is 5.9 % at the sinter basicity of 1.7 or 6.36% at the sinter basicity of 2.1.

Moisture of raw mix: 7.0%

Ignition: Temperature 1150-1200°C, 1 minute, negative pressure 5880 Pa.

Negative Pressure for sintering: 11760 Pa

Sinter indexes obtained under working mode according to above mentioned parameters are listed in table 7.21 and 7.22.

C-) Statistic quality control of sinter samples:

Statistical process control (IPC) techniques are used for Iron and Steel Plants in which data are measured for certain periods to increase physical, chemical and reducibility properties of sinter production. Thus, some important advantages are provided in terms of control and applicability. In this study, Static and dynamic test, low temperature reduction disintegration and reduction under load of the samples taken out regularly from sinter plant of Isdemir.For each parameter, X-R control charts using moving range method were drawn. It was observed that quantity of raw ores and grades should be fixed using suitable homogenization. Results are given in figure from 7.12 to 7.31 and in table from 7.24 to 7.34.

D-) Studies on sinter quality with macrostructure, microstructure and mineralogical analysis:

The sintered samples were collected from ISDEMIR sintering plant. Various defects existing on the surface and inside of the sintered ore made easy to form crack propagates under the action of external forces and ambient media, which would finally lead to the breaking and cracking of the sintered ore. Porosity is low for S1 and S4 samples as shown in figure 7.38. But these sinter samples of micro porosity is

better. Macro porosity is high S2 and S3 samples and fractures may be sintered. It may be mentioned that while micro porosity of these samples changed quite significantly, the total porosity (consisting of micro- & macro-pores) of the sinter samples remained more or less same. We may infer that the reducibility of sinter was well correlated with the proportion of micro pores out of total pores available in the sinter. Total porosity of the samples corresponds to the combined micro porosity and macro porosity. It is apparent from the figure 7.37 that reducibility of the sinter also increased marginally when pyroxenite was replaced by dolomite.

The following conclusions can be drawn about cracks:

- The number of fine pores increased in the sinter when smaller sized fraction of limestone or/and coke breeze was used in the mix. Fine pores could be prevented from coalescing into coarser pores if expansion of the heat pattern is suppressed.

- Our results, similar to others, suggest that if the coalescence of fine pores was suppressed and the number of small pores was increased in the sintering process, the surface area of sinter could be increased to achieve a substantial improvement in its reducibility.

- The pores produced by thermal decomposition of sinter are mostly micropores measuring 18 μm or less, and the strengthening mentioned in the study has little effect. Also, the micropores of 10 μm or less reported do not readily combine regardless of the melt viscosity or liquid phase ratio, unlike the macro pores, and are retained in the texture of the sinter.

The studies show that a small increase in the alumina content of sinter blends can have a significant adverse impact on the strength and reduction degradation characteristics of final sinter leading to deterioration in gas permeability in the upper part of the blast. Tumbler strength tests also showed that the mechanical strength of sinter decreased with increase in $A1_2O_3$ content. In this study, iron ore B sintered with the lowest coke rate showed the largest increase in porosity; iron ore C sintered was less marked. For iron ore A sintered with a high coke rate, the porosity was

lowest and the porosity was reduced slightly as the alumina content increased (seen in figure 7.52).

The following conclusions can be drawn about mineralogical analysis:

- Calcium ferrite is the main bonding phase in sinter. It increases with an increase of basicity. Generally, the high content of calcium ferrite favors the tumbler strength of sinter, but probably not for the RDI. The hematite includes the unreacted and the secondary hematite and most of the secondary hematite is reoxidized hematite. With an increase in basicity, the reoxidized hematite decreased in sinter (seen in figure 7.39-b&d). The secondary hematite is the most disadvantage phase to the RDI of sinter.

- During the cooling stage, the calcium ferrite has previously been supposed to form from the solidification of the melt. However, in this study it has been clearly shown that a large amount of calcium ferrite also may be generated by the reaction of magnetite with silicate melt and oxygen (seen in figure-7.39 a&c).

- To summarize the mineralogical study, it apperas that the microstructure of sinter is mainly determined by the maximum temperature of sintering. Depending on whether this temperature exceeds or not the critical range 1300 ^{o}C-1320 ^{o}C, two very different types of structure are observed: Large amounts of fine needle like calcium ferrites, limited amount of magnetite below this value, massive structure with heavy iron oxides precipitation from liquid, plate like calcium ferrites over this value.

The results of the present study bring about some information on the relationships between sintering operating parameters and sinter reducibility, which is one of the major aspects of sinter quality. Reducibility of sinter is determined by the properties of ores and the sintering conditions, mostly the heat pattern. The various parameters affecting reducibility as well as their inter relationships are summarized in figure 7.48.

Ideal blast furnace performance with respect to high productivity, low consumption of reducing agents and constant hot metal quality can only be achieved employing a high quality sinter with the following characteristics:

- Optimum grain size distribution:
  - ✓ Grain size between approximately 5-50 mm
  - ✓ Harmonic diameter of >10 mm
- High sinter strength:
  - ✓ Shatter Index (SI) = >92%
- High reducibility
  - ✓ Reduction Index (Rl): > 65%
  - ✓ Reduction Disintegration Index (RDI <3.15 mm): < 20%
- High porosity
- Softening temperature above approximately 1250 °C, depending on total burden mixture
- Narrow cohesive zone temperature
- Constant FeO content in the range of 8%

The production of homogeneous sinter already commences in the blending beds and continues to the storage and proportioning bins. Special attention must be placed on the design of these facilities to prevent undesirable material segregation. For ideal results the operation of the proportioning, dosing and weighing system of the storage bins has to be computer controlled on a real time basis.

A decisive precondition for the production of high quality sinter is a homogeneous sinter raw mix of high permeability. All additives should be uniformly distributed throughout the mixture. A high and uniform permeability allows the bed height on the sinter machine to be increased, which accordingly lowers the fuel consumption for the sintering process. Excessively high sintering temperatures can thus be avoided, positively contributing to the sinter strength, the reducibility of the sinter and indirectly,

the FeO content, among other benefits. Sinter with an FeO content of less than 7% can only be produced with a sinter machine bed height of approximately 600 mm. In this context it is important to mention that the prober equipment must be installed to ensure that the fuel concentration continually decreases from the top to the bottom of the sinfer raw mix layer in order to prevent excessive sintering temperatures. The distance of the burn through point from the sinter machine discharge has a direct influence on the sinter machine capacity, the average FeO content of the sinter and the mean diameter of the sinter product.

Another aspect to consider is that most blast furnaces operate using a mixture of sinter, pellets and lump ore. Pellets and lump ore, however, are normally marketed with specific chemical compositions and qualities. An advantage in the use of sinter is that the burden can be optimized by adjusting the quality and the ratio of the charged sinter in accordance with the composition and characteristics of the other burden materials.

The production of high quality sinter depends to a high degree on the chemical composition of the raw materials, especially with respect to the gangue content (SiO_2, MgO, Al_2O_3), interstitial water and the CaO/SiO_2 ratio of the sinter raw mix. Particle grain size is paramount in importance. Investigations have shown that fuel consumption at the sinter plant and hence the FeO content of the sinter increases with an increasing fines content with grain sizes < 0.1 mm and >8 mm. The maximum grain size of the additives should be limited to approximately 3 mm with consideration to the targeted sinter strength, reducibility and porosity.

The results obtained in this study show very clearly that sintering is very complex process, controlled by several parameters that are interdependent. Although changing an ore blend can have a very profound effect on sintering performance and sinter strength, the link between the cause and the effect is not direct, as demonstrated in this study

It is believed that these approaches will contribute to understand more deeply the actual mechanisms of sintering, allowing thus the production of a sinter meeting closely the requirements for efficient blast furnace operation.

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