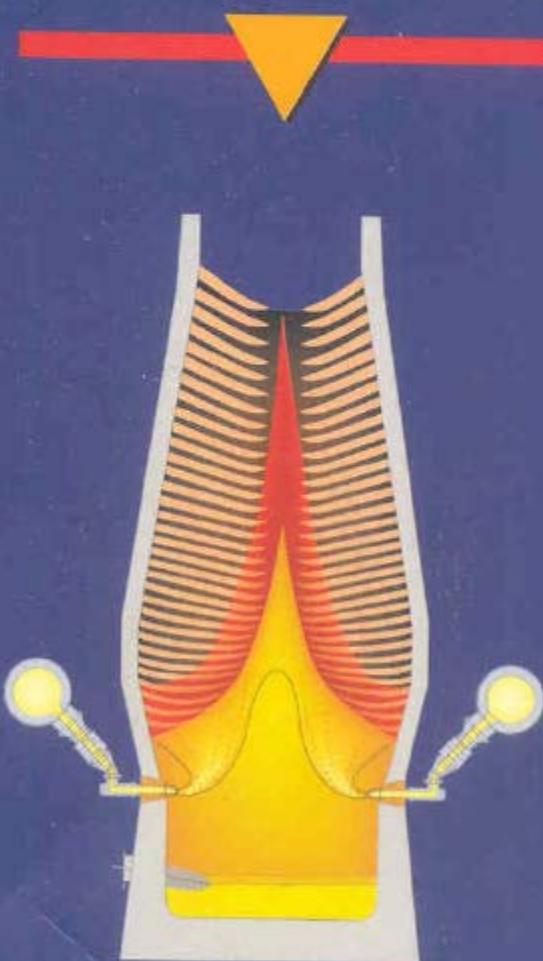


M. Geerdes, H. Toxopeus, C. van der Vliet

Modern Blast Furnace Ironmaking

an introduction



STAHL EISEN

M. Geerdes, H. Toxopeus, C. van der Vliet

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an introduction



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Printed in Germany

Distributors: Verlag Stahleisen GmbH
P.O. Box 105164
D-40042 Düsseldorf

ISBN 3-514-00713-6

Preface

The blast furnace can be described from various points of view. The mechanical engineer will describe the furnace as a strong steel shell with a refractory lining and a cooling system, which nevertheless is easily attacked by operating challenges. The chemical engineer will stress the abundance of chemical reactions taking place within the furnace. The operator will discuss the upsets frequently encountered when operating a blast furnace. This book reconciles the various points of view. The blast furnace process, i.e. what is going on within the furnace, is taken as a starting point.

The reason for writing this book was, that in our vision, optimisation of blast furnace operation is not only based on the experience showing best practices, but also on the conceptual understanding why something works or does not work. In other words, operational improvement is not only based on know-how but on know-why as well. The conceptual understanding of the process leads to different strategies to control the furnace, depending on the results of the analysis.

The authors have been involved in many operational improvement programs, and special blast furnace situations worldwide. They have developed their own understanding of the process and applied the results of their analysis in various operating furnaces. The present book is based on their experience of transferring their understanding to blast furnace operators. The term "modern" in the title refers to the relatively new application of coal injection into blast furnaces.

We are indebted to the people we have worked with. The book would not have been written without the extensive support of Danieli Corus BV in promoting blast furnace operational understanding (in particular Mrs. J. Alexander and Messrs. N. Bleijendaal, R. van Laar, B. Paramanathan, J. Plooij, E. van Stein Callenfels, P. Zonneveld). We thank XLmedia (Mr. E. Engel) for the editing.

We wish to thank people, who helped us in understanding the blast furnace process. First of all, we acknowledge the contributions of our home base steel works, Corus IJmuiden Blast Furnace Department and technologists T. Bakker, B. van den Berg, H. van den Berg, L. Bol, G. Flierman, R. Godijn, G.J. Gravemaker, F. Homminga, T. Huisman, J. van Ikelen, J. Jansen, W. Koen, C. Kolijn, K. van Laar, J. van Langen, R. Molenaar, J. de Pagter, E. Schoone, T. Spiering, A. Steeghs and G. Tijhuis. Furthermore, we are indebted to our many colleagues all over the world:

- In Argentina, at Siderar Messrs. E. Doro, J.M. Gonzales, F. Giandomenico and O. Lingiardi.
- in Australia, at OneSteel Messrs. P. Broadbent, A. Laird and J. Tsalapatis.
- in Italy, at ILVA Taranto works Messrs. S. DeFelice, V. Dimastromatteo, G. Dimaggio and G. Micci.
- in Mexico, at AHMSA Messrs. H. Buruato and M. Morales.
- in the United States, at Ispat Inland Messrs. W. Carter, P. Chaubal, M. Ranade, J. Ricketts and D. Zuke.
- in Germany, Mr. H. Lungen.

And many more who have not been mentioned.

IJmuiden, August 2004
Maarten Geerdes
Hisko Toxopeus
Cor van der Vliet

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List of Symbols and Abbreviations

B2, B3, B4	basicity, ratio of two, three or four components
bar	pressure, atmosphere relative
C	carbon
cm	centimetre
CO	carbon monoxide
CO ₂	carbondioxide
CRI	coke reactivity index
CSR	coke strength after reaction
Fe	iron
GJ	giga joule
H ₂	hydrogen
H ₂ O	water
HGI	hard grove index
HOSIM	hoogovens simulatie (blast furnace simulation)
HV	high volatile
IISI	International Iron & Steel Institute
ISO	International Organisation for Standardization
JIS	Japanese Industrial Standard
K	potassium
kg	kilogram
kmole	kilomole
LV	low volatile
m ³ STP	cubic metre at standard temperature and pressure
mm	millimetre
Mn	manganese
Mt	million ton
N ₂	nitrogen
Na	sodium
O ₂	oxygen
P	phosphorous
RAFT	raceway adiabatic flame temperature
RR	replacement ratio
s	second
Si	silicon
Standard Coke	coke with 87.5% carbon
t	ton
tHM	ton hot metal
Ti	titanium
VDEh	Verein Deutscher Eisenhüttenleute
VM	volatile matter

Chapter 1

Introduction of the blast furnace process

Two different process routes are available for the production of steel products, namely the blast furnace with oxygen steelmaking and the electric arc steelmaking route. The routes differ with respect to the type of products that can be made, as well as the raw materials used. The blast furnace–oxygen steelmaking route mainly produces flat products, while electric arc steelmaking is more focused on long products. The former uses coke and coal as the main reductant sources and sinter and pellets as the iron-bearing component, while the latter uses electric energy to melt scrap. The current trend is for electric arc furnaces to be capable of also producing flat products. Nevertheless, the blast furnace–oxygen steelmaking route remains the primary source for worldwide steel production, as shown in Figure 1.

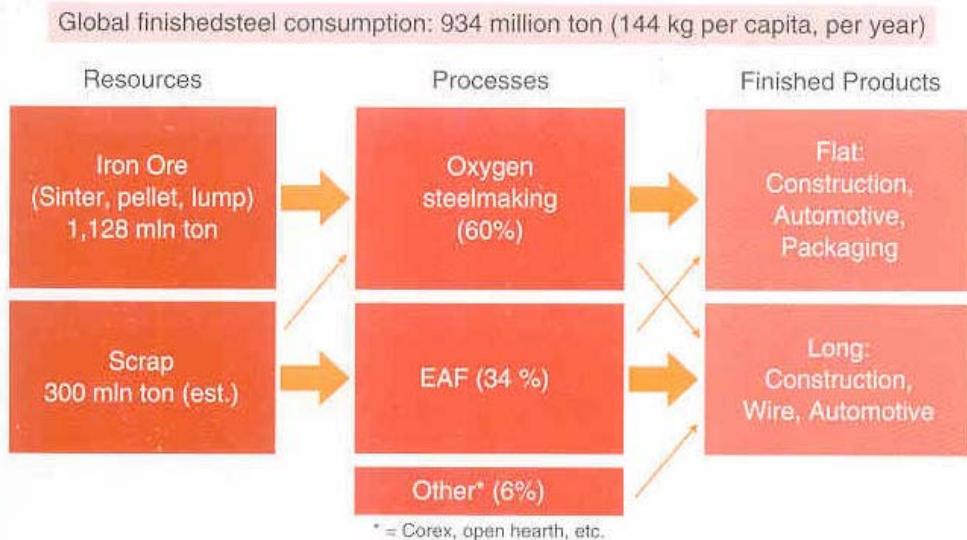


Figure 1: Steelmaking routes and raw materials (IISI Steel Statistical Yearbook, 2003)

1 – Introduction of the blast furnace process

Hot metal is produced in a blast furnace, from where it is transported as liquid hot metal to the steel plant where refinement of hot metal to steel takes place by removing elements such as sulphur, silicon, carbon, manganese and phosphorous. Good performance of the steel plant requires consistent hot metal quality of a given specification. Typically the specification demands silicon content between 0.3% and 0.7%, manganese between 0.2% and 0.4%, phosphorous in the range 0.06–0.08% or 0.1–0.13% and a temperature as high as possible.

In the blast furnace process iron ore and reducing agents (coke, coal) are transformed to hot metal and slag is formed from the gangue of the ore burden and the ash of coke and coal. Hot metal and liquid slag do not mix and remain separate from each other with the slag floating on top of the denser iron. The iron can then be separated from the slag in the casthouse, but before that can take place, the transformation to hot metal and slag must be understood.

Let us now consider the contents of a blast furnace at any given moment. Ore and coke are charged in discreet layers at the top of the furnace. From studies of quenched furnaces it was evident that these layers of ore and coke remain until the temperatures are high enough for softening and melting of the ore to begin. Quenched furnaces are “frozen in action” with the help of water or nitrogen and examples of quenched blast furnaces is presented in Figure 2. The quenched blast furnace shows clearly the layer structure of coke and ore. Further analysis reveals information about the heating and melting of the ore as well of the progress of chemical reactions (Bonnekamp et al, 1984).

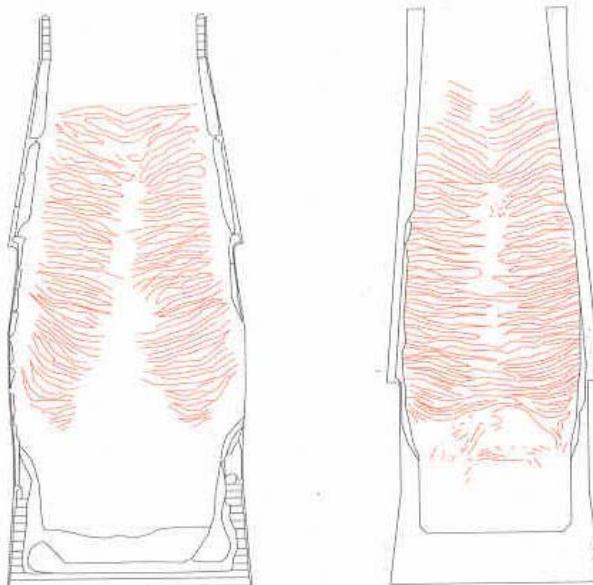


Figure 2: Dissections of quenched blast furnaces Kakogawa 1 and Tsurumi
(Based on Omori et al, 1987)

1 – Introduction of the blast furnace process

As indicated in Figure 3, at any moment, an operating blast furnace contains, from top downwards:

- Layers of ore and coke.
- An area where ore starts to soften and melt, known as the softening–melting zone.
- An area where there is only coke and liquid iron and slag, called the “active coke” or dripping zone.
- The dead man, which describes the coke in the hearth of the furnace.

A blast furnace has a typical conical shape. The sections from top down are:

- Throat, where the burden surface is.
- The shaft, where the ores are heated and reduction starts.
- The bosh parallel or belly and
- The bosh, where the reduction is completed and the ores are melted down.
- The hearth, where the molten material is collected and is cast via the taphole.

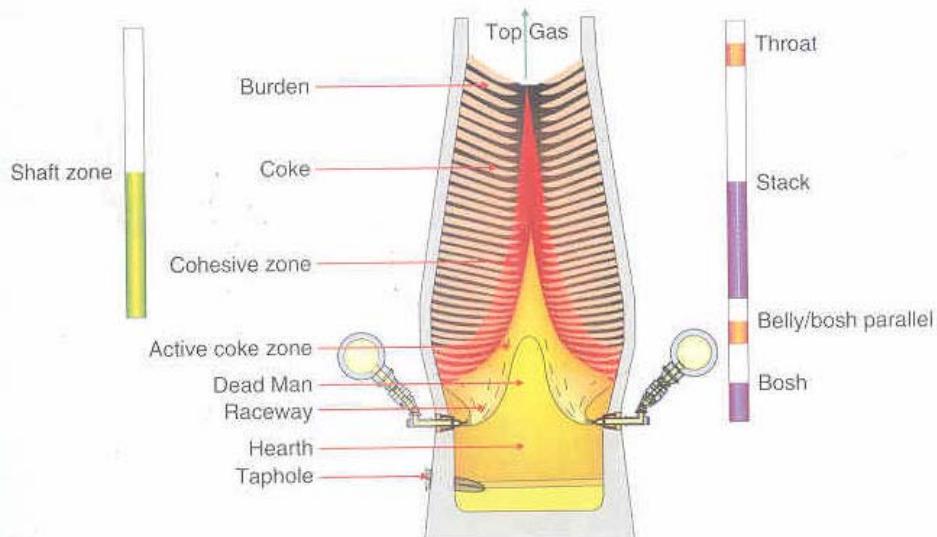


Figure 3: The zones in the blast furnace

1.1 What is driving the furnace?

1.1.1 Process Description

The driving forces in the blast furnace are illustrated in Figure 4 (next page)

- A blast furnace is filled with alternating layers of coke and the iron ore-containing burden.
- Hot blast is blown into the blast furnace via tuyeres. A tuyere is a cooled copper conical pipe numbering up to 12 in smaller furnaces, and up to 42 in bigger furnaces through which pre-heated air (up to more than 1,200°C) is blown into the furnace.

1 – Introduction of the blast furnace process

- The hot blast gasifies the reductant components in the furnace, those being coke as well as auxiliary materials injected via the tuyeres. In this process, the oxygen in the blast is transformed into gaseous carbon monoxide. The resulting gas has a high flame temperature of between 2,100 and 2,300°C . Coke in front of the tuyeres is consumed thus creating voidage.
- The very hot gas ascends through the furnace , carrying out a number of vital functions.
- Heats up the coke in the bosh/belly area.
- Melting the iron ore in the burden, creating voidage.
- Heats up the material in the shaft zone of the furnace.
- Removes oxygen of the ore burden by chemical reactions.
- Upon melting, the iron ore produces hot metal and slag, which drips down through the coke zone to the hearth, from which it is removed by casting through the taphole. In the dripping zone the hot metal and slag consume coke, creating voidage. Additional coke is consumed for final reduction of iron oxide and carbon dissolves in the hot metal, which is called carburisation.

The blast furnace can be considered as a counter current heat and mass exchanger, as heat is transferred from the gas to the burden and oxygen from the burden to the gas. Gas ascends up the furnace while burden and coke descend down through the furnace. The counter current nature of the reactions makes the overall process an extremely efficient one.

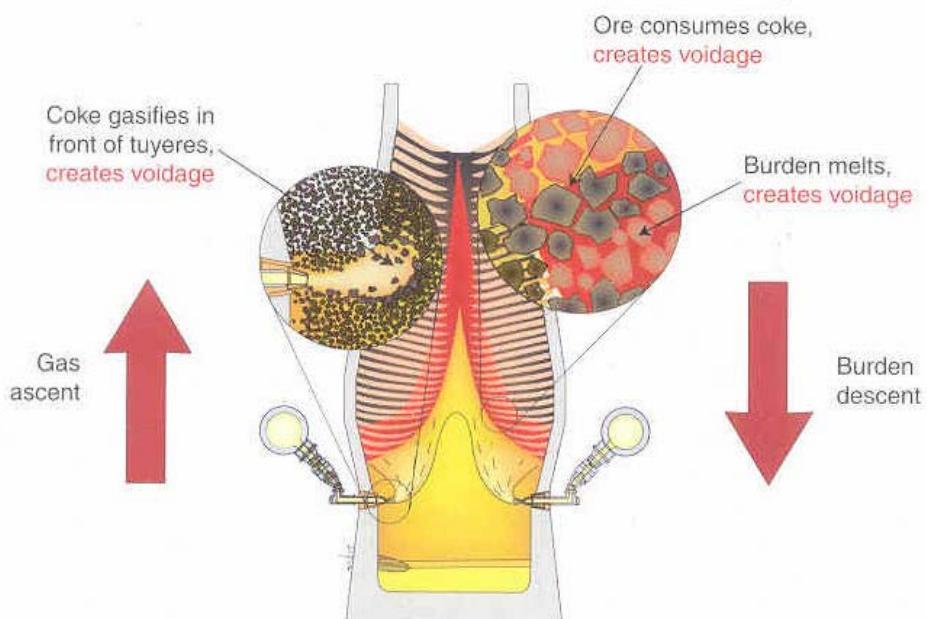


Figure 4: The driving force of a blast furnace: the counter current process creates voidage at the indicated areas causing the burden to descend

A typical example of the temperature profile in the blast furnace is shown in Figure 5. It is shown that the softening/melting zone is located in an area where temperatures

1 – Introduction of the blast furnace process

are between 1,100 and 1,450°C. The temperature differences in the furnace are large. In the example the temperature gradients are bigger in the horizontal direction than in the vertical direction, which will be explained in chapter 6.

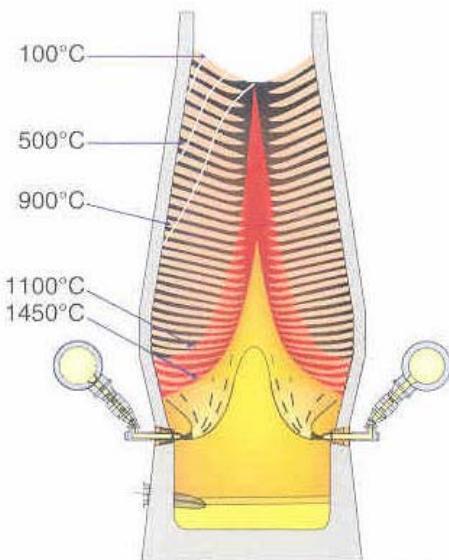


Figure 5: Temperature profile in a blast furnace (typical example)

1.2 The Equipment

1.2.1 Equipment overview

An overview of the major equipment is shown in Figure 6 (next page). These include:

- Hot Blast Stoves. Air preheated to temperatures between 1,000 and 1,250°C is produced in the hot blast stoves and is delivered to the furnace via a hot blast main, bustle pipe and finally through the tuyeres. The hot blast reacts with coke and injectants, forming the so-called raceway in front of the tuyeres.
- Stock house. The burden materials and coke are delivered to a stock house. The materials are screened and then weighted before final delivery into the furnace. The stock house is operated automatically. Corrections for coke moisture are generally made automatically. The burden materials and coke are brought to the top of the furnace via skips or via a conveyor belt, where they are discharged into the furnace in separate layers of ore and coke.
- Gas cleaning. The top gas leaves the furnace via uptakes and a down-comer. The top gas will contain many fine particles and so to remove as many of these as possible the top gas is lead through a dust catcher and wet cleaning system.
- Casthouse. The liquid iron and slag collect in the hearth of the furnace, from where they are tapped via the taphole into the casthouse and to transport ladles.
- Slag granulation. The slag may be quenched with water to form granulated slag, which is used for cement manufacture.

1 – Introduction of the blast furnace process

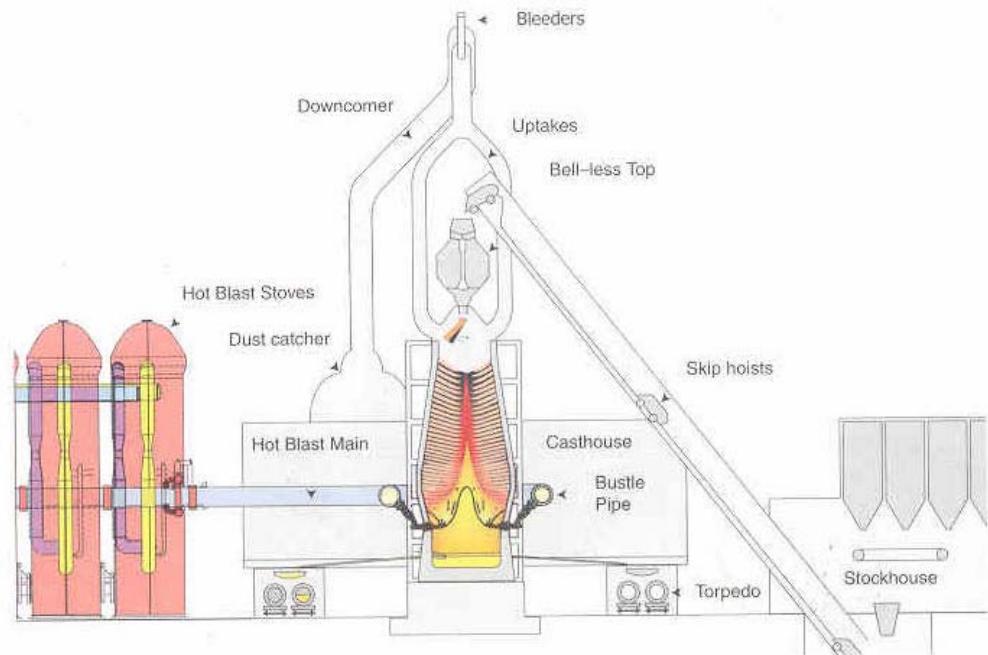


Figure 6: Blast furnace general arrangement

The top of the blast furnace is closed, as modern blast furnaces tend to operate with high top pressure. There are two different systems:

- The double bell system, which for burden distribution purposes, requires movable throat armour.
- The bell less top, which allows easier burden distribution.

Examples of both types are schematically shown in Figure 7.

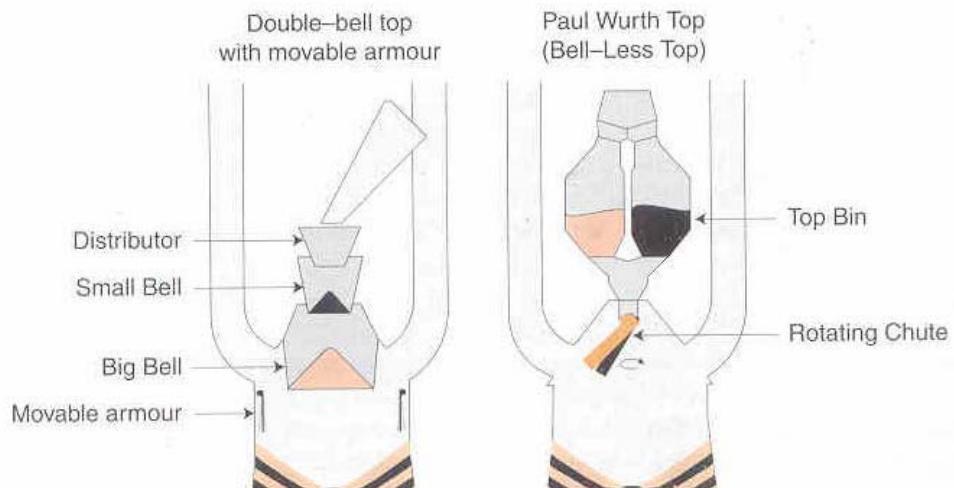


Figure 7: Blast furnace top charging systems

1 – Introduction of the blast furnace process

1.2.2 Blast furnace construction

There are basically two construction techniques to support blast furnaces. The classic design utilises a supported ring, or lintel at the bottom of the shaft, upon which the higher levels of the furnace rests. The other technique is a freestanding construction requiring an independent support for the blast furnace top and the gas system. The required expansion (thermal as well as from the pressure) for the installation is below the lintel that is in bosh/belly area for the lintel furnace, while the compensator for expansion in the freestanding furnace is at the top, as indicated in Figure 8.

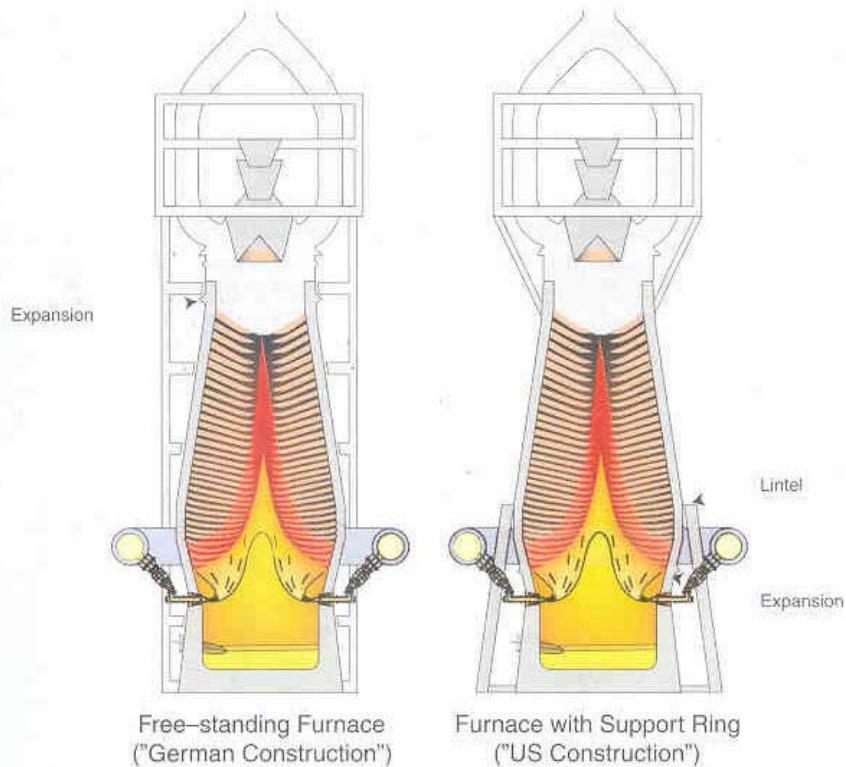


Figure 8: Blast furnace constructions

1.2.3 Blast furnace development

Blast furnaces have grown considerably in size during the 20th century. In the early days of the 20th century, blast furnaces had a hearth diameter of 4 to 5 metres and were producing around 100,000 tonnes hot metal per year, mostly from lump ore and coke. At the end of the 20th century the biggest blast furnaces had between 14 and 15 m hearth diameter and were producing 3 to 4 MT per year. A typical development of blast furnaces is shown in Table 1 for the situation of the IJmuiden steel works. The ore burden developed, so that presently high performance blast furnaces are fed with sinter and pellets. The lump ore percentage has generally decreased to 10 to 15%

1 – Introduction of the blast furnace process

or lower. The reductants used developed as well: from operation with coke only to the use of injectant through the tuyeres. Mainly oil injection in the 1960's, while since the early 1980's coal injection is used extensively. Presently, about 30 to 40% of the earlier coke requirements have been replaced by injection of coal and sometimes oil and natural gas.

Table 1: Development of the blast furnaces at Corus IJmuiden, The Netherlands

Blast Furnace No.		1	2	3	4	5	6	7
Hearth Diameter	m	4.8/5.6	4.8/5.6	5.2/5.9	8.5	8/9	10/11	13/13.8
Working Volume	m ³	519	519	598	1413	1492	2328	3790
Built		1924	1926	1930	1958	1961	1967	1972
Initial Productivity	t/d	280	280	360	1,380	1,700	3,000	5,000
Most Recent Productivity	t/d	1,000	1,000	1,100	3,500	3,700	7,000	10,500
Last Renovation							2002	1991
Demolished		1974	1974	1991	1997	1997		

The size of a blast furnace is often expressed as its hearth diameter or as its "working volume" or "inner volume". The working volume is the volume of the blast furnace that is available for the process i.e. the volume between the tuyeres and the burden level. Definitions of working volume and inner volume are given in Figure 9.

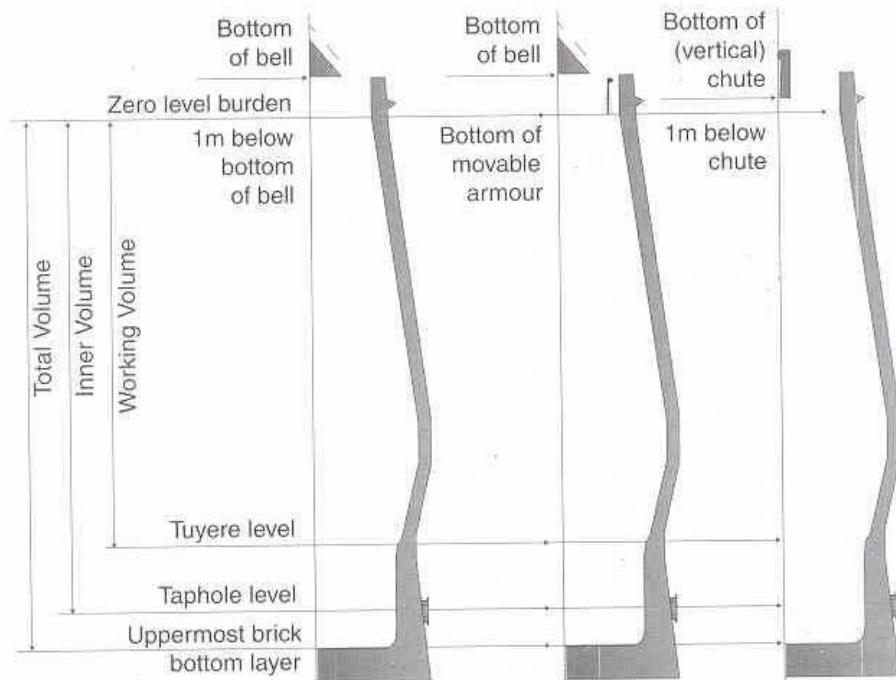


Figure 9: Definitions of working volume and inner volume

1 – Introduction of the blast furnace process

Presently, very big furnaces reach production levels of 12,000 t/d or more. E.g. the Oita blast furnace No. 2 (NSC) has a hearth diameter of 15.6 meter and a production capacity of 13,500 t/d. In Europe, the Thyssen-Krupp Schwelgern nr 2 furnace has a hearth diameter of 14.9 m and a daily production of 12,000 t/d.

1.3 Book Overview

Blast furnace ironmaking can be discussed from 3 different perspectives:

- The operational approach: discussing the blast furnace with its operational challenges.
- The chemical technology approach: discussing the process from the perspective of the technologist who analyses progress of chemical reactions and heat and mass balances.
- The mechanical engineering approach focussing on equipment.

The focus of this book is the “operators view”, with the aim to understand what is going on inside the furnace. To this end the principles of the process are discussed (Chapter 2) followed by the demands on burden quality (Chapter 3) and coke and auxiliary reductants (Chapters 4 and 5). Simplified calculations of burden and top gas are made (Chapter 6). The control of the process is discussed in Chapter 7: burden descent and gas flow control. The issues pertinent to understanding the blast furnace productivity and efficiency are presented in Chapter 8. Subsequently, hot metal and slag quality (Chapter 9), casthouse operation (Chapter 10) and special situations like stops and starts (Chapter 11) are discussed.

1.4 Exercises

The exercises refer to a large modern blast furnace operated on pulverised coal injection and equipped with high top pressure and high blast temperature. The working volume is 3,800 m³ and it has four tapholes.

The furnace is producing 10,000 tonne/day with a slag volume of 250 kg/tHM. As a reductant the furnace uses 300 kg coke per tonne hot metal (tHM) and 200 kg pulverised coal injection (PCI).

Questions (The answers to this exercise can be found in Annex III.):

1. The International Iron and Steel Institute (IISI) estimates that the worldwide pig iron production was 575.7 Mt in 2001. How many of these “modern furnaces” are required to produce this amount? And—if there are three of these furnaces on one site—how many pig iron producing sites are necessary?
2. Calculate the annual usage of pellets and coal at such a site. How much finished steel products are produced? Use the following data:
 - a) From 1 tonne iron 1.1 tonne liquid steel is produced and 0.9 tonne steel finished products.
 - b) To produce 1 tonne iron, one requires 1.6 tonne pellets.
 - c) Coke is produced mainly on local coke plants. 25% of the coal is gasified during cokemaking.

Chapter 2

The blast furnace: contents and gas flow

2.1 The generation of gas and permeability of the burden

The blast furnace is a counter current reactor (Figure 10). The driving force is the hot blast consuming coke at the tuyeres. In this chapter the gas flow through the furnace is analysed in more detail. The charge consists of alternating layers of ore burden (sinter, pellets, lump ore) and coke. The burden is charged cold and wet into the top of the furnace, while at the tuyeres the hot blast gasifies the hot coke. Towards the burden stockline (22 to 25 m from tuyeres to burden surface) the gas temperature drops from a flame temperature of 2,200 °C to a top gas temperature of 100 to 150 °C.

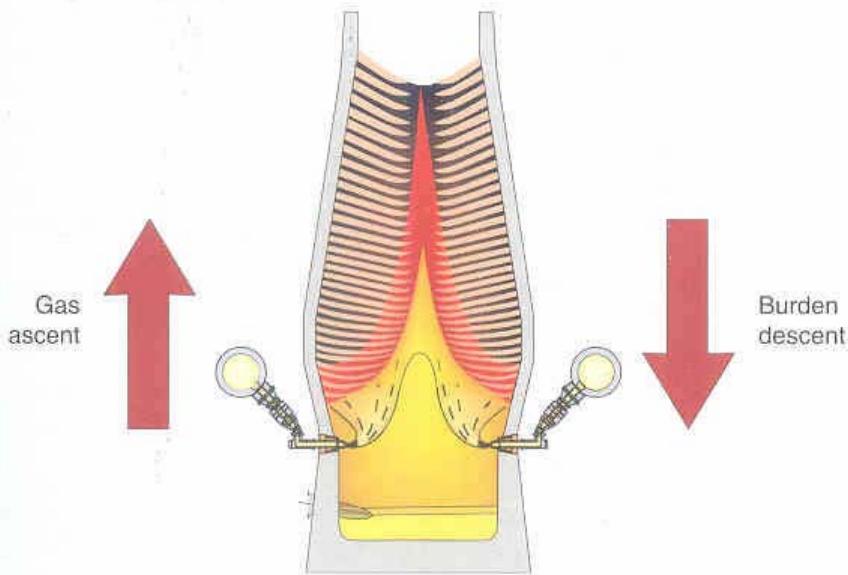


Figure 10: The blast furnace as a counter current reactor

2 – The blast furnace: contents and gas flow

The process starts with the hot blast through the tuyeres, which gasifies the coke and coal in the raceway (Figure 10). The reactions of the coke create hot gas, which is able to melt the ore burden. Consumption of coke and melting of the ore burden creates space inside the furnace, which is filled with descending burden and coke. The oxygen in the blast will gasify the coke to generate carbon monoxide (CO). For every molecule of oxygen 2 molecules of carbon monoxide are formed. If blast is enriched from its base level of 21% to 25% oxygen, then every cubic meter (m³ STP) oxygen will generate 2 m³ STP of CO. So if the blast has 75% of nitrogen and 25% of oxygen, the bosh gas will consist of 60% (i.e. 75%/(75%+2x25%)) nitrogen and 40% CO gas. In addition a huge amount of heat is generated in the raceway from the combustion of coke and coal (or oil, natural gas). The heat leads to a high flame temperature, which generally is in the range of 2,000 to 2,300°C. Since this temperature is higher than the melting temperature of iron and slag, the heat in the hot gas can be used to melt the burden. Flame temperature is discussed in more detail in section 8.1.3.

The hot gas ascends through the ore and coke layers to the top of the furnace. If there was only coke in the blast furnace, the chemical composition of the gas would remain constant but the temperature of the gas would lower as it comes into contact with the colder coke layers high in the furnace. A presentation of the gas flowing through a blast furnace filled with coke is presented in Figure 11. The gas flow has a prop-flow character. To the experienced blast furnace operator the furnace filled with coke only may seem a theoretical concept. However, in some practical situations, like the blow-in of a new furnace or when taking a furnace out of operation for a long time (banking) the furnace is almost entirely filled with coke.

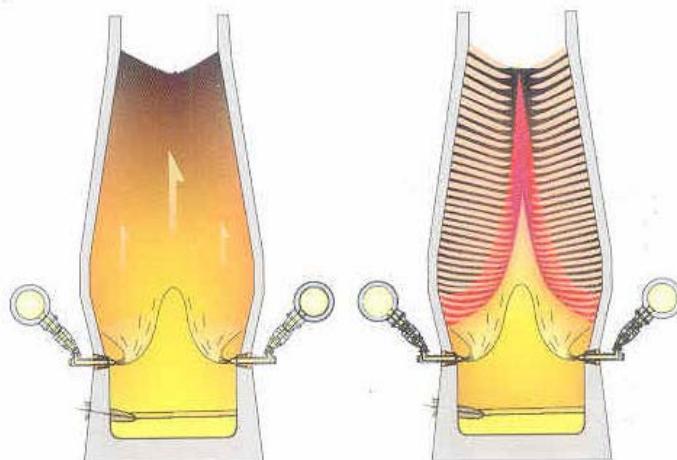


Figure 11: Gas flows in a furnace filled with coke only (left) and in a furnace filled with alternating layers of coke and ore (right).

In the normal operational situation the furnace is filled with alternating coke and ore layers. About 35 to 45 layers of ore separate the coke. It is important to note that the permeability of coke is much better than the permeability of ore (see also Figure 36). This is due to the fact that coke is much coarser than sinter and pellets and that the void fraction within the coke layer is higher. For example, the mean size of coke in a

2 – The blast furnace: contents and gas flow

blast furnace is typically 45 to 55 mm, while the average size of sinter is 18 to 25 mm and of pellets is 10 to 12 mm. Consequently, the burden layers determine how the gas flows through the furnace, while the coke layers function as gas distributors.

If gas flows from the bosh upwards, what happens to the gas as it gradually cools down?

Firstly, the heat with a temperature in excess of 1,400°C, the melting temperature of the slag, is transferred to the layered burden and coke, causing the metallic portion to melt. In the temperature range from 1,400 to 1,100 °C the burden will soften and stick together rather than melt. In the softening and melting zone the remaining oxygen in the ore burden is removed , which generates additional carbon monoxide. This is referred to as the direct reduction reaction (see section 7.2.1), which only occurs in the lower furnace.

The gas has now cooled to about 1,100°C and additional gas has been generated. Since the direct reduction reaction costs a lot of energy, the efficiency of the furnace is largely dependant on the amount of oxygen removed from the burden materials before reaching this 1,100°C temperature.

In summary:

- Heat is transferred from the gas to the ore burden, which melts and softens (over 1,100°C).
- Residual oxygen in the burden is removed and additional CO is generated. This is known as the direct reduction reaction.

Upon further cooling down the gas is capable of removing oxygen from the ore burden, while producing carbon dioxide (CO_2). The more oxygen that is removed, the more efficient the furnace is. Below temperatures of 1,100°C the following takes place:

- Heat is transferred from the gas to the burden.
- CO_2 gas is generated from CO gas, while reducing the amount of oxygen of the ore burden. This is called the gas reduction reaction, and in literature it is sometimes called “indirect reduction” as opposed to “direct reduction”. No additional gas is generated during this reaction.
- A similar reaction takes place with hydrogen. Hydrogen can remove oxygen from the burden to form water (H_2O).

Higher in the furnace, the moisture in the burden and coke evaporates and so is eliminated from the burden before any chemical reactions take place.

If we follow the burden and coke on its way down the stack, the burden and coke are gradually heated up. Firstly the moisture is evaporated, and at around 500°C the removal of oxygen begins. A simplified schedule of the removal of oxygen from the ore burden is shown in Figure 12 (next page).

2 – The blast furnace: contents and gas flow

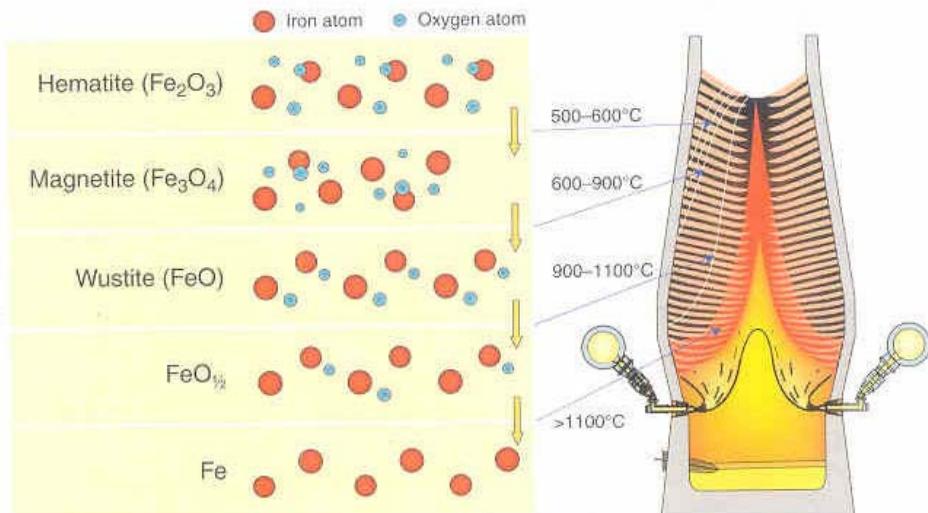


Figure 12: Schematic presentation of reduction of iron oxides and temperature

The first step is the reduction of the so-called hematite (Fe_2O_3) to magnetite (Fe_3O_4). The reduction reaction generates energy, so it helps to increase the temperature of the burden. In addition, the reduction reaction creates tension in the crystal structure of the burden material, which may cause the crystal structure to break into smaller particles. This property is called low-temperature disintegration. Several tests are available to quantify the effects (see chapter 3). Further down in the furnace the temperature of the burden increases gradually until the burden starts to soften and to melt in the cohesive zone. The molten iron and slag are collected in the hearth.

We now consider the interaction between the gas and the ore burden. The more the gas removes oxygen from the ore burden, the more efficient the blast furnace process is. Consequently, intimate contact between the gas and the ore burden is very important. To optimise this contact the permeability of the ore burden must be as high as possible. The ratio of the gas flowing through the ore burden and the amount of oxygen to be removed from the burden must also be in balance.

Experience has shown that many problems in the blast furnace are the consequence of low permeability ore layers. Therefore, the permeability of the ore layers across the diameter of the furnace is a major issue. The permeability of an ore layer is largely determined by the amount of fines (under 5 mm) in the layer. Generally, the majority of the fines are generated by sinter, if it is present in the charged burden or from lump ores.

The problem with fines in the furnace is that they tend to concentrate in rings in the furnace. As fines are charged to the furnace they concentrate at the point of impact where the burden is charged. They are also generated by low temperature reduction-disintegration. Thus, it is important to screen the burden materials well, normally with 5 or 6 mm screens in the stock house, and to control the low temperature reduction-disintegration characteristics of the burden.

2.2 Furnace Efficiency

The process efficiency of the blast furnace, generally considered to be the reductant rate per tonne hot metal, is continuously monitored through measurement of the chemical composition of the top gas.

The efficiency is expressed as the gas utilisation, that is the percentage of the CO gas that has been transformed to CO₂, as defined in the following expression:

$$\text{CO gas utilization} = \frac{\text{CO}_2\%}{(\text{CO}\% + \text{CO}_2\%)}$$

In addition, at modern furnaces the gas composition over the radius is frequently measured. The latter shows whether or not there is a good balance between the amount of reduction gas and the amount of ore in the burden. The wall zone is especially important and so the coke percentage in the wall area should not be too low. The wall area is the most difficult place to melt the burden as that is where the burden thickness is at its highest across the radius, and also because the gas at the wall loses much of its temperature to cooling losses.

The top gas analysis gives a reasonably accurate indication of the efficiency of the furnace. When comparing different furnaces one should realise that the hydrogen also takes part in the reduction process (paragraph 7.2.4).

The gas utilisation also depends on the amount of oxygen that must be removed. Since pellets have about 1.5 atoms of oxygen per atom of Fe (Fe₂O₃) and sinter has about 1.45 (mix of Fe₂O₃ and Fe₃O₄), the top gas utilisation will be lower when using sinter. It can be calculated as about 2.5% difference of the top gas utilisation, when comparing an all pellet burden with an all sinter burden.

2 – The blast furnace: contents and gas flow

2.3 Exercises: Gas flow and the contents of a blast furnace

The contents of a blast furnace can be derived from operational results. How long do the burden and gas reside within the furnace?

Consider an example of a large, high productivity blast furnace with a 14 metre hearth diameter. It has a daily production of 10,000 t hot metal (tHM) at a coke rate of 300 kg/tHM and a coal injection rate of 200 kg/t. Moisture in blast and yield losses are neglected. Additional data is given in Table 2.

Table 2: Data for calculation of blast furnace contents

	Consumption	Specific weight	Carbon content
Ore burden	1,580 kg/tHM	1,800 kg/m ³	
Coke	300 kg/tHM	470 kg/m ³	87 %
Coal	200 kg/tHM		78 %
Blast Volume	6,500 m ³ STP/min	1.3 kg/m ³ STP	
Top Gas		1.43 kg/m ³ STP	
O ₂ in blast	25.6 %		
Working volume	3,800 m ³	(500 m ³ used for active coke zone)	
Throat diameter	10 m		
A charge contains	94.8 t ore burden	18 t coke	
A ton hot metal contains	945 kg Fe	45 kg carbon	
Voidage in shaft	30 %		

1 tonne hot metal contains 945 kg Fe = 945/55.6 = 17.0 kmole

Questions (answers in Annex III):

1. How much blast oxygen is used per tonne hot metal?
2. How often are the furnace contents replaced?
3. How many layers of ore are in the furnace at any moment?
4. What happens with the carbon in the coke and coal?
5. How much top gas do we get?
6. Estimate how long the gas remains in the furnace
7. If you get so much top gas, is there a strong wind in the furnace?

Chapter 3

The ore burden: sinter, pellets, lump ore

3.1 Introduction

In the early days of ironmaking, blast furnaces were often located close to ore mines. In those days, blast furnaces were using local ore and charcoal, later replaced by coke. In the most industrial areas of the 19th century, many blast furnaces were operating in Germany, England and the US. After the application of the steam engine for ships and transportation, the centre of industrial activity changed from the ore sources to the major rivers, like the river Rhine, and later from the river banks to the coast. This trend may appear clear at present, but has a short history. As an example: in 1960 there were 60 operating blast furnaces in Belgium and Luxemburg. Presently (2004), only six are operating, of which two have the favourable coastal location.

The trend towards fewer but larger furnaces has made the option for a rich iron burden a more attractive one. A rich iron burden translates into a high Fe content and as fine ores are too impermeable to gas, the choice is narrowed down to sinter, pellets and lump ores. Sinter and pellets are both formed by agglomerating iron ore fines from the ore mines and have normally undergone an enrichment process, which is not described here. The quality demands for the blast furnace burden are discussed and the extent to which sinter, pellets and lump ore meet these demands.

A good blast furnace burden consists for the major part of sinter and/or pellets (Figure 13, next page). Sinter burdens are prominent in Europe and Asia, while pellet burdens are used in Northern America and Scandinavia. Many companies use sinter as well as pellets although the ratios vary widely.



Figure 13: Burden materials

Lump ores are becoming increasingly scarce and generally have poorer properties for the blast furnace burden. For this reason it is used mainly as a cheap alternative for pellets. For high productivity low coke rate blast furnace operation the maximum lump ore rate is in the range of 10% to 15%. The achievable rate depends on lump ore quality and the successful use of higher percentages is known.

3.2 Quality demands for the blast furnace burden

3.2.1 Qualitative description

The demands for the blast furnace burden extend to:

- The chemical composition of the burden. After the reduction and melting processes the correct iron and slag compositions must be made and this will be determined by the chemical composition of the materials charged in the furnace.
- The permeability for gas flow. Good resistance against degradation and no swelling of the material upon heating.
- The softening and melting properties. Fast transition from solid to liquid state.

The reducibility of the burden is controlled by the contact between gas and the burden particles as a whole, as well as the gas diffusion into the particles. Whether or not good reduction is obtained in the blast furnace is governed by the layer structure of the burden, the permeability of the layers and the blast furnace internal gas flow. The reducibility of the burden components will be of less importance if the gas flow within the furnace does now allow sufficient contact for the reactions to take place.

In the shaft zone the permeability of the burden is determined by the amount of fines (see Figure 14, next page). Fines may be defined as the fraction of the material under 5 mm. If there are too many fines, the void fraction used for the transport of the reduction gas will diminish and will affect the bulk gas flow through the burden (Hartig et al, 2000). There are two sources for fines, those that are directly charged into the furnace, and those that are generated in the shaft by the process.

3 – The ore burden: sinter, pellets, lump ore

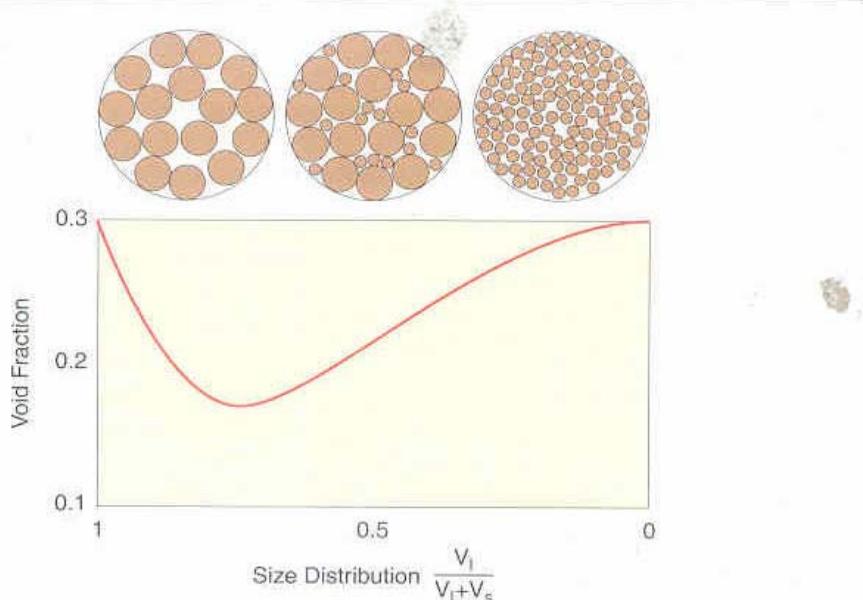


Figure 14: Permeability for gas flow depends on void fraction, which depends on the ratio of smaller and larger particles. Example of two types of spherical particles, which are blend in the ratio Volume small (V_s) and Volume large (V_l)

During the first reduction step from hematite to magnetite the structure of the burden materials weakens and fines are generated. Sinter and lump ore are especially prone to this effect, known as reduction-disintegration.

A major requirement for the blast furnace burden is to limit the level of fines within the furnace to as low as possible. This can be achieved by;

- Proper screening of burden materials before charging. Screens with 5 to 6 mm holes are normal operational practice.
- Good reduction-disintegration properties.

During charging, fines in the burden material tend to concentrate at the point of impact on the burden surface. The level of reduction-disintegration increases in areas where the material is heated and reduced slowly. A charged ring of burden with a high concentration of fines will impede gas flow, experience the slower warm-up and so result in a higher level of reduction-disintegration.

As soon as burden material starts softening and melting, the permeability for gas is greatly reduced. Therefore, the burden materials should start melting at relatively high temperatures. So that they do not impede gas flow while they are still high up the stack. Melting properties of burden materials are determined by the slag composition.

Melting of pellets and lump ore starts at temperatures of 1,000 to 1,100°C, while basic sinter generally starts melting at higher temperatures.

3.2.2 Ore burden quality tests

Ore burden material is characterised by the following.

- Chemical composition.
- Size distribution, which is important for the permeability of ore burden layers in the furnace.
- Metallurgical properties with respect to:
 - Cold strength, which is used to characterise the degradation of ore burden materials during transport and handling.
 - Reduction-disintegration, which characterises the effect of the first reduction step and is relevant in the stack zone of the furnace.
 - Softening and melting properties, which are important for the formation of the cohesive or melting zone in the furnace.

It is important for permeability to have a narrow size range and have minimal fines (less than 5% below 5 mm, after screening in the stockhouse). Measurement of the percentage of fines after screening in the stockhouse, although cumbersome, might give indications whether or not excessive fines are charged into the furnace.

A short description of tests used for characterisation of materials is given below with the objective being to understand the terminology. In many situations tumbler tests are used, where a sample of material is tumbled in a rotating drum for a fixed number of rotations. The size distribution after tumbling is determined and used as a quality indicator (Figure 15).

Principle of tumble test:
Sample is tumbled at fixed
number of rotations. Size
distribution determined
after tumbling. Weight
percentages over or below
certain screen sizes are
used as a quality parameter.

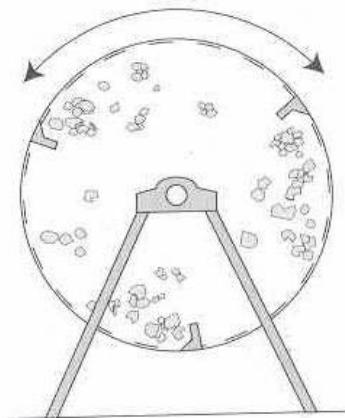


Figure 15: Principle of tumbler test

3.2.2.1 Tests for cold strength

Cold strength is mostly characterised by a tumbler test. For this test, an amount of material is tumbled in a rotating drum for a specified time interval. Afterwards the amount of fines are measured.

For pellets the force needed to crack the pellets, referred to as the cold compression strength, is determined.

3 – The ore burden: sinter, pellets, lump ore

3.2.2.2 Tests for reduction–disintegration

The reduction–disintegration tests are carried out by heating a sample of the burden to at least 500°C and reducing the sample with gas containing CO (and sometimes H₂). After the test the sample is cooled, tumbled and the amount of fines are measured. The quoted result is the percentage below 3.15 mm.

The HOSIM test (blast furnace simulation test) is a test where the sample is reduced to the endpoint of gas–reduction in a furnace. The sample is then tumbled after the test. The results are the reducibility defined by the time required to reduce the sample to the endpoint of gas reduction, and the reduction–disintegration represented by the percentage of fines (under 3.15 mm) after tumbling. Both tests simulate the upper part of the blast furnace process. The more advanced HOSIM test gives a more realistic description of the effects in the blast furnace.

3.3 Sinter

3.3.1 Production

Sinter making began as an efficient way to reuse the plant revert materials in the blast furnace. The sinter making process uses heat to fuse separate iron ore fines into larger particles that are suitable for charging to the blast furnace. The strength of the cohesive force depends on the amount and type of molten material between the individual particles.

Dwight and Lloyd constructed the first continuous sinter plant in 1906. A schematic presentation is shown in Figure 16. Sinter quality has improved progressively and in a number of countries (Europe, Japan, Brazil, Korea) sinter is the predominant blast furnace iron source. Plant reverts like dust from the blast furnace dust catcher, metallic fines from the blast furnace screens, mill scale and other materials are reused in sinter making. Modern, large sinter strands are 5 metres wide and an effective sinter area of 400 m². Productivity is typically 30 to 45 t/m²/day.

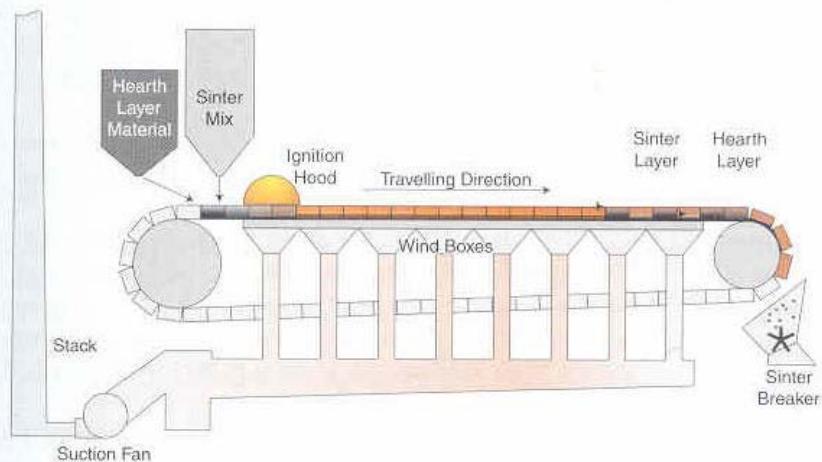


Figure 16: Sinter plant

3 – The ore burden: sinter, pellets, lump ore

After delivery of the ore fines to the steel works the sinter fines are stored and blended in pre-determined ratios. The blending can be done by mixing appropriate quantities on a conveyor belt before arrival at the sinter plant or with a blending pile. These piles are constructed by depositing the different iron ores and fluxes in layers in a longitudinal direction and then perpendicularly reclaiming them. This method of layering and reclaiming is to ensure a homogeneous mix.

The sinter feed also contains limestone, return sinter and coke breeze. The blend is mixed in a rotating drum, where water is added up to 5 to 7% for primary bonding between ore particles. It is important that the blend has a good permeability, so that air can be sucked through the blend and the coke breeze (and magnetite) can generate the heat required for the sintering process.

The blend is deposited in layers of 35 to 65 cm height on the sinter strand. The bottom layer is always made of a small layer of sinter (typical 15 to 25 mm size range) to prevent the flame front from reaching the sinter cars. The sintering process starts with ignition of the top layer with fans sucking the heated air down through the sinter bed. The fuels in the bed generate the heat to melt and fuse the ore particles together. At the end of the sinter strand the flame front has passed all the way through the blend to the bottom. The sinter cake then drops off the end of the strand into a collecting bin. From there it is broken up into manageable chunks, cooled and screened into appropriate fractions. The sinter fines (under 5 mm) are reused in the sinter blend, some of the material (15 to 25 mm) is recycled for the bottom strand layer, and the remaining sinter is sent to the blast furnace bins. The process on the sinter strand is shown in Figure 17.

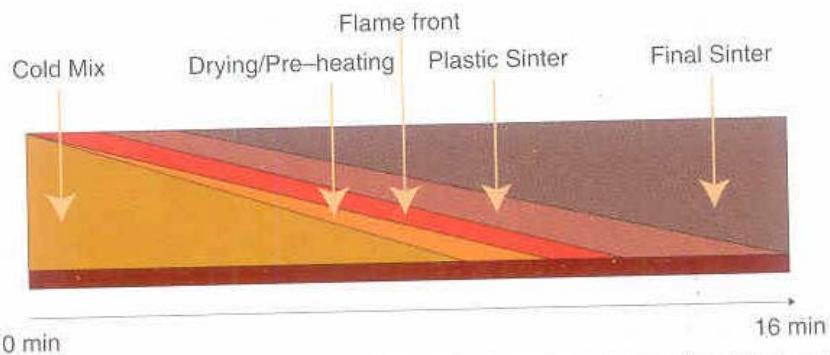


Figure 17: Sinter production progress: the temperature profile of the blend during sintering

From a chemical point of view it is important to note that the lime (CaCO_3 , CaO) added to the sinter is important as a flux for the blast furnace. In order to distinguish between the levels of flux in the sinter, it is categorised into acid sinter, self-fluxing sinter and super fluxed sinter. Self-fluxing sinter brings the lime required to flux its acid components (SiO_2). Super-fluxed sinter brings extra CaO to the blast furnace.

For self-fluxing and super-fluxed sinter, the lime reduces the melting temperature of the blend and at relatively low temperatures (1,100 to 1,300°C) strong bonds are

3 – The ore burden: sinter, pellets, lump ore

formed in the presence of FeO. The heat content of the flame front is lower with fluxed sinter than with acid sinter, which results in lower thermal exposure of the sinter strand and less wear on the plant.

The advantage of adding limestone outside the blast furnace instead of in the furnace is, that the decarbonisation reaction of the limestone ($\text{CaCO}_3 \rightarrow \text{CaO} + \text{CO}_2$, cost 117 kJ/mole) takes place outside the blast furnace and the required heat is not taken from the blast furnace gas flow.

Basic points influencing sinter strand productivity, costs and operation are:

- The selection of the correct blend of sinter feed is important from a cost point of view. Most companies use "value-in-use" models attributing premiums and penalties to all aspects of the sinter blend. Not only its Fe-value, but also the fuel requirements and the value of other elements like phosphorous, which will have a penalty.
- The correct blend includes the size distribution of the various materials. Very fine materials cannot easily be sintered because of the negative impact on the sinter bed permeability. The coke breeze size and amount are important for the sinter quality.
- The chemical composition of the required sinter is important. At higher basicity, sinter productivity increases. MgO content must be over 1.4% for good break down properties during reduction (see section 3.3.3).
- Since heat is required to decarbonise the limestone, use of burnt lime increases the sinter strand productivity, also by increasing the permeability.
- Environmental concerns have major local impact on sinter plant operation. Not only from the perspective of the use of "waste" materials, but also the emissions of a sinter plant may be a concern.

3.3.2 Sinter quality

As mentioned above, sinter is made in three different types: acid sinter, self-fluxing and super-fluxed sinter. The self-fluxing sinter is most common. Since sinter properties vary with the blend type and chemical composition, only some qualitative remarks can be made.

The sinter quality is defined by:

- Size distribution: sinter mean size ranges from 15 to 25 mm as measured after the sinter plant. The more basic the sinter, the smaller the average size. Sinter degrades during transport and handling so sinter has to be re-screened at the blast furnaces to remove all of the fines. Sinter from the stockyard may have very different properties from freshly produced sinter directly from the sinter plant. If stock sinter must be used in the blast furnace, it should be charged in a controlled fashion, and diluted with as much fresh sinter as is possible, e.g. by using a dedicated bin in the stockhouse to stock sinter.
- Cold strength: normally measured with a tumble test. The more fuel that is used, the stronger the sinter. The cold strength influences the sinter plant productivity because a low cold strength results in a high fines rate.

3 – The ore burden: sinter, pellets, lump ore

- Reduction-disintegration properties. The reduction from hematite to magnetite generates internal stresses within a sinter particle. The stronger the sinter, the better the resistance to these stresses. The reduction-disintegration properties improve with denser sinter structure, i.e. when the sinter is made with more coke breeze. As a consequence of the higher coke breeze usage, the FeO content of the sinter will increase. From experimental correlations it is well known that for a given sinter type reduction-disintegration improves with FeO content.
- The melting of sinter is determined by the chemical composition, that is, the local chemical composition. Most important is the basicity, the presence of FeO and SiO₂. The latter two functioning as components that lower the melting temperature. At temperatures of 1,200 to 1,250 °C sinter starts softening and melting. Very basic parts ($\text{CaO}/\text{SiO}_2 > 2$) melt at higher temperatures, but will still have melting temperatures around 1300 °C in the presence of sufficient FeO. If FeO is low, then melting temperatures exceeding 1,500 °C can be observed. However, melting in a blast furnace differs from melting of "pure" burden materials, since strong interactions between different burden components have been observed.

A summary of acceptability ranges for sinter quality parameters is given in Table 3.

Table 3: Characterisation of sinter and lump ore

	What is measured	Results	Acceptability range	Reference
Mean size	Size distribution	Average size mm < 3.15 mm	15–25 mm < 2%	ISO 4701
Cold strength	Size distribution after tumbling Japanese tumble	> 6.3 mm > 10 mm	> 70–80% > 52%	ISO 3271 JIS
Reduction-disintegration	Size distribution after reduction and tumbling (500°C, RDI=1, static)	< 3.15 mm	< 28 %	ISO 4696-1
Reducibility	Weight decrease during reduction	%/min	> 0.8%	ISO 4695
Decrepitation (lump ore)	Thermal shock effect (700°C)	< 6.3 mm	< 8%	ISO 8371

3 – The ore burden: sinter, pellets, lump ore

3.3.3 Background of sinter properties

Sinter is a very heterogeneous type of material. Research of various type of sinter in a cooled furnace has made clear, that various phases are simultaneously present. Figure 18. The most important phases present are:

- Primary and secondary magnetite (Fe_3O_4). Secondary magnetite is formed during sintering in the high temperature, reducing areas at the sinter strand, i.e. close to coke.
- Primary and secondary hematite (Fe_2O_3). Secondary hematite is formed on the sinter strand during the cooling down of the sinter in the presence of air (oxygen).
- Calcium ferrites are structures formed from (burnt) lime (CaO) and iron oxides.

It is clear from Figure 18, that at increasing basicity an increased fraction of calcium ferrites can be found. This has major consequences, for the sintering process as well as for the use of sinter in the blast furnace.

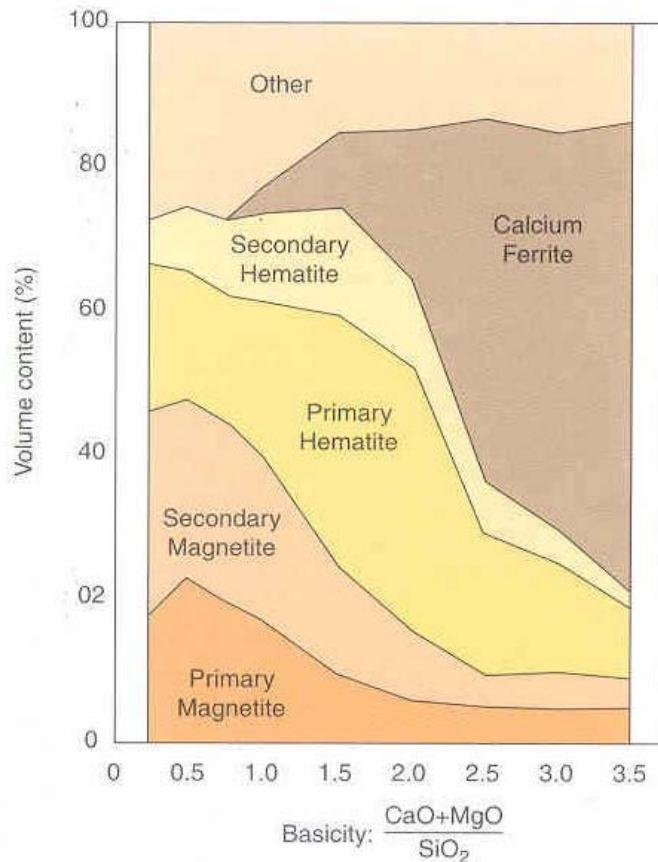


Figure 18: Phase composition of sinters (after Grebe et al, 1980)

3 – The ore burden: sinter, pellets, lump ore

First, let us consider the liquidus temperatures of sinter-type materials. The acid types of materials have much higher liquidus temperature than basic sinter. This is due to the fact, that calcium ferrite type structures have liquidus temperatures as low as 1,200°C (Figure 19), while the acid sinter have liquidus temperatures well above 1,400°C. It means also, that sintering of fluxed or superfluxed sinter can be accomplished at lower temperatures than sintering of a more acid sinter blend. Because of this, acid sinter is generally coarser and has a higher cold strength than basic sinter.

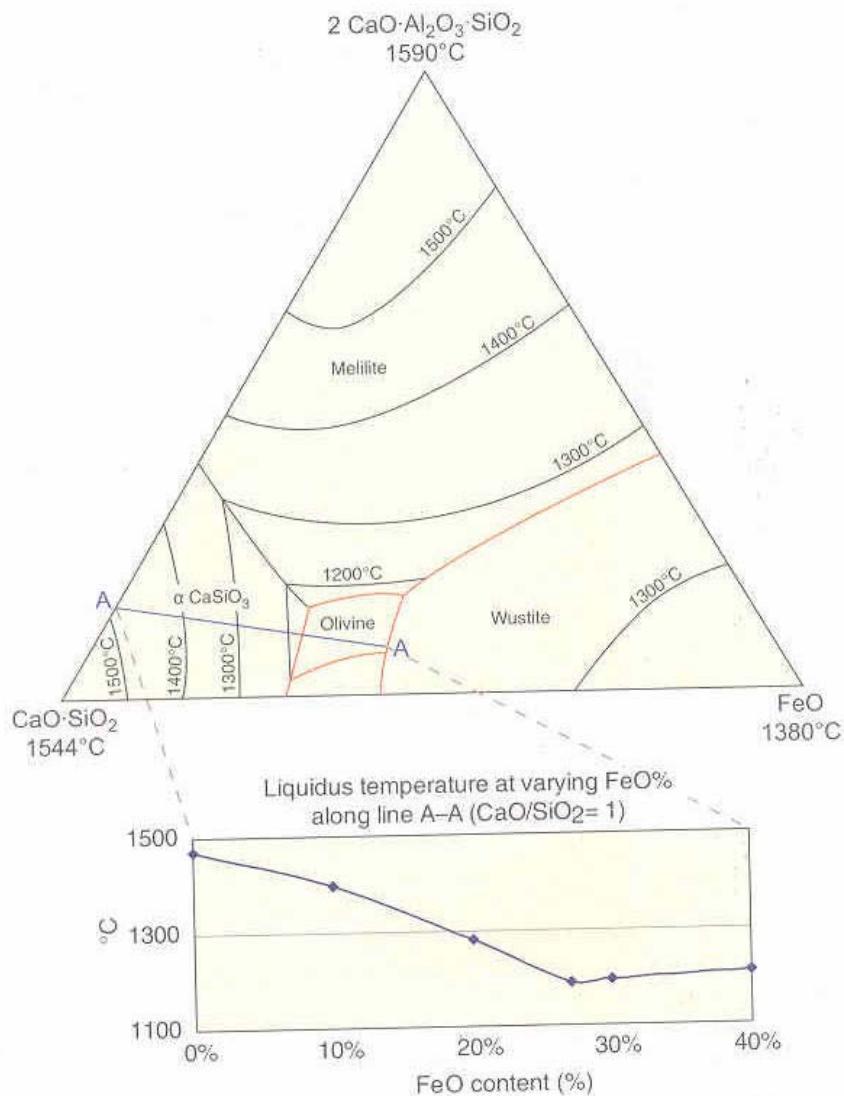


Figure 19 Example of liquidus temperatures of sinter type materials

3 – The ore burden: sinter, pellets, lump ore

Second, we consider the reduction-disintegration properties of the sinter. The driving force of low temperature reduction-disintegration of sinter is the changeover of the crystal structure from hematite to magnetite, which causes internal stress in the iron oxide crystal structure. So, reduction-disintegration of sinter is related to the fraction of hematite in the sinter. As shown in Figure 18, there are primary and secondary hematites in the sinter. Especially the latter causes reduction-disintegration, since it is more easily reduced in the upper part of the furnace than primary hematite.

The higher the hematite percentage in the sinter, the more the sinter is prone to reduction-disintegration. Or vice versa: there is strong relationship between the FeO content of the sinter and the reduction-disintegration: the higher the FeO content, the less reduction disintegration will take place. The FeO content of sinter can be increased by adding more fuel (coke breeze) to the sinter blend. However, the precise relationship between the FeO content of the sinter and the sinter quality depends on the ore blend used and is plant-specific. The reduction-disintegration properties depend on the type of FeO present in the crystal structure. As an example: a high fraction of magnetite in the sinter blend also gives a sinter with a high (primary) magnetite fraction. Moreover, in the presence of sufficient SiO₂ so-called fayalite structures ($2\text{FeO} \cdot \text{SiO}_2$) can be formed. These structures are very stable and are difficult to reduce. They are reduced at high temperatures i.e. by the so-called direct reduction reactions (see section 8.2.1). On the other hand, in the presence of MgO, spinel structures containing large amounts of FeO can be formed (spinel: $\text{MgO} \cdot \text{Al}_2\text{O}_3$ with FeO). The spinel structures are relatively easy to reduce. Finally, sinter that has been formed at high temperatures (acid sinter), will contain glass-like structures and the FeO is relatively difficult to reduce.

It is possible to suppress the formation of secondary hematites by cooling the sinter with air with a low O₂ percentage (12 to 14%). This is done at Corus IJmuiden with a gas recycling system. As a result the FeO content of the sinter is relatively high, because less secondary hematite is formed. This has a major benefit for the reduction-disintegration properties of this type of sinter. In addition, the calorific value of the blast furnace top gas increases, since less oxygen has to be removed from the ore burden, which gives a clear financial advantage as well.

During the sintering process there is a major difference between the use of CaO and MgO as fluxes. Both materials are normally added as the carbonate (limestone: CaCO₃ or dolomite CaCO₃.MgCO₃). The carbonates are decomposed on the sinter strand which costs a lot of energy. However, the melts containing substantial amounts of lime CaO have low liquidus temperatures, i.e. as low as 1,100°C for mixtures of 20 to 27% CaO and iron oxides. For the melts containing MgO – the spinel structures mentioned above – the melting temperatures are much higher. Therefore, it is easier to form slag-bonds in the sinter with the help of CaO than with MgO. And generally: making sinter with CaO can be done at lower temperature. But sinter with high MgO is more resistant against reduction-disintegration.

3 – The ore burden: sinter, pellets, lump ore

For the final result of the produced sinter, it is important to note, that the sinter blend prior to sintering is rather inhomogeneous. It contains various types of material and locally there are widely varying compositions present. Ore particles can be as large as 5 mm, coke breeze up to 3 mm and limestone and dolomite up to 2.5 mm. All types of chemical compositions are present on the micro-scale, where the sintering takes place. Type of materials used, size distribution of the various materials, the blending of the sinter mix, the amount of slag-bonds forming materials in the blend as well as the amount of fuel used for the sintering all have their own draw-back on sinter quality. It makes optimisation of sinter-quality a plant-specific technological challenge.

In the above sections we stress the importance of reduction-disintegration of sinter. The lower the reduction-disintegration, the poorer the reducibility of the sinter. Needle-like structures of Ca-ferrites have a relatively open structure and are easily accessible for reduction gas in the blast furnace. In cold conditions the sinter is strong (i.e. good tumbler test results), the degradation during transportation is also good, but the relatively fast reduction in the blast furnace makes the sinter very prone to reduction-disintegration. More solid structures in the sinter have better properties in this respect. Reduction-disintegration leads to poorer permeability of the ore-burden layers in the furnace and impedes proper further reduction of the iron oxides in the blast furnace.

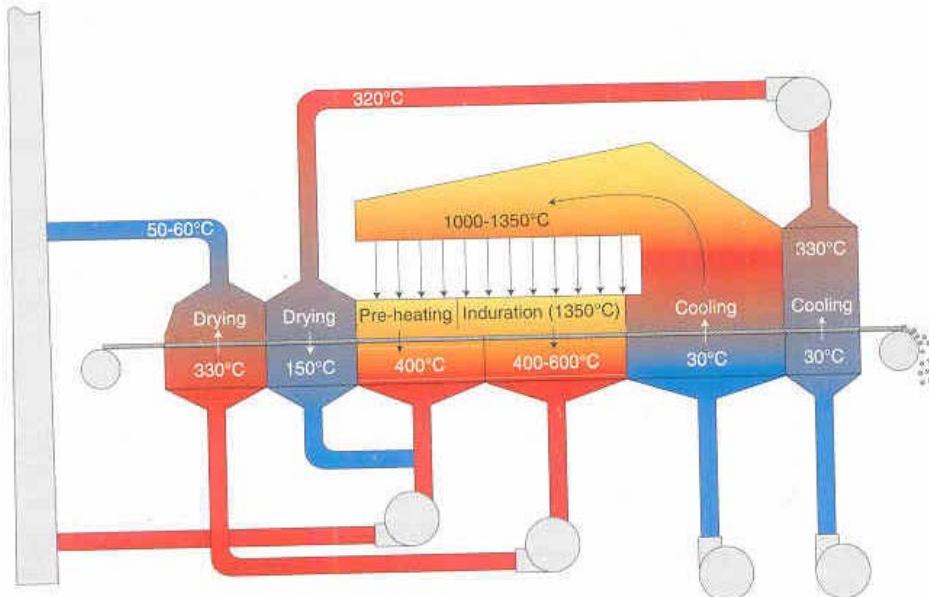


Figure 20: Pellet Plant

3.4 Pellets

3.4.1 Production of pellets

Production of pellets is a relatively new development and was driven by the increase in very fine ore fractions as a consequence of the ore enrichment methods used. Most pellet plants are located close to mines and a schematic of one such plant, a travelling grate pellet plant, is shown in Figure 20 (left). Another type of pellet plant uses rotary kiln.

Pellet production may be divided into four distinct stages in the travelling grate pellet plant.

3.4.1.1 Stage 1—Formation of green pellets

A blend of very fine ores is made, to which fine coke breeze (maximum 1%) may be added. The blend is pre-pelletised into 'green pellets' on rotating disks or in rotating drums. During this step, water and a binder are added to the blend. The adhesion of the particles is accomplished with the thin layer of moisture that is present on each ore particle forming a bridge with the thin layer of moisture on another particle. When the particle is rotated, more and more particles can adhere and the appropriate size fraction is selected by screening. The fines are re-circulated in the rotating drums or disks.

3.4.1.2 Stage 2—Drying

The green pellets are charged via a screen to a travelling grate. The green pellets have to be dried and the water bond must be replaced with a chemical bond. To accomplish this, the pellets are gradually heated to 300 to 350°C, which will allow the binding agent present in the mix, such as bentonite or an organic equivalent, to react.

3.4.1.3 Stage 3—Firing

The dry pellets are heated to a temperature of 1,250 to 1,350° C and the ore fines are fused together in a reaction known as induration. The required heat comes from burners as well as from coke breeze added to the original blend.

3.4.1.4 Stage 4—Cooling

The fired pellets have to cool down in a controlled way to prevent degradation of the pellets.

For the heat treatment of the pellets various alternatives are available. A moving bed, a shaft furnace, or a grate kiln system can be used.

3.4.2 Pellet quality

With correct chemical composition and induration, pellets remain generally intact in the blast furnace. Therefore, when judging pellet quality the main issues are:

- Cold strength, measured as compression strength and the fines generated through tumbling. A low figure for compression indicates bad or lean firing.
- Swelling properties. With incorrect slag composition pellets tend to have extreme swelling properties. Since the phenomenon is well known, it normally does not happen with commercially available pellets.
- The reduction-disintegration properties. These properties are less of a concern with pellets than with sinter and lump ore.
- Softening and melting. Pellets tend to melt at lower temperatures than sinter.
- Moisture level. Moisture in the pellets can interfere with the process. Excessive moisture can arise during stocking of pellets.

The slag volume, composition and bonding forces mainly determine the quality of pellets. The three main pellet types are:

- Acid pellets.
- Basic and fluxed pellets.
- Olivine fluxed pellets.

Typical properties of the three types of pellets are shown in Tables 4 and 5.

Table 4: Overview pellet properties

Pellet Type	Compression	Reducibility	Swelling
Acid	++	-	-
Basic/ fluxed	+	+	+
Olivine	+	+	+

Table 5: Typical analysis and compression of various types of pellets

Type	Basicity CaO/SiO_2	Fe%	$\text{SiO}_2\%$	MgO%	Compression strength kg/pellet
Acid pellets	< 0.15	64%–67%	2–5%	<0.2%	250–300
Olivine pellets	< 0.15	64%–67%	2.5–5%	1.3–1.8%	150–200
Basic pellets	~ 0.8–1.0	60%–64%	3.5%–5.5%	1.3%	200–250
Fluxed pellets	~ 1.1–1.3	60%–63%	3.5–5.5%	1.5%	200–250

Acid pellets are strong, but have moderate metallurgical properties. They have good compression strength (over 250 kg/pellet), but relatively poor reducibility. In addition the acid pellets are very sensitive to the CaO content. At $\text{CaO}/\text{SiO}_2 > 0.3$ the pellets have a strong tendency to swell, which may jeopardise blast furnace operation.

3 – The ore burden: sinter, pellets, lump ore

Basic and fluxed pellets have good metallurgical properties for blast furnace operation. By adding limestone to the pellet blend, the energy requirement of the firing/induration increases because of the decarbonisation reaction. For this reason production capacity of a pellet plant can sometimes be 10 to 15 percent lower when producing basic pellets.

Olivine pellets have MgO instead of CaO, which is added to the blend as olivine sand or serpentine. The pellets are somewhat weaker when tested for cold compression strength.

The difference in compression strength might seem large. However, in the blast furnace the pellets are reduced and the difference diminishes during reduction. After the first reduction step to Fe_3O_4 , the cold compression strength drops to 45 to 50 kg for acid pellets and to 35 to 45 kg for olivine pellets and fluxed pellets. Therefore, the lower compression strength has no drawback for the blast furnace process.

The compression strength, especially the fraction with a low compression strength (under 60 kg/p), is a good indicator for the pelletising process: the more pellets collapse at low compression strength, the poorer the pellets have been fired. Therefore, pellet quality can be influenced by the production rate: the slower the grate is moving, the stronger the firing can be, so the induration period increases and the pellets are stronger.

A final conclusion on the “optimum” pellet properties has not yet been reached. A summary of acceptability ranges for pellet quality parameters is given in Table 6.

Table 6: Characterisation of pellets

	What is measured	Results	Acceptability range	Reference
Mean size	Size distribution	8–16 mm < 6.3 mm	> 90% < 5%	ISO 4701
Cold strength	Compression strength Tumbler strength	Average kg/p *) % < 60 kg/p > 6.3 mm < 0.5 mm	> 150 **) < 4% > 95% < 5%	ISO 4700 ISO 3271 or ASTM
Low temperature disintegration	Size distribution after reduction and tumbling (dynamic)	Strength > 6.3 mm Abrasion < 0.5 mm	> 80% < 15%	ISO 13930
Reducibility	Weight decrease during reduction	%/min (dR/dt)40	> 0.8%	ISO 4695

*) kg/p corresponds with metric unit daN/p: decaNewton per pellet (=0.98 kg/p)

**) depends on pellet type. See Table 6.

3 – The ore burden: sinter, pellets, lump ore

3.5 Lump ore

Lump ores are naturally iron-rich materials that are used directly from the mines. The mines generally produce lump ore as well as (sinter) fines. Major lump ore deposits are present in Australia (Hamersley, Mount Newman), South America (Carajas, Robe River) and South Africa (Sishen). In many other places limited amounts of lump ores are produced.

Lump ores are cheaper than pellets and so in many blast furnaces high amounts of lump ore are being considered. The lower cost of the lump ore compared with pellets is offset by the poorer metallurgical properties. Generally speaking, in comparison with pellets, lump ore:

- generates more fines during transport and handling.
- has poorer reduction-disintegration properties.
- has a lower melting temperature.

The fines sometimes tend to accumulate in socalled "piggy-back-fines".

Lump ore is used in an appropriate size fraction, e.g. 8 to 30 mm and the quality parameters for lump are similar to those for sinter.

For blast furnace operation at high productivity and high coal injection levels, lump ore is a poorer type of burden material, however, since lump ore is a natural material, properties can differ from type to type. Siderar blast furnace in Argentina, for example, operates successfully with a Brazilian lump up to 40% of the burden at high furnace productivity.

Chapter 4

Coke

The blast furnace is charged with alternating layers of coke and ore burden. The coke forms a structure through which the gas can distribute itself and penetrate into the ore layers.

Since the 18th century, coke has been the most important source of carbon for the blast furnace process. From 1960 to now more and more auxiliary reductants are used, such as oil, tar, coal and natural gas. The auxiliary reductants are injected through the tuyeres and in doing so the flame temperature drops, so to compensate for this oxygen enrichment of the blast is often used simultaneously with injection in the tuyeres.

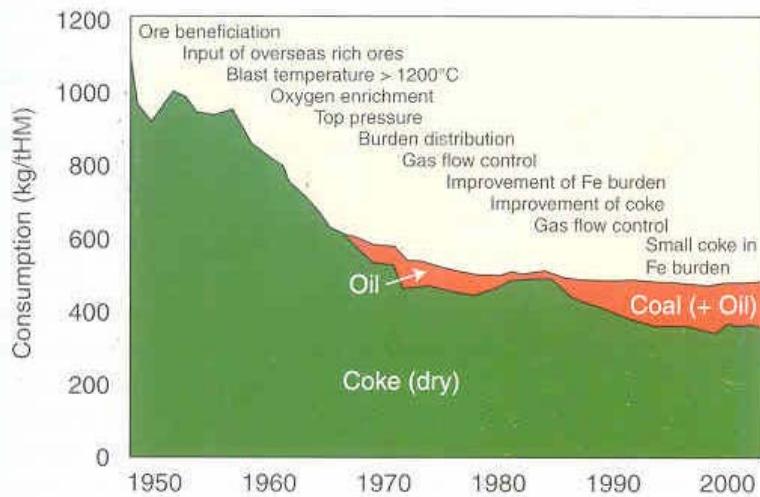


Figure 21: Coke rate decrease (Courtesy VDEh)

Over the last 50 years, the coke rate of the blast furnace process has considerably decreased, as illustrated in Figure 21. This is due to the application of auxiliary reductant injection, and also factors such as improved burden quality, higher blast temperatures,

bigger furnaces and improved process control. Presently, the reductant rate of larger blast furnaces is well below 500 kg/tHM coke equivalent, of which 30 to 40% is injected as coal. Coke rates slightly above or below 300 kg/tHM have been reached and can be considered normal operational practice.

4.1 Functions of coke

The most important functions of coke are:

- Coke is consumed and generates the heat to melt the burden.
- Coke provides the structure through which gas can ascend and be distributed through the burden. Coke is a solid and permeable material up to very high temperatures (over 2,000°C), which is of particular importance in the hearth and melting and softening zone. In the area below the melting zone coke is the only solid material remaining. The weight of the blast furnace contents is supported by the coke structure. The coke bed has to be permeable, so that iron and slag can accumulate in the hearth and flow to the taphole.
- Coke provides the carbon used for carburisation of the hot metal.

4.2 Coke quality

The coke quality can be described in two broad categories of its composition and its mechanical cold and hot strength. The relevant aspects of the composition are the ash and moisture content. Both should be as low as possible. Ash contents generally are 8–11%, but depend on the local situation. Moisture is a consequence of the quenching and subsequent handling of the coke. Further aspects of the chemical composition of the coke are the sulphur and alkali content, which is discussed later. The physical quality of the coke is characterised by:

- Size and size distribution. Average mean size of metallurgical coke is typically in the range of 45 to 55 mm. The size distribution should be tight to allow for a good permeability of the coke. A large fraction of coke over 80 mm is an indication of poor control of the coking process.
- The resistance to physical degradation due to transport and other mechanical stress. The parameters used for this property are: I40, M40, and stability. These parameters indicate the size distribution after stabilisation of coke.
- The resistance against abrasion, which is characterised by a test result called I10 and M10 and hardness.
- The coke reactivity. The coke can react with CO₂, which attacks the coke matrix and leads to higher coke rate in the blast furnace. Coke with a lower reactivity (CRI, coke reactivity index) and higher strength after reaction (CSR, coke strength after reaction) has higher mechanical strength in the lower part of the furnace.
- The day-to-day consistency of coke quality, especially size and size distribution, is often a neglected area in blast furnace operation. Inconsistency in coke quality leads to differences in burden distribution and may influence blast furnace internal gas flow.

4 – Coke

The various test results to characterise coke and the relevance for the process are indicated in Table 7 and in Figure 22.

Table 7: Acceptability range for coke quality parameters

	What is measured	Results	Acceptability Range	Best	Reference
Mean size	Size distribution	Average size mm % < 40 mm % < 10 mm	45–60 < 25% < 2%		
Cold strength	Size distribution after tumbling	I40 % > 40 mm I20 % > 20 mm I10 % < 10 mm M40 % > 40 mm M10 % < 10mm % > 1"	> 45 >78 < 19 > 80 < 7 > 58	55 16 87 5.5	Irsid test Micum test
	Stability at wharf Stability at stockhouse Hardness	% > 1" % > 1/4"	> 60 > 70		ASTM test
Strength after reaction	CSR: reaction(CO_2) + tumbling	% > 10 mm	> 58	70	
Reactivity	CRI: % weight loss during test	%	< 29%	24	

IRSID, MICUM and ASTM tests differ in their sample preparation. IRSID test takes coke sample > 20mm. MICUM test takes coke sample > 63 mm. ASTM test takes coke sample 2"-3"

Reference: after Kolijn (2002), Vander et al (1996)

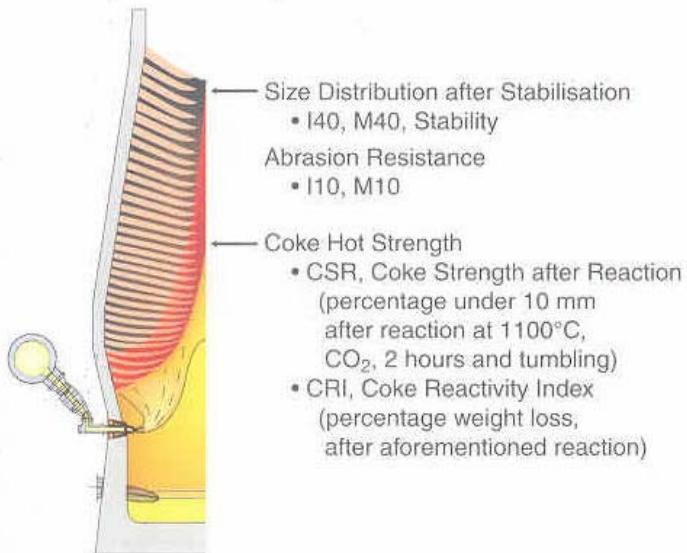


Figure 22: Coke tests and the blast furnace process

The coke quality is mainly determined by the coal blend used, although, coal preparation, carburisation time, equipment and quenching also influence coke quality. Usually a blend of different types of coal is used. Coking coals are coal types suitable for cokemaking and are priced higher than non-coking coals. Optimisation of the coal blend is an art in itself with a variety of important factors to be taken into account. These include the volatile matter of the coal and the gas pressure generated during carbonisation. The final coke should shrink sufficiently so that it can be pushed easily from the coke chamber, and should have low enough gas pressure not to damage the walls of the coke chamber. The carbonisation rate depends on the temperature of the coke battery, the faster the carbonisation rate the more fissures occur and the smaller the resulting coke. Generally, carbonisation times for slot-oven coke plants are in the range of 16–24 hours.

4.3 Coke production

The coal blend is heated to high temperatures ($1,200^{\circ}\text{C}$) in an air-free environment (Figure 23). As the temperature increases, volatile matter of the coal escapes and a solid carbon matrix is formed, termed the carbonisation of coal. A simplified coke production process is shown in Figure 24 (next page). The coal is heated from the wall to the centre, forming a so-called plasticity zone. As soon as the plasticity zones reach the centre, the coking process is completed.

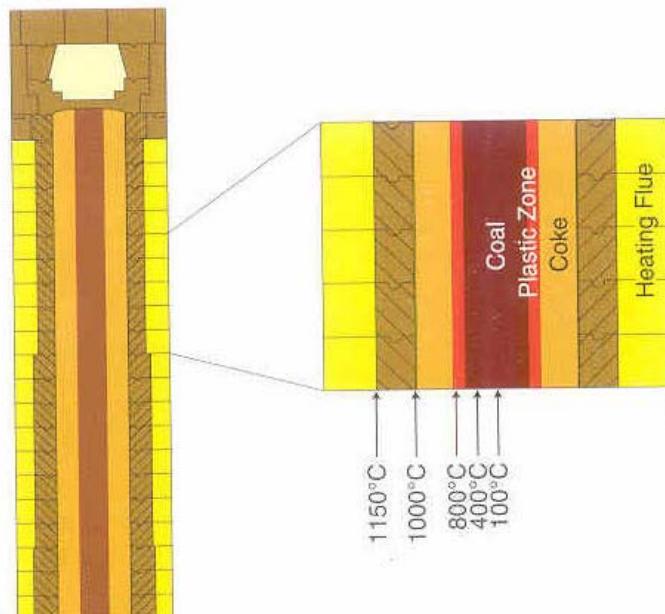


Figure 23: A coke chamber

4 – Coke

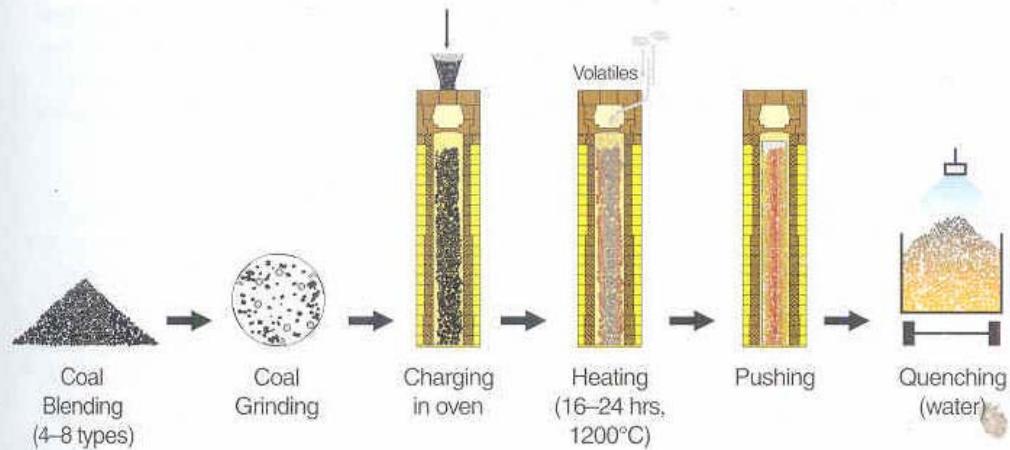


Figure 24: Coke production schedule

4.4 Coke quality in the blast furnace

Coke is subjected to mechanical and chemical attack during its descent from stockline through the stack and down into the hearth. The average size of coke decreases during its descent, as presented in a typical example of coke degradation in Figure 25. It was found that coke gets smaller over the route from coke plant to tuyeres while measurement after stabilisation (I_{40}) remained constant. The larger the coke charged to the furnace, the larger it arrives at the tuyeres.

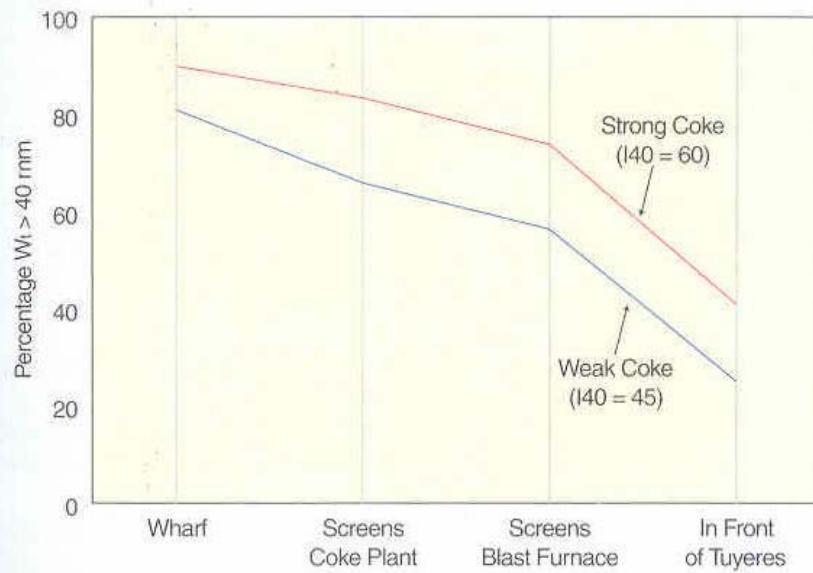


Figure 25: Coke degradation from wharf to tuyeres (after Van der Velden, 2001)

4 – Coke

The coke quality in the hearth has been the subject of much debate (e.g. Geerdes et al, 2001). The liquids must be cast from the hearth and permeability of the coke bed must be good. However, the reaction rate in the hearth is slow since coke is consumed only by direct reduction reactions and iron carburisation. For this reason the coke residence time in the hearth is long.

Wear of the hearth refractory by exposure to ring currents of iron along the hearth wall may be affected by coke quality – the smaller the void fraction, the stronger the ring currents. Due to the internal configuration of the furnace it is considered likely that the coke charged into the centre of the furnace is the coke that reaches the hearth. Some companies have put extra effort in the charging of large, strong coke into the centre of the furnace in an effort to maintain good dead man permeability.

Coke can react with carbondioxide ($C + CO_2 \rightarrow 2 CO$). The temperature, where this reaction starts, is in normal situations 1,100 to 1150 °C. However, the temperature depends especially on the presence of catalysts such as alkalis (Na, K) and on the ash composition of the coke: the more Fe in the coke ash, the higher the reactivity and the lower the temperature at which coke gasification by CO_2 starts. Higher coke reactivity leads to higher coke consumption per tonne hot metal.

Chapter 5

Injection of coal, oil and gas

Use of injection of pulverised (or granular) coal, oil and natural gas can lower the cost of hot metal. The auxiliary reductants are mainly coal, oil and natural gas, but tar and other materials can also be used. The precise financial balance depends very much on local situations.

Up until the early 1980's oil injection was a commonly used, however the changes in relative prices between coal and oil has resulted in coal becoming the more widely used injectant. Note, that the preparation of coal for injection involves a rather high investment cost. The pay-back of the investment heavily depends on the hot metal production level. Most major sites have been equipped with coal injection. When coke is scarce and expensive, the feasibility of coal injection for smaller sites increases. The most important arguments for the injection of coal (or natural gas) in a blast furnace are;

- Cost savings by lower coke rates. Cost of coke is substantially higher than that of coal, moreover, the use of an injectant allows higher blast temperatures to be used, which also leads to a lower coke rate.
- Increased productivity from using oxygen enriched blast.

The reason for the apparent versatility of the blast furnace in consuming all types of carbon containing materials is that at the tuyeres the flame temperatures are so high that all injected materials are converted to simple molecules like H₂ and CO and behind the raceway the furnace "does not know" what type of injectant was used.

Coal injection was applied in the blast furnace Amanda of ARMCO (Ashland, Kentucky) in the 60's. In the early days of coal injection, injection levels of 60–100 kg coal per tonne hot metal were common. Presently, the industrial standard is to reach a coke rate of 300 kg/t with injection levels of 200 kg coal per tonne hot metal (Toxopeus et al, 2001).

5 – Injection of coal, oil and gas

5.1 Coal injection: equipment

The basic design for coal injection installations requires the following functions to be carried out:

- Grinding of the coal. Coal has to be ground to very small sizes. Most commonly used is pulverised coal: around 60% of the coal is under 75 µm. Granular coal is somewhat coarser with sizes up to 1 to 2 mm.
- Drying of the coal. Coal contains substantial amounts of moisture, 8% to more than 10%. Since injection of moisture increases the reductant rate, moisture should be removed as much as possible.
- Transportation of the coal through the pipelines. If the coal is too small the pneumatic transport will be hampered. It may result in formation of minor scabs on the walls and also lead to coal leakage from the transportation pipes.
- Injection of the pulverised coal: Coal has to be injected in equal amounts through all the tuyeres. Particularly at low coke rate and high productivity the circumferential symmetry of the injection should be maintained.

There are various suppliers available for pulverised coal injection (PCI) installations, which undertake the functions mentioned above in a specific way. The reliability of the equipment is of utmost importance, since a blast furnace has to be stopped within one hour, if the coal injection stops.

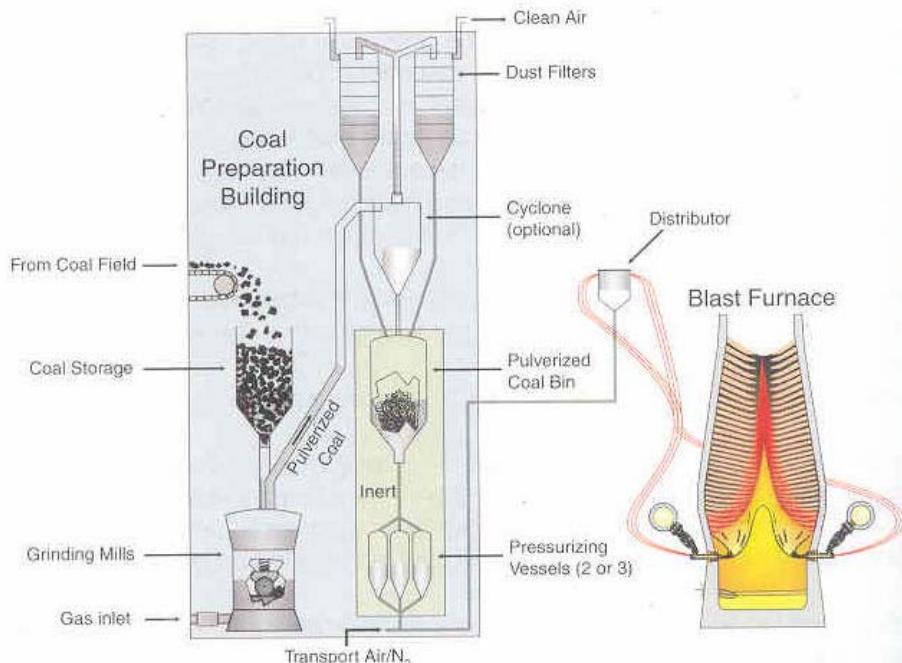


Figure 26: Example of PCI installation

5 – Injection of coal, oil and gas

5.2 Coal specification for PCI

Coal types are discriminated according to their volatile matter content. The volatile matter is determined by weighing coal before and after heating for three minutes at 900°C. Coals that have between 6 and 12% volatile matter are classified as low volatile, those between 12 and 30% are mid volatile and anything over 30% are high volatile. All types of coal have successfully been used.

The most important properties of the injection coals are:

- High replacement ratio (RR) of coke. The composition and moisture content of the coals determine the amount of coke replaced by a certain type of coal. A simple formula for the replacement ratio (compared with coke with 87.5% carbon) is:

$$RR = 2x C\%(coal) + 2.5x H\%(coal) - 2x \text{ moisture}\%(coal) - 86 + 0.9x \text{ ash}\%(coal)$$

- Composition: high sulphur and high phosphorous are likely to increase costs in the steel plant. These elements should be evaluated prior to the purchase of a certain type of coal. Young coals (high oxygen content) are known to be more susceptible to self-heating and ignition in atmospheres containing oxygen. This is also an important factor that must be considered with regard to the limitations of the ground coal handling system.
- Volatile matter: high volatile coals are easily gasified in the raceway, but have lower replacement ratio in the process.
- Hardness. The hardness of the coal, characterised by the Hardgrove Grindability Index (HGI) must correspond to the specifications of the grinding equipment. The resulting size of the ground coal is also strongly dependent on this parameter and must correspond to the limits of the coal handling and injection system.
- Moisture content. The moisture content of the raw coal as well as the surface moisture in the ground coal must be considered. Surface moisture in the ground coal will lead to sticking and handling problems.

Potential injection coals can be evaluated on the basis of "value in use", where all effects on cost are taken into account. It is often possible to use blends of two or three types of injection coals, so that unfavourable properties can be diluted.

5.3 Coal injection in the tuyeres

Coals are injected via lances into the blowpipes, ignited and gasified. The coal is in the raceway area only for a very short time (5 milliseconds) and so the characteristics of the gasification reaction are very important for the effectiveness of a PCI system. Coal gasification consists of several steps as outlined in Figure 27 (next page). Firstly the coal is heated and the moisture evaporates. Gasification of the volatile components then occurs after further heating. The volatile components then ignite and are gasified, which causes an increase in the temperature, and then finally the coal char is gasified. All of these steps occur sequentially with some overlap.

5 – Injection of coal, oil and gas

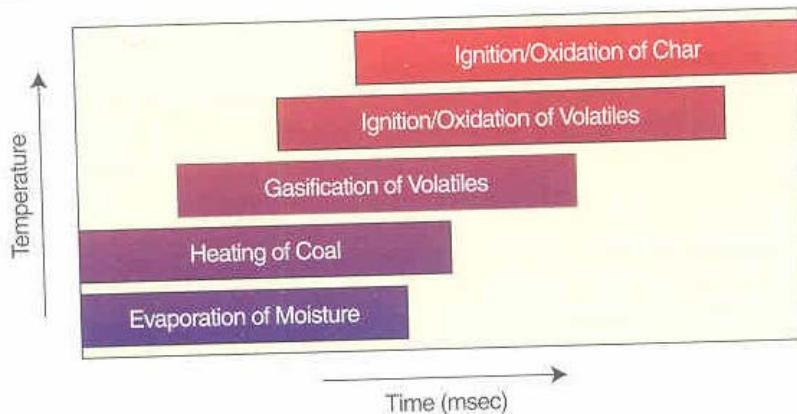


Figure 27: Coal gasification

The effects of lance design, extra oxygen and coal type on the coal combustion have been analysed. Originally, the coal lances were straight steel lances that were positioned at or close to the tuyere/blowpipe interface as indicated in Figure 28. Generally, the simple arrangement works well, although, occasionally "char" (unburnt degasified coal) or "soot" (very fine carbon formed from gas) are detected as they leave the furnace through the top. To avoid this problem, especially at high injection rates, companies have installed different types of injection systems at the tuyeres, such as:

- Coaxial lances with oxygen flow and coal flow.
- Specially designed lances with a special tip to get more turbulence at the lance tip.
- Use of two lances per tuyere.
- Coal preheating.

When using PCI, deposits of coal ash are occasionally found at the lance tip or within the tuyere. The deposits can be removed by periodic purging of the lance by switching off the coal while maintaining air (or nitrogen) flow.

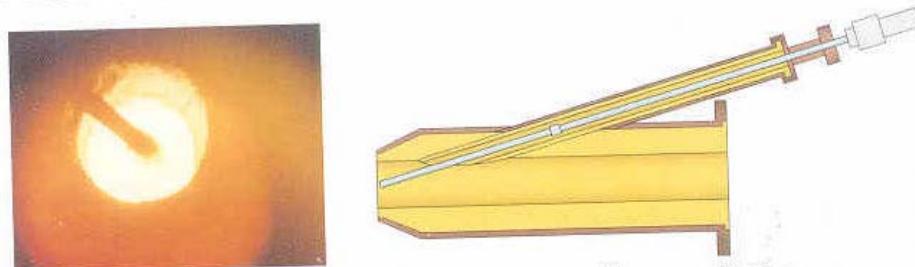


Figure 28: Coal injection in the tuyeres and lance positioning

The speed of gasification increases as;

- The volatility of the coals increases.
- The size of the coal particles decreases.
- The blast and coal are mixed better.

Moreover, as the injection level increases, the amount of coal that leaves the raceway without being gasified increases.

5 – Injection of coal, oil and gas

The gasification of coal also depends on the percentage of volatile matter (VM). If low volatile coals are used, a relatively high percentage of the coal is not gasified in the raceway and is transported with the gas to the active coke zone. This "char" will normally be used in the process, but might affect the gas distribution. The high volatile (HV, over 30% VM) and ultra high volatile coal (over 40% VM) produces a large quantity of gas in the raceway and a small quantity of char. If the gas combustion is not complete, soot can be formed. Blending a variety of injection coals, especially high volatile and low volatile coals, gives the advantage of being able to control these effects.

It has been found that the coke at the border between raceway and dead man contains more coke fines when working at (high) injection rates. This region has been termed the "bird's nest".

5.4 Process control with Pulverised Coal Injection

At high Pulverised Coal Injection operation about 40% of the reductant is injected via the tuyeres. Therefore, it is important to control the amount of coal per tonne hot metal as accurate as the coke rate is controlled. The feed tanks of the coal injection are weighed continuously and the flow rate of the coal is controlled. It can be done with nitrogen pressure in the feed tanks or a screw or rotating valve dosing system. In order to calculate a proper flow rate of coal (in kg/minute) the hot metal production has to be known. There are several ways to calculate the production. The production rate can be derived from the amount of material charged into the furnace. Short-term corrections can be made by calculating the oxygen consumption per tonne hot metal from the blast parameters in a stable period and then calculating the actual production from blast data. Systematic errors and/or the requirement for extra coal can be put in the control model.

The heat requirement of the lower furnace is a special topic when using PCI. Coal is not only used for producing the reduction gases, but use of coal has an effect on the heat balance in the lower furnace. The heat of the bosh gas has to be sufficient to melt the burden: define the "melting heat" as the heat needed to melt the burden. The heat requirement of the burden is determined by the "pre-reduction degree", or how much oxygen has still to be removed from the burden when melting. The removal of this oxygen requires a lot of energy. The "melting capacity" of the gas is defined as the heat available with the bosh gas at a temperature over 1,400°C. The melting capacity of the gas depends on:

- The quantity of tuyere gas available per tonne hot metal. Especially when using high volatile coal there is a high amount of H₂ in the bosh gas.
- The flame temperature in the raceway.

The flame temperature in itself is determined by coal rate, coal type, blast temperature, blast moisture and oxygen enrichment.

5 – Injection of coal, oil and gas

From the above, the oxygen percentage in the blast can be used to balance the heat requirements of the upper and lower furnace. The balance is dependent on the local situation. It depends e.g. on burden and coke quality and coal type used. For the balance there are some technical and technological limitations, which are presented as an example in Figure 29. For higher injection rates more oxygen is required. The limitations are given by:

- Too low top gas temperature. If top gas temperature becomes too low it takes too long for the burden to dry and the effective height of the blast furnace shortens.
- Too high flame temperature. If flame temperature becomes too high burden descent can become erratic.
- Too low flame temperature. Low flame temperature will hamper coal gasification and melting of the ore burden.
- Technical limitations to the allowed or available oxygen enrichment.

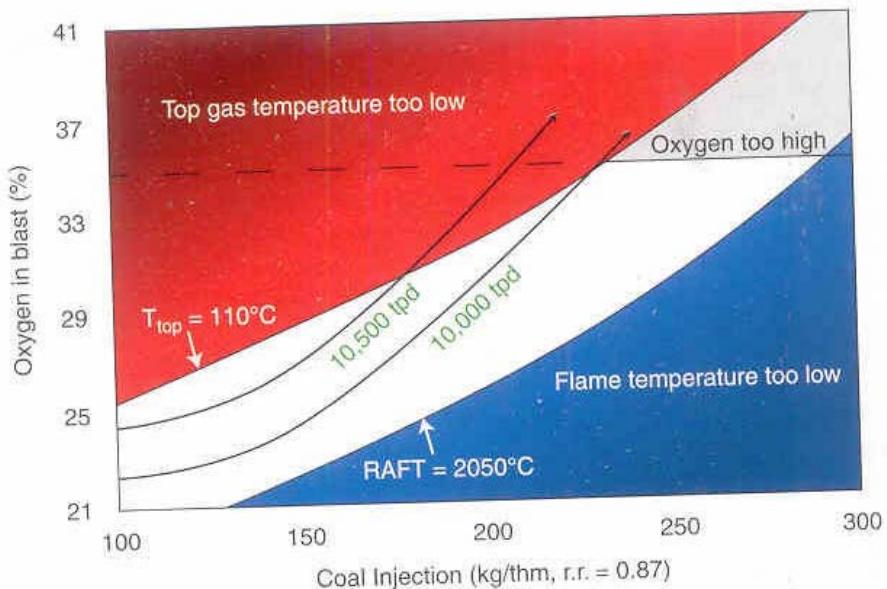


Figure 29: Limiting factors affecting raceway conditions with Pulverised Coal Injection
(RAFT = Raceway Adiabatic Flame Temperature)

Note especially the effect of the use of extra coal injection for recovery of a cooling furnace. By putting extra coal on the furnace the production rate decreases by about 2.5% for every 10 kg extra coal per tonne hot metal. Simultaneously, the flame temperature drops by about 32°C. Therefore, depending on the specific situation, the melting capacity per tonne hot metal may even decrease. If the flame temperature drops from 2,050 to 2,018°C, the melting capacity decreases by 5% ($32/2,050 \times 1,400$) and the production decreases by 2.5% resulting in a decrease of melting heat per tonne hot metal by 2.5%. If the chilling furnace has insufficient melting capacity of the gas, extra PCI may worsen the situation. In such a situation the efficiency of the process must be improved, i.e. by a lower production rate and lower blast volume.

5 – Injection of coal, oil and gas

5.5 Gas flow control

For optimised blast furnace process the control of the blast furnace internal gas flow is critical. A detailed discussion is presented in Chapter 6. Generally speaking, it has been found that the balance between central gas flow and wall gas flow has to be maintained. In the Corus IJmuiden works the presence of an ore-free centre is used in order to distribute the bosh gas through the coke slits (Exter et al, 1997). As a consequence the root of the melting zone can descend quite close to the tuyeres. The associated risk of a low root of the melting zone is damage to the tuyeres (leakage and/or tipping).

High PCI requires a better circumferential symmetry of the process, not only from the burdening, but also from the blast and coal injection distribution through the tuyeres.

5.6 Circumferential symmetry of injection

If every tuyere in a blast furnace is considered as part of the blast furnace pie and is responsible for the process to the stock-line, it is self evident that the circumferential symmetry of the process has to be assured to reach good, high performance. The various systems in use for PCI have different methods to ensure a good distribution.

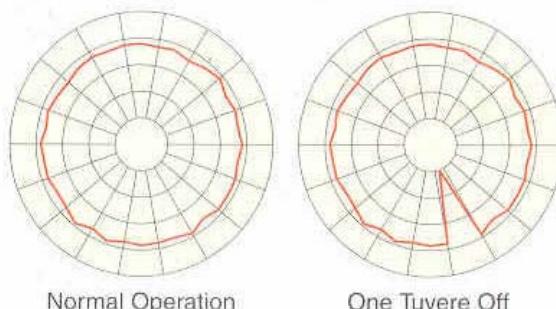


Figure 30: Schematic presentation of the effect of no PCI on one tuyere

Table 8: Coke use per tuyere in case a single tuyere receives no coal

PCI at all tuyeres	PCI at one tuyere off
• Coke rate 300 kg coke/t	• Tuyere uses 3,300 kg/hr, coke only
• PCI 200 (R.R. = 0.85) 170 kg coke/t	• Production increase at this tuyere without PCI of 3,300/1,600 = 205%
• Total 470 kg coke/t	
• Production 10 t/hr per tuyere	
• Carbon balance per tuyere:	
• Coke 3,000 kg/hr	
• Coal 1,700 kg/hr	
• Total 4,700 kg/hr	
• Iron carburisation -500 kg/hr	
• To direct reduction* -900 kg/hr	
• Used at tuyeres 3,300 kg/hr	
• of which coal 1,700 kg/hr	
• of which coke 1,600 kg/hr	

5 – Injection of coal, oil and gas

However, the largest deviation from circumferential symmetry occurs when no coal is injected in a particular tuyere. If no injection is applied, the production rate at that particular tuyere doubles. Consequentially, the blast furnace operator has to take care that all tuyeres are injecting coal. In particular, where two tuyeres next to each other are not injecting coal the equalising effects between the tuyeres are challenged. Especially if the furnace is operating at high PCI rates, the situation is rather serious and short-term actions have to be taken to correct the situation.

This point can be illustrated from Table 8 and Figure 30 (previous page). The calculation shows, how much coke is consumed in front of a tuyere, where coal injection is switched off. At high injection rates, the production can increase twofold. Note, that this is an example, since in such a situation neighbouring tuyeres will tend to contribute. Moreover, the calculation does not take the oxygen of the coal itself into account.

5.7 Other injectants

As stated earlier, all types of (hydrocarbon) injectants can be used. A comparison of replacement ratio, typical chemical composition and effect on flame temperature are given in Table 9.

Table 9: Typical data for injectants

Injectant	Replacement Ratio *)	C %	H %	Moisture %	Effect flame temperature °C **)
Coal	0.80	78–82	4–5.5	1–4	–32
Oil	1.17	87	11	2	–37
Natural gas	1.05	57	19	–	–45
Tar	1.0	87	6	2	–25

*) Compared with standard coke with 87.5% C

**) When injecting additional 10 kg/tHM

Chapter 6

Burden calculation and mass balances

6.1 Introduction

The blast furnace is charged with pellets, sinter, lump ore and coke, while additional reductant might be injected through the tuyeres. The steel plant requires a defined quality of hot metal and the slag has to be chosen for optimum properties with respect to fluidity, desulphurising capacity and so on. Therefore, the blast furnace operator has to make calculations to select the blast furnace burden. The present chapter first indicates the conditions for a burden calculation, which is then illustrated with a practical example. Later in the chapter the burden calculation is taken a step further to indicate the process results. To this end a simple one-stage mass balance is used.

6.2 Burden calculation: starting points

Starting points for burden calculations are the hot metal and slag quality.

- Hot metal quality: silicon, typically 0.4 to 0.5%. Low sulphur (under 0.03%) and defined phosphorous levels, which vary due to variation in burden materials from 0.05% to 0.13%.
- Slag quality: generally the lower the slag volume the better. Typically the four major constituents of slag contain about 96% of the total volume: Al_2O_3 (8 to 20%), MgO (6 to 12%), SiO_2 (28 to 38%) and CaO (34 to 42%). For slag design, see chapter 10.

The burden has to fulfil requirements with respect to:

- Maximum phosphorous input, since phosphorous leaves the furnace with the iron.
- Maximum alkali input, especially potassium, which can attack the refractory and affect the process. Typically a limit of 1 to 1.5 kg/tHM is used.
- Maximum zinc input: zinc can condense in the furnace and can, similar to alkali, lead to a Zn cycle. Typically, limits for zinc input are 100–150 g/tHM. With high central gas temperatures, zinc and alkali are partly removed with the top gas.

Burden calculation and mass balances

6.3 An example of a burden calculation

The burden calculation uses the chemical composition (on a dry basis) and the weights of the various materials in a charge as input parameters. A charge consists of a layer of burden material and coke with its auxiliary reductants as injected through the tuyeres. In order to be able to do the calculation, the yield losses when charging the furnace are also taken into account. The present example is restricted to the components required to calculate the slag composition. The four main components (SiO_2 , CaO , MgO and Al_2O_3) represent 96% of the total slag volume. The other 4% consist of MnO , S , K_2O , P and many more. The losses from the materials charged through the top into the blast furnace are taken into account and are normally based on samples of material from the dust catcher and scrubber systems. The calculation is presented in Table 10.

Table 10: Simplified burden calculation

Chemical analysis								
	Ash	Moisture	Loss	Fe	SiO_2	CaO	MgO	Al_2O_3
Coke	9%	5%	2%	0.5%	5.0%			3.0%
Coal	6%	1%	0%	0.2%	3.0%			1.5%
Sinter		1%	1%	58%	4.0%	8.3%	1.4%	0.6%
Pellets		1%	1%	65%	3.5%		1.3%	0.8%
Lump		3%	1%	61%	4.0%			1.0%
	Weight	After losses						
Burden	kg/tHM	kg/tHM		Input in kg/tHM				
Coke	300	294		1	15	0	0	9
Coal	200	200		0	6	0	0	3
Sinter	1,000	990		575	40	82	14	6
Pellets	500	495		322	17	0	6	4
Lump	80	79		49	3	0	0	1
Total	1580			947	81	82	20	23
Correction HM silicon 0.46% = 10 kg SiO_2 /tHM					-10			
Slag					71	82	20	23
Results								
Slag volume *)			kg/tHM	204	SiO_2	CaO	MgO	Al_2O_3
Slag composition					35%	40%	10%	11%
Basicity	CaO/SiO_2			1.16				
	$(\text{CaO}+\text{MgO})/\text{SiO}_2$			1.45				
	$(\text{CaO}+\text{MgO})/(\text{SiO}_2+\text{Al}_2\text{O}_3)$			1.10				
	Al_2O_3			11%				
	Ore/coke ratio			5.3				

*) $1.04 \times (\text{SiO}_2 + \text{CaO} + \text{MgO} + \text{Al}_2\text{O}_3)$

Burden calculation and mass balances

6.4 Process calculations: a simplified mass balance

The calculations of the previous section can be extended to include the blast into the furnace. In doing so the output of the furnace can be calculated: not only the hot metal and slag composition and the reductant rate, but the composition of the top gas as well. Calculation of the top gas composition is done in a stepwise manner in which the balances of the gas components (nitrogen, hydrogen, oxygen, CO and CO₂) and iron and carbon are made. For the calculations the example of a 10,000 t/d furnace is used. The stepwise approach indicated in Table 11.

Table 11: Stepwise approach for a simplified mass balance

Input Element	Nitrogen (N ₂)	Hydrogen (H ₂)	Iron (Fe)	Carbon (C)	Oxygen (O ₂)
Main Sources	Blast	Injection Blast Moisture	Burden	Coke Injection	Burden (52%) Blast (48%)
What to know	N ₂ % in blast	H % in reductant	%Fe ore burden	%C in coke and injectant	%O ₂ wind
Main output via	Top gas	Top gas	Hot metal	Top gas (85%) Hot metal (15%)	Top Gas • CO (32%) • CO ₂ (64%) • H ₂ O (4%)
What to know	N ₂ % in top gas	H ₂ efficiency	Hot metal composition	Rates per tonne Composition	
Calculation of	Top gas volume	H ₂ % in top gas	Oxygen input via burden	Top gas composition CO & CO ₂ %	

The approach is as follows:

Step 1: nitrogen balance: from the nitrogen balance the total top gas volume is estimated.

Step 2: hydrogen balance: from hydrogen input and a hydrogen utilisation of 40% the top gas hydrogen can be estimated. In practice hydrogen utilisations of 38–42% are found.

Step 3: iron and carbon balance: the carbon use per tonne is known from the hot metal chemical composition and coke and coal use per tonne.

Step 4: oxygen balance: the burden composition gives the amount of oxygen per tonne hot metal input at the top, while also the amount of oxygen with the blast is also known per tonne hot metal.

Step 5: the balances can be combined to calculate the top gas composition.

The calculations are based on basic chemical calculations. Starting points for the calculations are, that:

- 12 kilogram of carbon is a defined number of carbon atoms defined as a kilomole.
- Every mole of an element or compound has a certain weight defined by the periodic table of the elements.

Burden calculation and mass balances

- 1 kmole of a gas at atmospheric pressure and 0°C occupies 22.4 m³ STP.
- The properties of the various components used for the calculations are indicated in Table 12. The present balance is used for educational purposes figures and compositions are rounded numbers. Effects of moisture in pulverised coal and the argon in the blast are neglected.

Table 12: Properties of materials used for mass balance calculations

	Atomic weight		Molecular weight	
N ₂	28	kg/kmole	CO	28 kg/kmole
O ₂	32	kg/kmole	CO ₂	44 kg/kmole
H ₂	2	kg/kmole		
C	12	kg/kmole		
Fe	55.6	kg/kmole		
Si	28	kg/kmole		

1 kmol gas (N₂, O₂, etc) = 22.4 m³ STP
1 tonne hot metal contains 945 kg Fe = 945/55.6 = 17.0 kmole

6.4.1 The Nitrogen Balance

Nitrogen does not react in the blast furnace, so it escapes unchanged via the top gas. At steady state the input equals the output and the top gas volume can be calculated with a nitrogen balance given the nitrogen input and the nitrogen concentration in the top gas. The input data for a simplified model are shown in Table 13 and the top gas volume is calculated in Table 14.

Table 13: Mass balance input

Blast volume	6500	m ³ STP/min	
Oxygen in blast	25.6	%	
Moisture	10	g/m ³ STP	
Production	6.9	tHM/min	
Coal rate	200	kg/tHM	
Coke rate	300	kg/tHM	
Top gas			
CO ₂	22	vol%	
CO	25	vol%	
H ₂	4.5	vol%	
N ₂	48.5	vol%	
	C%	H%	O%
Coal	78	4.5	7
Coke	87	0.2	1.4

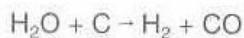
Burden calculation and mass balances

Table 14: The nitrogen balance and top gas volume

Nitrogen from blast	$(1-0.256)x$ 6500	4836	m ³ STP/min
From coal		16	m ³ STP/min
From coke		23	m ³ STP/min
Total input		4875	m ³ STP/min
Top gas nitrogen		48.5	%
Top gas volume		10051	m ³ STP/min

6.4.2 The hydrogen balance

Moisture in the blast and coal reacts to H₂ and CO according to:



All hydrogen in coal and coke are converted to H₂ in the furnace. In the furnace the H₂ is reacting to H₂O; part of the hydrogen is utilised again.

Since the top gas volume is known as well as the hydrogen input, the top gas hydrogen can be calculated, if a utilisation of 40% is assumed. There are ways to check the hydrogen utilisation, but it is beyond the scope of the present exercise. Table 15 shows the input and calculates the top gas hydrogen.

Table 15: The hydrogen balance

	kg/min	in m ³ STP/ min
From blast	7	
From coal	56	
From coke	4	
Total input	67	750
Utilisation 40%, so		
Left in top gas 60%		450
Top gas volume		10051
H ₂ in top gas		4.5%

6.4.3 The iron and carbon balance

Hot metal contains 945 kg Fe per tonne. The balance is taken by carbon (45 kg), silicon, manganese, sulphur, phosphorous, titanium and so on. The precise Fe content of hot metal depends slightly on the thermal state of the furnace and quality of the input. For the balance we use 945 kg Fe/tHM. This amounts to 17 kmole (947/55.6).

Burden calculation and mass balances

The carbon balance is more complicated. The carbon is consumed in front of the tuyeres and is used during the direct reduction reaction (see section 8.2.1). The carbon leaves the furnace via the top gas and with the iron. The carbon balance is made per tonne hot metal. Table 16 shows the results. The carbon via the top gas is also given in katom per tonne hot metal.

Table 16: Carbon balance

Carbon used	In kg/tHM	Katom/tHM
Carbon from coke	281	
Carbon from coal	156	
Total carbon use	417	
Carbon via iron	-45	
Carbon via top gas	372	31.0

6.4.4 The oxygen balance

The oxygen balance is even more complicated. Oxygen is brought into the furnace with the blast, with PCI, with moisture and with the burden. It leaves the furnace through the top. The burden with sinter contains not only Fe_2O_3 (O/Fe ratio 1.5) but some Fe_3O_4 (O/Fe ratio 1.33) as well. The O/Fe ratio used here is 1.46, which means that for every atom of Fe there is 1.46 atom O. On a weight basis it means, that for every tonne hot metal, which contains 945 kg Fe atoms there is 397 kg O-atoms. All this oxygen leaves the furnace with the topgas. The balance is given in Table 17.

Table 17: The oxygen balance

		m ³ STP /tHM	kg O/tHM	Katom O/tHM
Input	From blast	240	342	
	From blast moisture		8	
	From coal		14	
	From burden		397	
Total input			762	
Output via top gas			762	47.6

6.4.5 Calculation of top gas analysis

The oxygen in the top gas is leaving the furnace in three different states:

- Bound to the hydrogen. The quantity is known since we know how much hydrogen has been converted to process water.
- Bound to carbon as CO.
- Bound to carbon as CO_2 .

Burden calculation and mass balances

From the combination of the carbon balance and the oxygen balance we can now derive the top gas utilisation, as shown in Table 18.

Table 18: Calculation of top gas utilisation

	Katom/tHM
Carbon via top gas	31.0
Oxygen via top gas	47.6
Oxygen bound to hydrogen	-1.9
Oxygen as CO and CO ₂	45.7

Oxygen balance: CO + 2x CO ₂	45.7
Carbon balance: CO + CO ₂	31.0
CO ₂	14.7
CO	16.3
Utilisation	CO ₂ / (CO+CO ₂)
CO ₂ volume	2283 m ³ STP/min
CO	2539 m ³ STP/min
	CO ₂ % 22.7%
	CO% 25.3%

The calculations can be used to check the correct input data. More advanced models are available, which take into account the heat balance of the chemical reactions as well (e.g. Rist and Meysson, 1966). The models are useful for analysis, especially questions like "are we producing efficiently?" and for prediction: what if PCI is increased? hot blast temperature is increased? and so on.

Chapter 7

The process: burden descent and gas flow control

7.1 Burden descent: where is voidage created?

The burden descends in the blast furnace from top to bottom. Figure 32 shows a representation of the burden descent. It is indicated with stock rods, which are resting on the burden surface and descending with the burden between charging. The burden surface descends with a speed of 8 to 15 cm/minute.

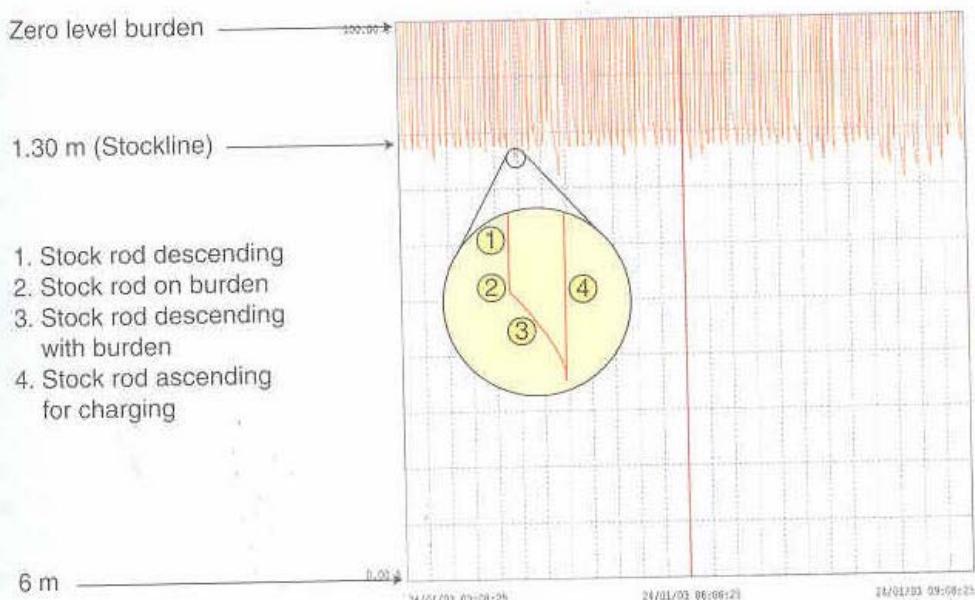


Figure 31: Stable burden descent

7 – The process: burden descent and gas flow control

In order for the burden to descend, voidage has to be created somewhere in the furnace. Where is this voidage created? See Figure 32.

- Firstly, coke is gasified in front of the tuyeres, thus creating voidage at the tuyeres.
- Secondly, the hot gas ascends up the furnace and melts the burden material. So the burden volume is disappearing into the melting zone.
- Thirdly, the dripping hot metal consumes carbon. It is used for carburisation of the iron as well as for the direct reduction reactions, so below the melting zone coke is consumed.

It is possible to indicate how much each of the three mechanisms contributes to the amount of voidage created. A large part of the voidage is created at the melting zone. In a typical blast furnace on high PCI, only about 25% of the voidage is created at the tuyeres.

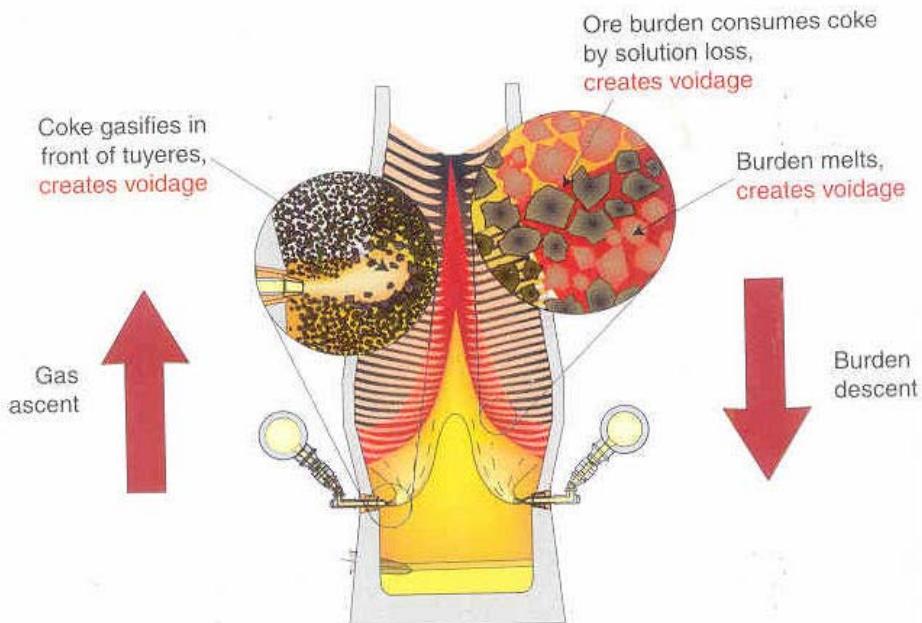


Figure 32: Creation of voidage in the Blast Furnace

This implies that the mass flow of material is strengthened towards the ring where the highest amount of ore is charged into the furnace. Therefore, at low coke rates high ore concentration at any ring in the circumference, especially in the wall area, has to be avoided.

Sometimes the burden descent of a blast furnace is erratic. How is the mechanism? Ore burden materials and coke flow rather easily through bins, as can be observed in the stock house of a blast furnace. Hence in the area in the blast furnace where the material is solid, the ore burden and coke flow with similar ease to the void areas. Nevertheless, blast furnace operators are familiar with poorly descending burden (Figure 33, next page, for an example). Also the phenomenon of "hanging" (no burden descent) and "slips" (fast uncontrolled burden descent) are familiar. From the analysis in this section it follows that, in general, the cause of poor burden descent

7 – The process: burden descent and gas flow control

must be found in the configuration of the melting zone. The materials "glue" together and can form internal bridges within the furnace. Poor burden descent arises at the cohesive zone. The effect of a slip is, that the layer structure within the furnace is disrupted and the permeability for gas flow deteriorates (See Figure 48).

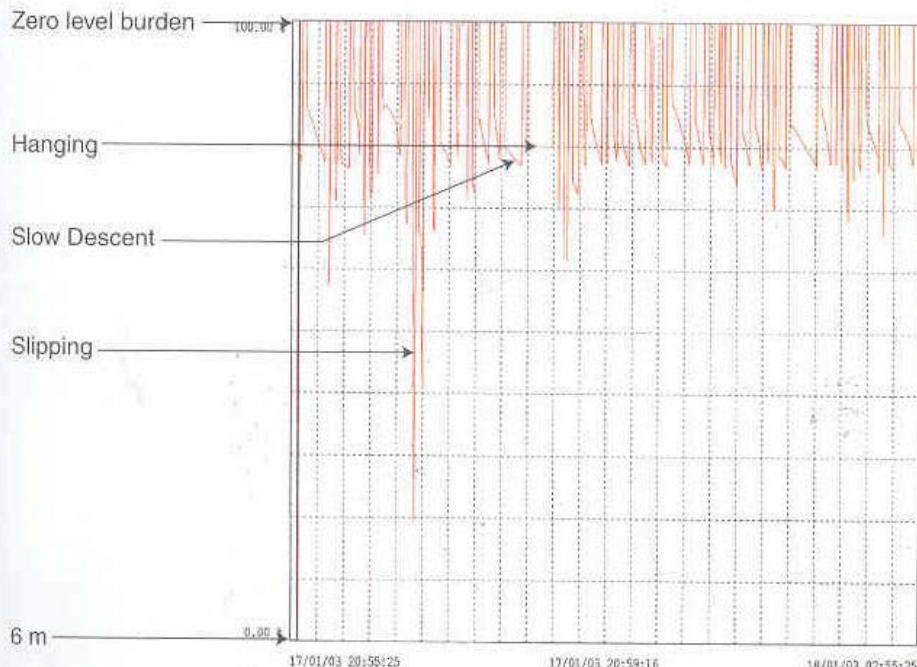


Figure 33: Irregular Burden Descent

7.2 Burden Descent: System of Vertical Forces

The burden descends because the downward forces of the burden exceed counteracting upward forces. The most important downward force is the weight of the burden; the most important upward force is the pressure difference between the blast and top pressure. The cohesive zone is the area with the highest resistance to gas flow, which leads to a high pressure drop over the cohesive zone and to a large upward force. If this pressure difference becomes too high, the burden descent can be disturbed (Figure 34, next page). This happens for instance, when a blast furnace is driven to its limits and exceeds the maximal allowable pressure difference over the burden.

In addition to the upward force arising from the blast pressure, friction forces from the descending burden are impacting on the burden descent: the coke and burden are pushed outward over a cone of stationary or slowly descending central coke. Also the wall area exerts friction forces on the burden. These friction forces can become rather large.

7 – The process: burden descent and gas flow control

The coke submerged in hot metal also exerts a high upward force on the burden due to buoyancy forces (Figure 35) as long as the coke is free to move upwards and does not adhere to the bottom.

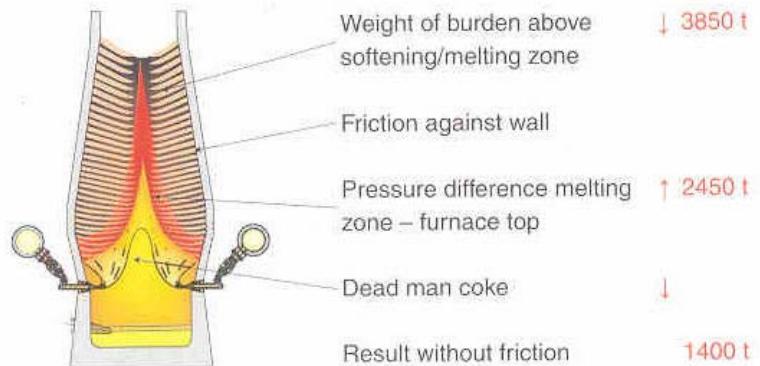


Figure 34: System of vertical forces in the Blast Furnace

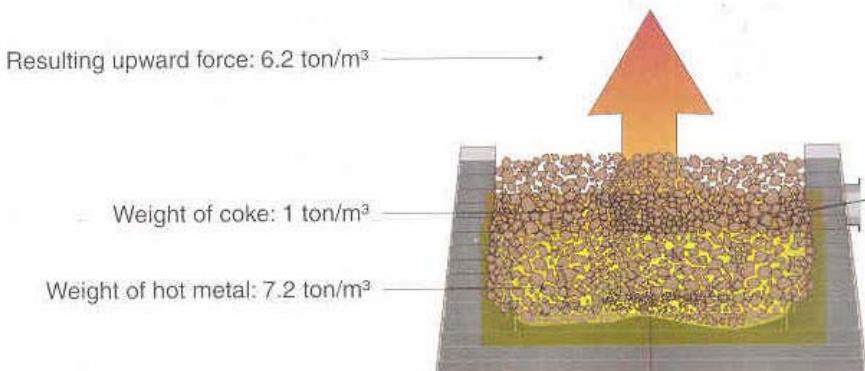


Figure 35: Upward force from hearth liquids

In operational practice poor burden descent is often an indicator of a poor blast furnace process. The reasons can be:

- The upward force is too high. Experienced operators are well aware of the maximum pressure difference over the burden that allows smooth operation. If the maximum allowable pressure difference is exceeded (generally 1.6 to 1.9 bar), the process is pushed beyond its capabilities; burden descent will become erratic, resulting in frequent hanging, slipping and chills.
- A hot furnace is also known to have poorer burden descent. This is because the downward force decreases due to the smaller weight of burden above the melting zone. In addition, there is more slag hold-up above the tuyeres, because of the longer distance and the (primary) slag properties.
- Burden descent can be very sensitive to casthouse operation because of the above-mentioned upward force on the submerged coke.
- The friction forces become high due to softening and melting material in the melting zone leading to bridge formation.

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7.3 Gas flow in the blast furnace

The gas generated at the tuyeres and at the melting zone has a short residence time of 6 to 8 seconds in the blast furnace (section 2.4). During this time the gas cools down from the flame temperature to the top gas temperature, from 2,000 to 2,200°C down to 100 to 150°C, while simultaneously removing oxygen from the burden. The vertical distance between tuyeres and stockline is around 22 metres. Therefore, the gas velocity in the furnace is rather limited, in a vertical direction about 2 to 5 m/s, which is comparable with a wind speed of 2 to 3 Beaufort. During the 6 to 8 seconds the chemical reactions take place. How is the gas distributed through the furnace?

First consider the difference between the coke layers and the ore burden. It is important to note, as indicated in Figure 36, that ore burden has a higher resistance to gas flow than coke. The resistance profile of the furnace determines how gas flows through the furnace. The gas flow along the wall can be derived from heat losses or hot face temperatures as the gas will heat the wall as it travels past as described in the next section.

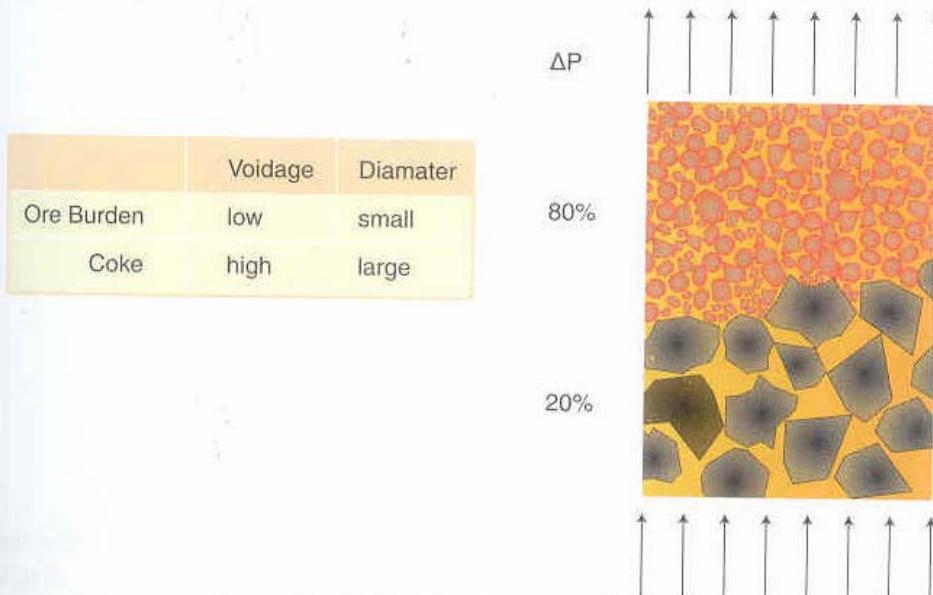


Figure 36: Pressure loss through coke and ore

As soon as the ore burden starts to soften and melt at about 1,100°C, the burden layer collapses and becomes (nearly) impermeable for gas. If this happens in the centre of the furnace the central gas flow is blocked.

7.3.1 Observation of heat fluxes through the wall

Figure 37 shows the temperature at the hot face of the furnace wall. It has been observed in many furnaces, that suddenly the temperature rises in minutes and decreases over the next hour(s). This is often attributed to the loss of scabs (build-up) on the furnace wall.

The explanation put forward in this book is that such temperature excursions are due to "short-circuiting" of gas along the furnace wall. These "short-circuits" are due to the formation of gaps along the furnace wall creating a very permeable area where the hot gasses preferentially flow. Low CO₂ concentrations in the wall area during such excursions have been observed and confirm the "short-circuiting". The basic premise of the present book is that heat losses through the wall are caused by gas flow along the walls. The gas is more or less directly coming from the raceway.

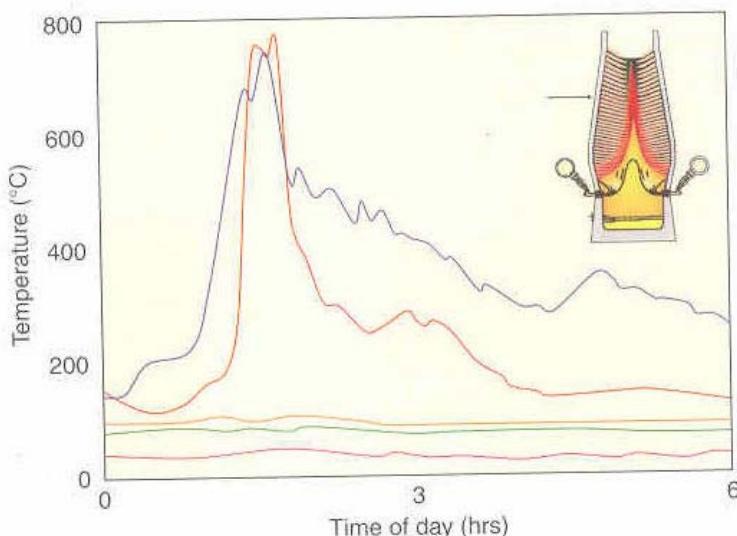


Figure 37: Temperature at hot face

Why does the gas flow along the wall? Gas takes the route with the lowest resistance. The resistance for gas flow in a filled blast furnace is located in the ore layers, since its initial permeability is 4 to 5 times less than the permeability of coke layers. There are two areas in the blast furnace that have the highest permeability: the centre of the furnace if it contains sufficient coke and the wall area. At the wall there can be gaps between the descending burden and the wall. In the centre of the furnace there can be a high percentage of coke and there can be relatively coarse ore burden due to segregation.

7.3.2 Two basic types of melting zone

The efficiency of the furnace is determined by the amount of energy used in the process. Heat losses to the wall and excess top gas temperature are examples of energy losses. The top gas contains CO and H₂, which have a high calorific value,

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therefore, the efficiency of a blast furnace is determined by the progress of the chemical reactions and thus by the gas flow through the furnace.

Two basic types of gas distribution can be discriminated: the "central working" furnace and the "wall working" furnace. The typology has been developed to explain differences in operation. Intermediate patterns can also be observed.

In the "central working" furnace the gas flow is directed towards the centre. In this case the centre of the furnace contains only coke and coarse burden materials and is the most permeable area in the furnace. The melting zone takes on an "inverted V shape".

In a "wall working" furnace the gas flow through the centre is impeded, e.g. by softening and melting burden material. The gas flows preferentially through the zone with highest permeability, i.e. the wall zone. In this case the melting zone takes the form of "W shape". Figure 38 shows both types.

Both types of gas flow can be used to operate a blast furnace, but have their own drawbacks. The gas flow control is achieved with burden distribution.

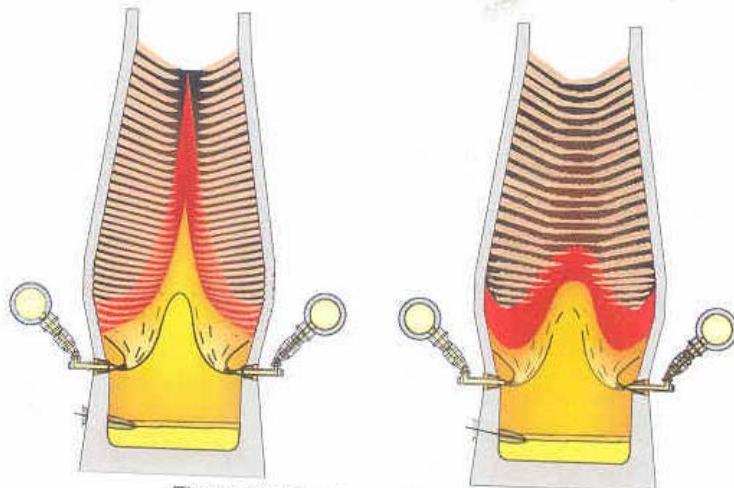


Figure 38: Two types of melting zone,
Central working (left) and Wall-working (right)

7.3.3 Central working furnace

The two types of gas flow through a furnace can be achieved with the help of the burden distribution. In Figure 39 (next page) the ore to coke ratio over the radius is shown for a central working furnace. In the figure the centre of the furnace only contains coke. Therefore, in the centre of the furnace no melting zone can be formed and the gas is distributed via the coke slits from the centre towards outside radius of the furnace. The melting zone gets an inverted V or even U shape. The central coke column not only serves as a gas distributor, but as well as a type of pressure valve: it functions to stabilise the blast pressure.

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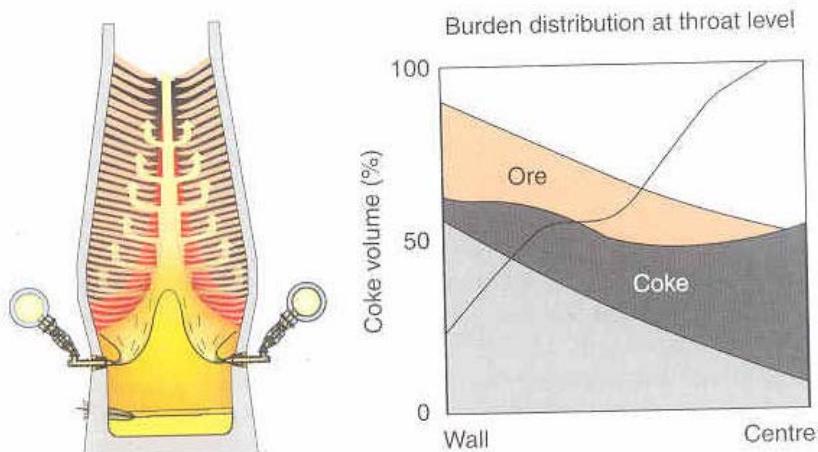


Figure 39: Central working furnace

It depends on the type of burden distribution equipment how the coke can be brought to the centre. With a bell-less top the most inward positions of the chute can be used. With a double bell system the coke has to be brought to the centre by coke push (see below) and by choosing the right ore layer thickness in order to prevent the flooding of the centre with ore burden materials.

In the central working furnace there is a relatively small amount of hot gas at the furnace wall: hence low heat losses. As a result the melting of the burden in the wall area takes place close to the tuyeres, so the root of the melting zone is low in the furnace. The risk of this type of process is that ore burden is not melted completely before it passes the tuyeres. This could lead to the observation of lumps of softened ore burden through the tuyere peep sites. This can lead from slight chilling of the furnace (by increased direct reduction) and irregular hot metal quality to severe chills and damage of the tuyeres.

Limiting the risk of a low melting zone root can be done with gas and burden distribution. Operational measures include the following.

- Maintain a sufficiently high coke percentage at the wall. Using nut coke in the wall area can also do this. Note that an ore layer of 55 cm at the throat needs about 20 to 22 cm of coke for the carburisation and direct reduction. So if the coke percentage at the wall is under 27%, a continuous ore burden column is made at the wall.
- Ensure a minimum gas flow along the wall in bosh and belly, which can be monitored from heat loss measurements and/or temperature readings. If the gas flow along the wall becomes too small, it can be increased by means of burden distribution (more coke to the wall or less central gas flow) or by increasing the gas volume per tonne hot metal (by decreasing oxygen).
- Control the central gas flow. Note that the gas flow through the centre leaves the furnace at a high percentage of CO and H₂ and a high temperature. The energy content of the central gas is not efficiently used in the process and thus the central gas flow should be kept to a minimum.

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The central working furnace can give very good, stable process results with respect to productivity, hot metal quality and reductant rate. It also leads to long campaign length for the furnace above the tuyeres. However, the process is very sensitive for deviations in burden materials, especially the size distribution, casthouse operation, circumferential symmetry of injection and stops /starts of the furnace.

7.3.4 Wall working furnace

In Figure 40 the wall working furnace is presented. Melting ore burden blocks the centre of the furnace and the gas flow is directed towards the wall area.

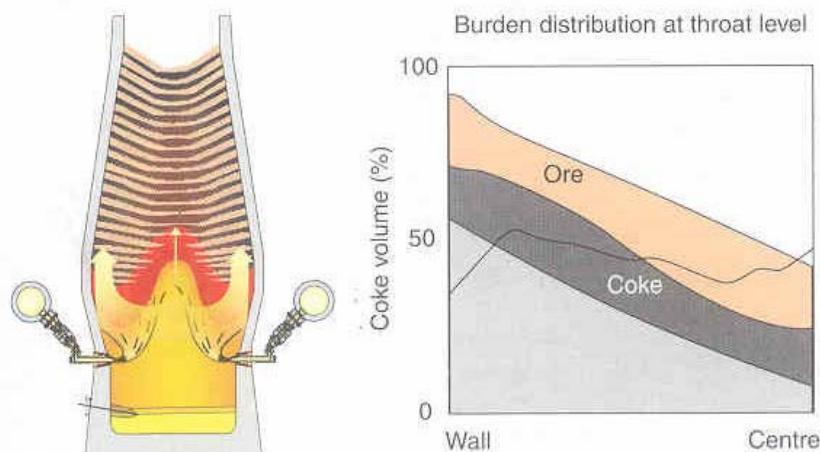


Figure 40: Wall working furnace

The gas flow causes high heat losses in the area of the furnace where a gap can be formed between burden and wall i.e. in lower and middle shaft. The melting zone gets a W shape or even the shape of a disk. In this situation the root of the melting zone is higher above the tuyeres, which makes the process less sensitive for inconsistencies. The process can be operated rather efficiently, however, due to the high heat losses the wear of the refractory in the shaft is much more pronounced than with the central working furnace. The gas passing along the wall can also cool down rapidly and in doing so lose its reduction capabilities. Moreover the fluctuations in the pressure difference over the burden are more pronounced, which leads to limitations in productivity.

7.3.5 Gas distribution to ore layers

Gas produced in the raceway is distributed through the coke layers in the cohesive zone and into the granular coke and ore layers, as shown in Figure 41.

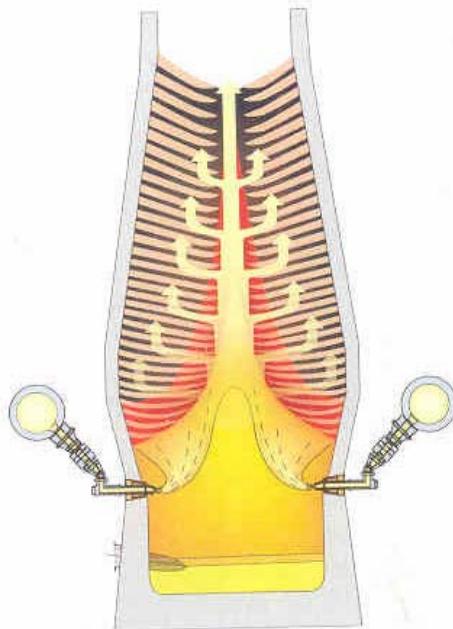


Figure 41: Schematic presentation of gas distribution through coke layers

The ore burden layers account initially for about 80% of the resistance to gas flow. The reduction process takes place within these layers.

What determines the contact between the gas and the ore burden layers? The most important factor determining the permeability to gas flow is the voidage between particles. As shown in Figure 14 (Section 3.2.1), the voidage between particles depends heavily on the ratio of coarse to small particles. The wider the size distribution, the lower the voidage. Moreover, the finer the materials, the lower the permeability (Chapter 3).

In practical operations the permeability of ore burden material is determined by the amount of fines (percentage under 5mm). Fines are very unevenly distributed over the radius of the furnace, as is indicated by the typical example shown in Figure 42 (next page). Fines are concentrated along the wall especially under the point of impact of the new charge with the stockline.

If a bell-less top is used, the points of impact can be distributed over the radius. With a double bell charging system the fines are concentrated in a narrow ring at the burden surface and close to the wall. When the burden is descending the coarser materials in the burden follow the wall, while the fines fill the holes between the larger particles and do not follow the wall to the same extent as the coarser particles. Therefore, upon descent the fines in the burden tend to concentrate even more.

Moreover, sinter and lump ore can break down during the first reduction step (from hematite to magnetite). This effect is stronger if the material is heated more slowly. Thus, the slower the material is heated the more fines are generated, the extra fines impede the gas flow even more, giving rise to even slower heating.

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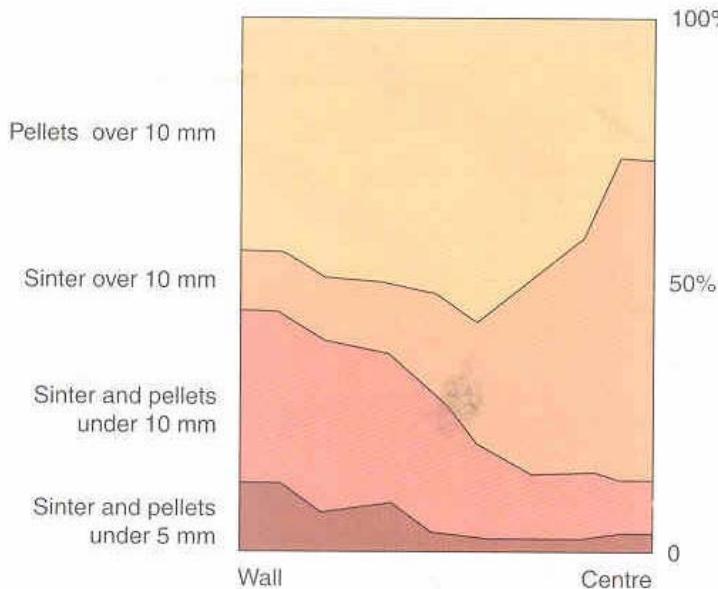


Figure 42: Distribution of fines over the radius, double bell simulation (after Geerdes et al, 1991)

In summary:

- The permeability of the ore burden is determined by the amount of fines.
- The amount of fines is determined by:
 - The screening efficiency in the stock house.
 - The physical degradation during transport and charging.
 - The method of burden distribution used.
 - The low temperature degradation properties of the burden.

These effects cause a ring of burden material with poor permeability in many operating blast furnaces. This ring of material in particular is often difficult to reduce and to melt down. Sometimes, unmolten ore burden materials are visible as scabs through the peepsites of the tuyeres. These scabs can cause operational upsets like chilling the furnace or tuyere failures. It is a misunderstanding to think that these scabs consist of accretions fallen from the wall.

7.4 Fluidisation and channelling

The average gas speed above the burden is rather low, as shown in chapter 2. However, in a central working furnace the gas speed might locally reach 10 m/s or more especially in the centre of the furnace. This is well above theoretical gas velocities at which fluidisation can be observed.

Coke fluidises much more easily than ore burden because of its lower density. It is believed that the ore burden secure the coke particles in the centre, nevertheless, if local gas speeds become too high, fluidisation may occur. Fluidisation of coke has been observed in operating furnaces as well as models of the furnace. It leads to a relatively open structure of coke. It has even been observed, that pellets on the border of fluidising coke "dives" into the coke layers.

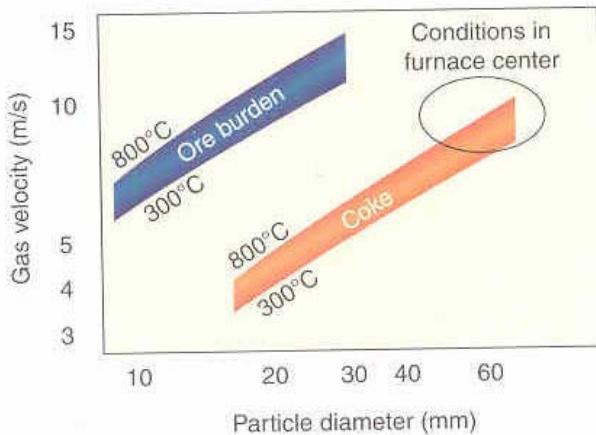


Figure 43: Gas velocities for fluidisation of ore burden and coke. Area indicated is range from 800°C to 300°C at 1 atmosphere pressure (after Biswas, 1981)

If the fluidisation stretches itself into the lower furnace, channelling can take place, short-circuiting the lower furnace (or even the raceway) with the top. These are open channels without coke or ore burden in it. Channelling is observed as a consequence of operational problems, for example, delayed casts can create higher local gas speeds, resulting in channelling. During channelling, the gas might escape through the top with a very high temperature and low utilisation, since the gas was not in good contact with the burden.

7.5 Burden distribution

Burden distribution can be used to control the blast furnace gas flow. The conceptual framework of the use of burden distribution is rather complex, since the burden distribution is the consequence of the interaction of properties of the burden materials with the charging equipment.

7.5.1 Properties of burden materials

Figure 44 shows the angles of repose of the various materials used in a blast furnace. Coke has the steepest angle of repose, pellets have the lowest angle of repose and sinter is in between. Hence, in a pellet charged furnace the pellets have the tendency to slide to the centre.

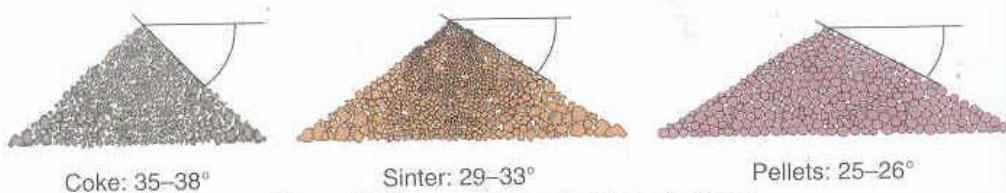


Figure 44: Segregation and angles of repose

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Fines concentrate at the point of impact and the coarse particles flow “downhill” while the fine particles remain below the point of impact. This mechanism, known as segregation, is also illustrated in Figure 44.

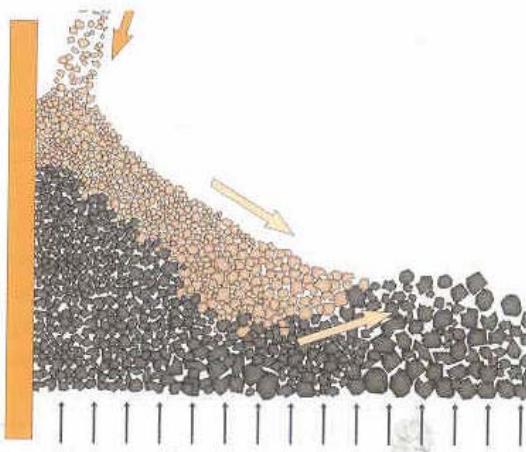


Figure 45: Coke push effect with gas flow

When burden is charged into the furnace, it pushes the coarse coke particles on the top of the coke layer towards the centre. This effect is called coke push and is more pronounced when the furnace is on blast. It is illustrated in Figure 45.

7.5.2 The charging equipment

The type of charging mechanism used has a major impact on the distribution of fines. Figure 46 shows the bell-less top and double bell systems.

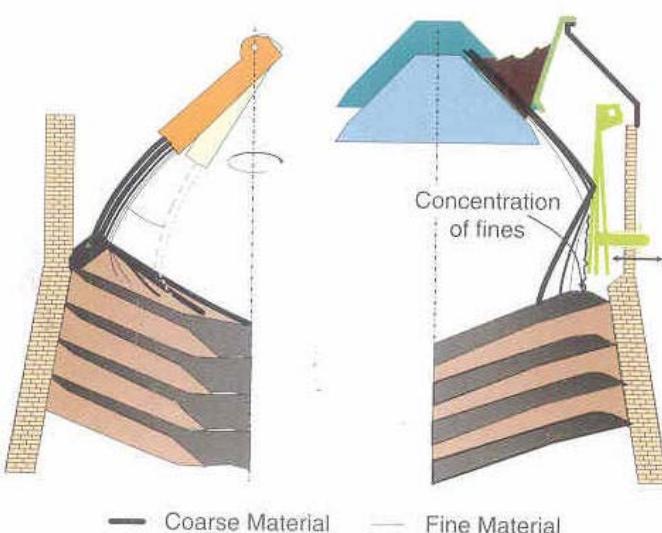


Figure 46: Bell-less top charging (left) and double bell charging (right): comparison of the segregation of fines on the stockline

In a bell-less top the possibility exists to distribute the fines in the burden over various points of impact by moving the chute to different vertical positions. Coke can be brought to the centre by programming of the charging cycle. With a double bell charging system there is less possibility to vary the points of impact and fines will be concentrated in narrower rings. Modern blast furnaces with a double bell charging system are mostly equipped with movable armour, which give certain flexibility with respect to distribution of fines and the ore to coke ratio over the diameter. However, its flexibility is inferior to the more versatile bell-less system.

7.5.3 Mixed layer formation

The model of thinking applied up to here takes clean ore and coke layers as a starting point. However, since the average diameter of coke 45 to 55 mm is much larger than that of pellets and sinter (typically under 15 mm and 25 mm respectively) burden components dumped on a coke layer will tend to form a mixed layer. This mixed layer will have permeability comparable with the ore layer. The formation of mixed layers is also produced by protruding or recessed parts of the wall: such as protruding cooling plates, missing armour plates, wear of refractory at the throat and so on. The mixed layers have a different permeability and can give rise to circumferential process asymmetry.

7.5.4 Gas flow control

The optimised gas flow in a modern furnace operated at high productivity and low coke rate has the inverted V shaped melting zone type as described above. However, the gas escaping through the (ore-free) centre leaves the furnace with a low utilisation. This loss of "unused" gas should be minimised.

If the central gas flow is too high, there is a too small gas flow along the wall for heating, reduction and melting of the ore burden and consequently the root of the melting zone comes close to the tuyeres. In this situation the reductant rate will be high and there is a high chance of tuyere damage. It is essential that the gas flowing through the centre distributes itself through the coke slits to the burden layers. Therefore, the permeability of the central coke column must not be too high, which means that the diameter of the central coke column must not be too wide.

If the central gas flow is (partially) blocked, a relatively large part of the gas escapes along the wall and is cooled down. The result is the part of the gas is cooled down low in the furnace and the reduction reactions slow down. In this situation the central gas flow is small and heat losses are high.

Experience has shown that wall gas flow and central gas flow are strongly correlated. Gas flow control is based on keeping the balance between central and wall gas flow to the optimum.

The difficulty with gas flow control is that the gas flow is influenced by many changes in burden components, process parameters and installation specifics. The variation in the percentage of fines near (but not at) the wall and the low temperature breakdown properties of the burden are especially important.

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The gas flow is closely monitored in order to control it. Instrumentation of the blast furnace is described in the next section.

The most important parameters to define the actual gas flow are:

- Burden descent (stock rods, pressure taps) and pressure difference over the burden.
- The wall heat losses or temperatures at the wall.
- Stockline gas composition and temperature profile.

Gas flow control and optimised burden distribution are found on a trial and error basis, and have to be developed for every furnace individually. Some general remarks can be made.

1. Gas flow is mainly controlled with coke to ore ratio over the radius. An example of a calculated burden distribution is shown in Figure 47. Note the ore free centre.
2. The centre of the furnace should be permeable and no or minimal (coarse) ore burden should be present.
3. The coke percentage at the wall should not be too low. Note that 55 cm of ore in throat consumes about 20 cm of coke for direct reduction and carburisation. A continuous vertical column of burden material should be prevented. A coke slit should be maintained between all ore layers.
4. Concentration of fines near the wall should be prevented.
5. The central gas flow is governed by the amount of ore burden reaching the centre. The amount of ore reaching the centre heavily depends on the ore layer thickness and the amount of coarse coke lumps. To reach a stable gas flow the central gas flow should be kept as consistent as possible and consequently, when changes in ore to coke ratio are required, the ore layer should be kept constant. This is especially important when changing the coal injection level as this will result in big changes in the relative layer thickness of ore and coke are made.
6. The coke layer thickness at the throat is typically in the range of 45 to 55 cm. In our example in section 2.3 it is 49 cm. The diameter of the belly is more than 45% higher than the diameter of the throat. Hence, the surface more than doubles during burden descent and the layer thickness is reduced to less than half the layer thickness at the throat. Japanese rules of thumb indicate that the layer thickness at the belly should not be less than 18 cm. The authors have, however, successfully worked with a layer thickness of coke at the belly of 14 cm.

In the practical situation small changes in ore layer thickness can strongly influence central gas flow. This effect is generally stronger in double bell– movable armour furnaces than in furnaces equipped with a bell-less top.

An example for a burden distribution control scheme is given in Table 19 (next page). If more central gas flow is required then Coke 3 replaces schedule Coke 2. Replacing Coke 2 with Coke 1 reduces central gas flow.

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Table 19: Bell-less top charging schedules with varying central gas flow

	Position	11	10	9	8	7	6	5	4	3	2	1
		Wall										
Coke 1	More central	–	14%	14%	16%	14%	14%	14%	–	6%	–	8%
Coke 2	Normal	–	14%	14%	14%	14%	14%	14%	–	6%	–	10%
Coke 3	Less central	–	14%	14%	12%	14%	14%	14%	–	6%	–	12%
Ore		16%	16%	16%	12%	10%	10%	10%	10%	10%	10%	10%

Similar schedules can be developed for a double bell charging system. With a double bell system, the use of ore layer thickness can also be applied: a smaller ore layer gives higher central gas flow and vice versa.

If a major change in coke rate is required, the operator has the choice either to change the ore base and keep the coke base constant, or change the coke base and keep the ore base constant. Both philosophies have been successfully applied. The operators keeping the coke base constant point to the essential role of coke for maintaining blast furnace permeability, especially the coke slits.

The authors, however, favour a system in which the ore base is kept constant. The gas distribution is governed by the resistance pattern of the ore burden layers and – as mentioned above – by the amount of ore burden that reaches the centre. The latter can change substantially when changing the ore base, especially in furnaces equipped with double bell charging.

An illustrative example showing a change in coke rate from 350 kg/tHM to 300 kg/tHM is presented in Table 20. The ore base is kept constant and coke base reduced. Experience has shown that relatively minor changes in burden distribution will be required for optimisation of the central gas flow (i.e. coke distribution). The burden distribution adjustments can be applied as a second step if required.

Table 20: Coke base change when PCI rate changes

	Old situation	New Situation
Coke rate	350 kg/tHM	300 kg/tHM
Coke base	21 t	18 t
Ore base	90 t	90 t
Burden distribution	No change until required	

Burden distribution changes should be based on an analysis of the causes of changes in gas flow. The gas flow can also be influenced by operational problems, such as a low burden level or problems in the casthouse. In this situation adjustments in the burden distribution will not give satisfactory results. Heat losses through the wall are very closely related to burden descent. Therefore, the cause of high heat loads should be analysed together with other process data. An example of a burden distribution is shown in Figure 47 (next page).

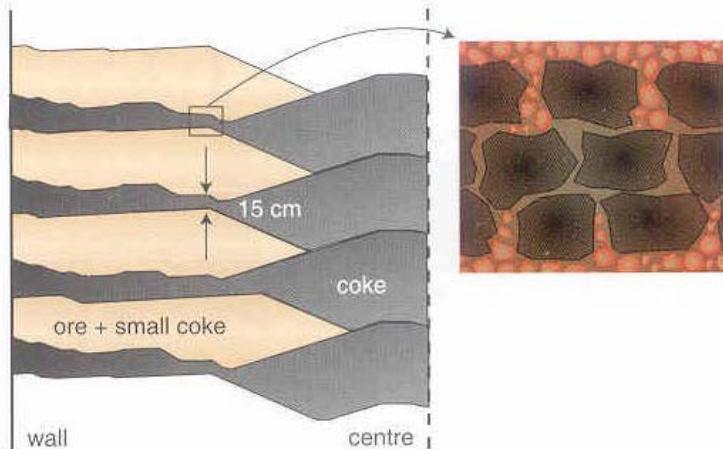


Figure 47: Example of burden distribution with an ore-free centre and ore burden penetration in coke layer

7.6 Erratic burden descent and gas flow

The burden descent sometimes becomes erratic (see Figure 33). What happens in the furnace if it hangs and slips? The mechanism of hanging and slipping is illustrated in Figure 48 on the next page.

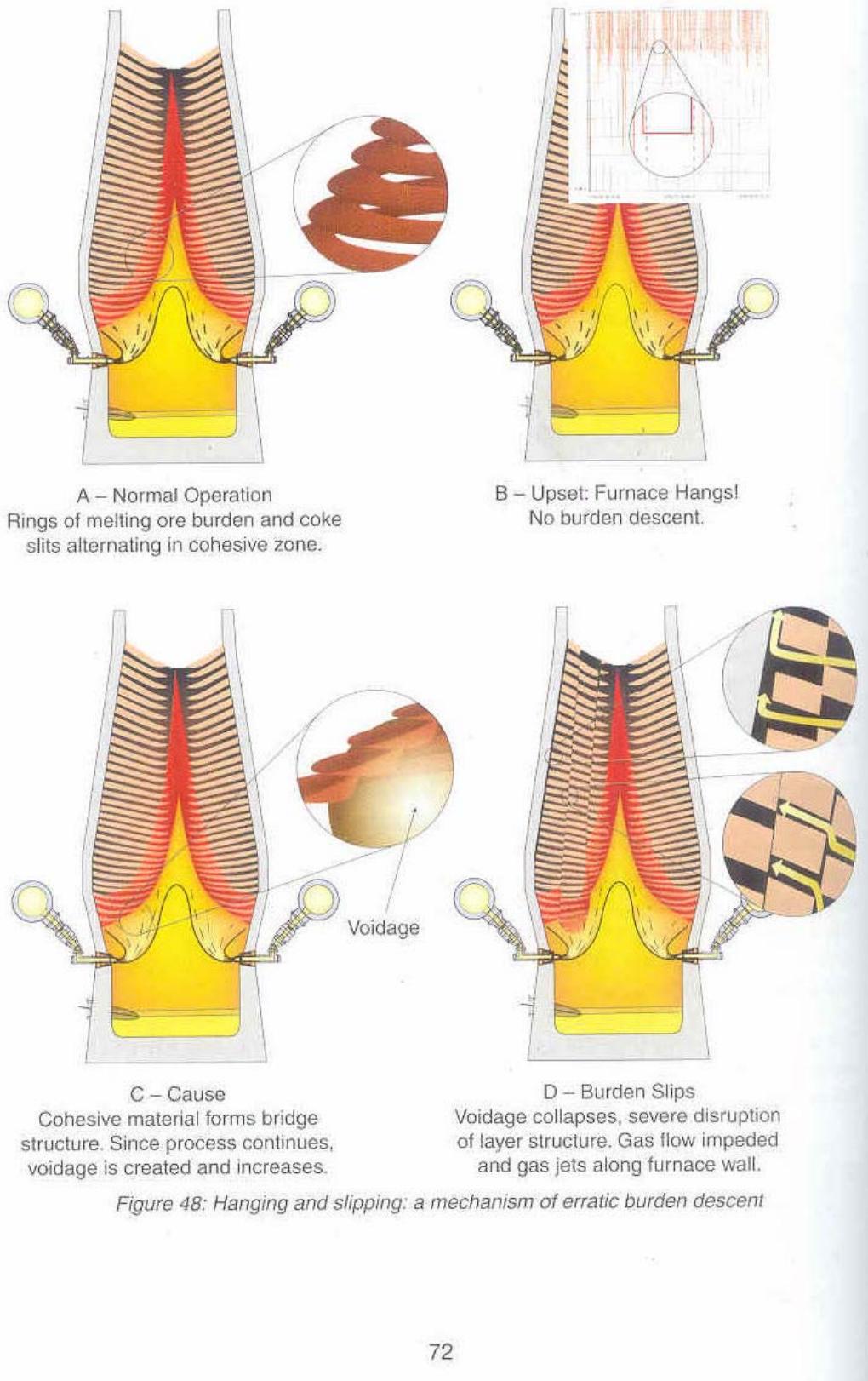
First, the furnace hangs because the net downward force is too low. This can be caused by high friction forces. Note, that granular coke, pellets and sinter flow easily downwards, as can be observed in the stockhouse. Therefore, high friction forces arise at the cohesive zone, where bridges of melting ore burden are formed.

Second, while the furnace hangs, the process continues: coke is consumed and ore burden melts. Therefore, voidage arises in or below the melting zone.

Third, when this voidage becomes too big, it collapses: the furnace slips. The layer structure is completely disrupted and the gas flow through these layers is impeded. This leads again to areas in the furnace where ore burden is insufficiently reduced and remains in a cohesive state for too long. These areas will form the bridges for next time the furnace hangs.

The problem can only be solved by re-establishing the layer structure within the furnace, which means, that the complete content of the furnace has to be refreshed: the furnace has to be operated on reduced blast volume for five to ten hours.

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7.7 Blast furnace instrumentation

An overview of blast furnace instrumentation as discussed in various parts of the text is given in Figure 49.

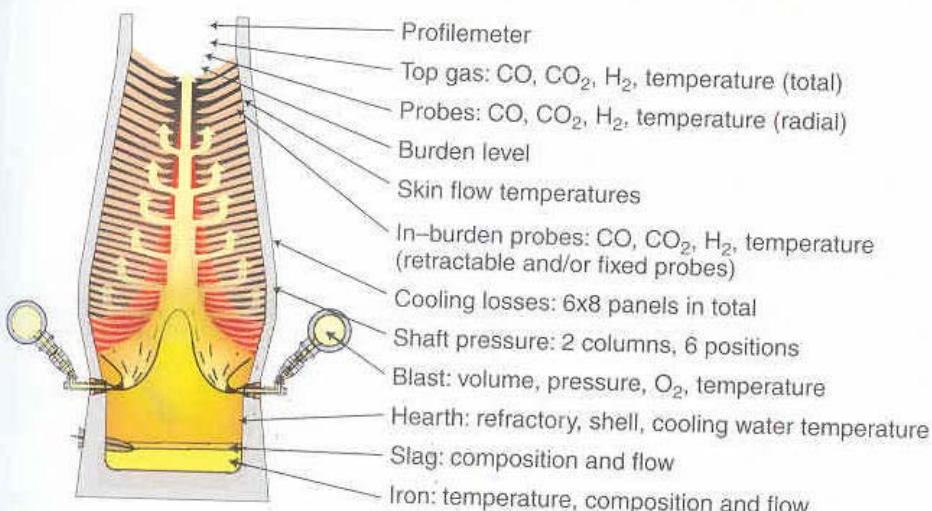


Figure 49: Overview of Blast Furnace instrumentation

7.8 Blast Furnace daily operational control

In this section the blast furnace daily operational control is discussed. The better the consistency of the blast furnace input, the lower the need for adjustments in the process. Ideally, a good consistency of the input allows the operator to "wait and see". The need for daily operational control is a consequence of the variability of the input and – sometimes – the equipment.

The process must be controlled continuously, which may require changes to be made on a daily or even shift basis. The changes are aimed towards:

- Correct iron and slag composition. The burden and coke are adjusted to get the correct chemical composition of the iron and slag. For the latter especially the basicity of the slag is important because of its effect on hot metal sulphur.
- Correct iron and slag composition also implies control of thermal level, since the hot metal silicon is correlated with the hot metal temperature. So, there are daily requirements for burden calculations with updated chemical analysis of the burden components and actual burden, and frequent adjustments of the thermal level of the furnace. Adjusting the coke rate or the auxiliary reductant injection through the tuyeres can achieve the latter.
- Stable process control. Burden descent (as measured by the stock rods, Figure 31, or pressure taps, Figure 50), blast furnace productivity and efficiency are evaluated on the basis of hourly data. Raceway conditions (e.g. flame temperature) are monitored or calculated. The total process overview gives an indication whether or not adjustments are required. Pressure taps indicate whether

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or not "short circuiting" of gas flow along the wall takes place. In stable periods the layers of coke and ore can be followed when passing the pressure taps.

- Gas flow control. The subject of gas flow control is discussed in more detail below.

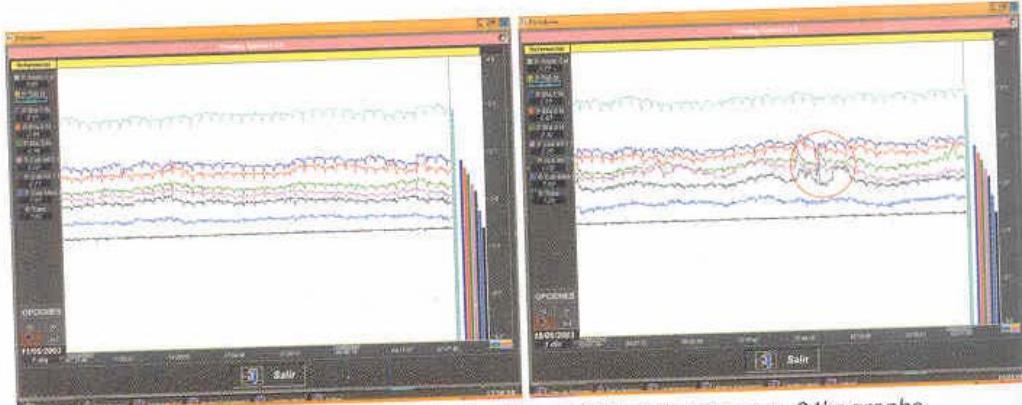


Figure 50: Pressure taps indicating the stability of the process, 24hr graphs.
The example shows stable (left) and unstable (right) operation, with short-circuiting of gas flow encircled in red. (Courtesy: Siderar, Argentina)

Measurements and data required for daily gas flow control are shown in Figure 52 (next page). The gas flow through the furnace can be monitored with the help of global top gas composition, top gas composition across the radius, heat losses at the wall and gas flow along the wall. The latter can be measured with the short in-burden probes: the probes measure the temperature about three metres below the burden level up to 50 cm into the burden. If temperatures are low (under 100°C) the burden is not yet dry and more gas flow in wall area is required to increase the drying capacity at the wall.

If the furnace seems in need of an adjustment of the gas flow, a change to the burden distribution can be considered. However, a thorough analysis of the actual situation has to be made. For example, consider the situation whereby high central temperatures are observed. If these high central temperatures are observed together with low heat losses and low gas utilisation, then the central gas flow can be considered to be too high. The appropriate action in this case would be to consider changes to the burden distribution to decrease the central gas flow. If, on the other hand, the high central temperatures are combined with a good gas utilisation and good wall gas flow, then there is no need to change the layers of ore and coke. The appropriate action in this scenario would be to consider working with lower gas volume per tonne HM i.e. with higher oxygen enrichment.

Note also, that the heat losses are very sensitive to the burden descent. Irregular burden descent leads to gaps at the wall and high heat losses. So, if a furnace is showing high heat losses, again, the cause should be investigated in detail before adjusting burden distribution. For example, if a blast furnace is pushed to its production limits and burden descent suffers due to the high pressure difference over the burden, the solution of the high heat losses is to reduce production level (or gas volume) and not to adjust burden distribution.

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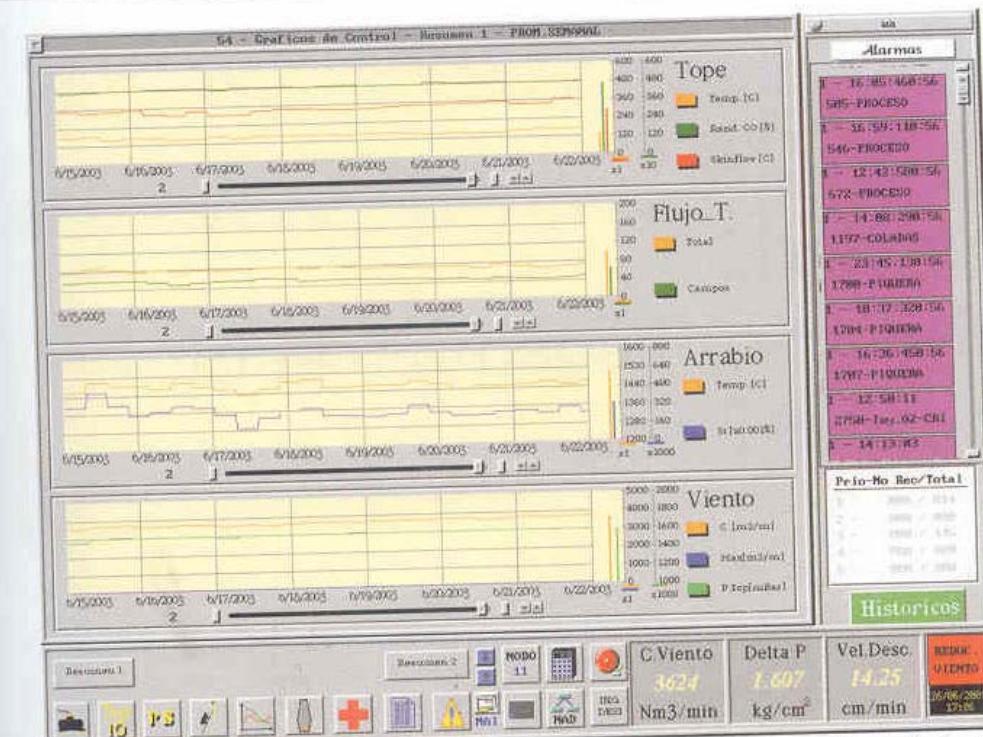


Figure 51: Presentation of process data in an operational furnace. The weekly graph gives an overview from the stability and development of the process. From top downwards:
 Tope – CO utilisation (%), skin flow temperature (°C) and top temperature (°C);
 Flujo T – Total heat loss and sum of fields (GJ/hr);
 Arrabio – Hot metal temperature (°C) and silicon (%);
 Viento – Blast volume (Nm³/min) and top pressure (bar)

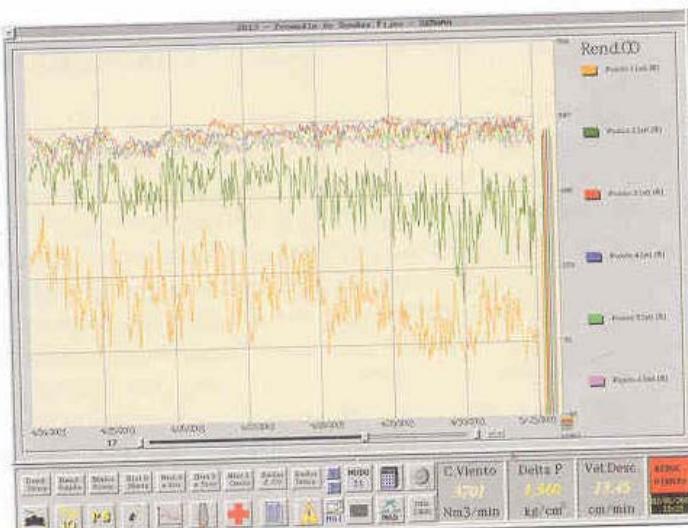


Figure 52: Example of gas flow control. The radial gas distribution is measured with above burden probes, expressed as CO utilisation (7 day graph). The decreasing gas utilisation in the centre of the furnace (point 1 and 2, yellow and dark green) shows increased central working.

Chapter 8

Blast Furnace Productivity and Efficiency

The production rate of a blast furnace is directly related to the amount of coke used in front of the tuyeres in a stable situation. This is due to every charge of coke at the top of the furnace bringing with it an amount of ore burden materials. In a stable situation the hot metal is produced as soon as the coke is consumed. The productivity of a blast furnace increases as less reductant is used per tonne hot metal. In the present chapter the basics behind blast furnace productivity, the chemical reactions and efficiency are discussed (see also Hartig et al, 2000).

8.1 The raceway

8.1.1 Production rate

In the raceway hot gas is formed which melts the burden material and is used to drive the chemical reactions in the furnace. Given a certain amount of coke and coal used per tonne hot metal, the production rate of a blast furnace is determined by the amount of oxygen blown through the tuyeres. The more oxygen that is blown into the furnace, the more coke and coal are consumed and form carbon monoxide (CO), and the higher the production rate becomes. In addition, the lower the reductant requirement per tonne of hot metal (tHM), the higher the production rate. A quantitative example is indicated below. Coke (and coal) are not only gasified in front of the tuyeres, but are also used for carburisation of iron (hot metal contains 4.5% C) and for direct reduction reactions (section 8.2). The coke rate is expressed as standard coke, i.e. coke with a carbon content of 87.5%.

In an operating blast furnace the use of the reductants can be as follows:

Coke rate	300 kg/tHM	300 kg/tHM
Coal injection	200 kg/tHM	replacement ratio 0.85 170 kg/tHM
Total (as standard coke)		470 kg/tHM

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Total (as standard coke)	470 kg/tHM
Required for carburisation	50 kg/tHM
Required for direct reduction	100 kg/tHM
Gasifies in front of the tuyeres	320 kg/tHM

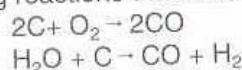
The 320 kg/tHM standard coke which is used in front of the tuyeres consists of 170 kg/tHM coke equivalent injected as coal and so per tonne hot metal, 150 kg coke (320–170 kg) is gasified at the tuyeres.

Note the issue of efficiency: if the same amount of oxygen is blown into the furnace, thus maintaining same blast volume and blast conditions, while the reductant rate is 10 kg/tHM lower, the production rate will increase. At a 10 kg/tHM lower reductant rate the production will increase by 3% (320/310-100%). Conversely, if extra coal is put on the furnace for thermal control, the production rate will decrease if blast conditions are maintained. This is a simplified approach. Secondary effects, like the effect on gas flow throughput, the effect on flame temperature and the oxygen content of the coal, have been neglected.

8.1.2 Bosh gas composition

The heat of the blast and the heat generated by the reactions of coke (and coal/auxiliary reductants) in the raceway are used to melt the burden. The heat available to melt the burden depends on the amount of gas produced and on the flame temperature, known as the “raceway adiabatic flame temperature” (RAFT).

The amount and composition of the raceway gas can be calculated using the following reactions that take place in the raceway:



In and directly after the raceway all oxygen is converted to carbon monoxide and all water is converted to hydrogen and carbon monoxide.

Consider the following example; the blast furnace in section 2.3 has a blast volume of 6,500 m³ STP wind with 25.6% oxygen. Ignoring the effects of moisture in the blast and the coal injection, what would be the raceway gas volume and composition?

Blast into the furnace (per minute):

- Nitrogen: 4,836 m³ STP/min ((1-0.256)x6,500)
- Oxygen: 1,664 m³ STP/min (0.256x6,500)

The oxygen generates two molecules of CO for every O₂ molecule, so the gas volume is 8,164 m³ STP/min (4,836+2x1,664). The gas consists of 59% nitrogen (4,836/8,164) and 41% CO (2x1,664/8,164).

The calculation can be extended to include the moisture in the blast and the injection of coal (or other reductants). This is done in section 6.4.

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8.1.3 Raceway flame temperature

The flame temperature in the raceway is the temperature that the raceway gas reaches as soon as all carbon, oxygen and water have been converted to CO and H₂. The flame temperature is a theoretical concept, since not all reactions are completed in the raceway. From a theoretical point of view it should be calculated from a heat balance calculation over the raceway. For practical purposes linear formulas have been derived. Flame temperature is normally in the range of 2,000 to 2,300°C and is influenced by the raceway conditions. The flame temperature increases if:

- Hot blast temperature increases.
- Oxygen percentage in blast increases.

The flame temperature decreases, if:

- Moisture increases in the blast.
- Reductant injection rate increases, since cold reductants are gasified instead of hot coke. The precise effect depends also on auxiliary reductant composition.

Table 21 gives some basic rules with respect to flame temperature effects.

Table 21: Flame temperature effects, rules of thumb

	Unit	Change	Flame temp. (°C)	Top temp. (°C)
Blast temp.	°C	+	100	+ 65 – 15
Coal	kg/t	+	10	– 30 + 9
Oxygen	%	+	1	+ 45 – 15
Moisture	g/m ³ STP	+	10	– 50 + 9

The top gas temperature is governed by the amount of gas needed in the process; the less gas is used, the lower the top gas temperature and vice versa. Less gas per ton hot metal results in less gas for heating and drying the burden.

8.2 Carbon and iron oxides

In the preceding section the formation of gas in the raceway has been described. What happens with the gas when it ascends through the furnace and cools down? First consider what happens with the carbon monoxide.

Carbon can give two types of oxides:

- C + 1/2 O₂ → CO + heat (111 kJ/mole).
This reaction takes place in the raceway
- C + O₂ → CO₂ + heat (391 kJ/mole).
This reaction does not take place in the raceway and is more typical in an area such as a power plant.

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Note that in the second step much more heat is generated than in the first step, therefore, it is worthwhile to convert CO to CO_2 as much as possible in the process. The ratio $\text{CO}_2 / (\text{CO} + \text{CO}_2)$ is called the gas utilisation or gas efficiency and is used extensively in blast furnace operation.

In Figure 53, the equilibrium between $\text{CO} = \text{C} + \text{CO}_2$ is presented for various temperatures. The line indicates the equilibrium of the "Boudouard" reactions. At temperatures above 1,100°C all CO_2 is converted to CO, if in contact with coke. Therefore, at the high temperatures in the bosh and melting zone of the blast furnace only carbon monoxide is present. At temperatures below 500°C all CO has the tendency to decompose into C+ CO_2 . The carbon formed in this way is very fine and is called "Boudouard" carbon.

In operational practice the carbon monoxide decomposition can be observed in refractory material, where there is a CO containing atmosphere in the correct temperature region. This generally is a very slow process.

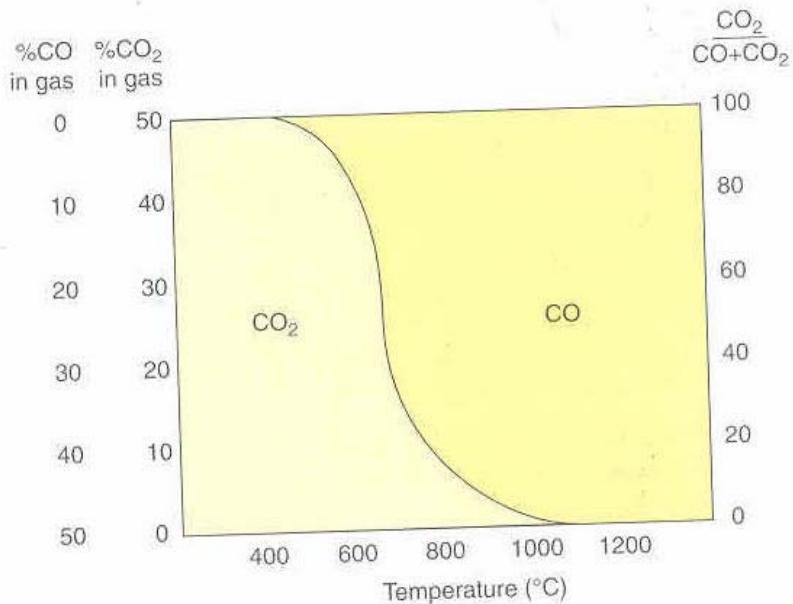


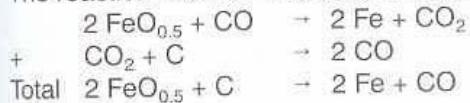
Figure 53: Boudouard reaction: the drawn line indicates equilibrium

8.2.1 Direct reduction

As the hot reducing gases produced in the raceway ascend through the lower furnace, they transfer heat to the ore burden to the extent that it becomes molten at the lower levels of the melting zone. They also remove oxygen from the iron oxides, i.e. they reduce the iron oxides, which contain approximately one oxygen for every two iron atoms. The CO_2 produced from the reaction immediately reacts with the carbon in the coke to produce CO. The total reaction is known as direct reduction, because carbon is directly consumed.

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The reactions can be indicated as below



The direct reduction reaction requires an enormous amount of heat, which is also provided by the specific heat contained in the hot raceway gas.

The direct reduction reaction is very important for understanding the process. In a modern blast furnace the direct reduction removes about a third of the oxygen from the burden, leaving the remaining two-thirds to be removed by the gas reduction reaction. The amount of oxygen to be removed at high temperatures, as soon as the burden starts to melt, is very much dependent on the efficiency of the reduction processes in the shaft. See section 8.2.2.

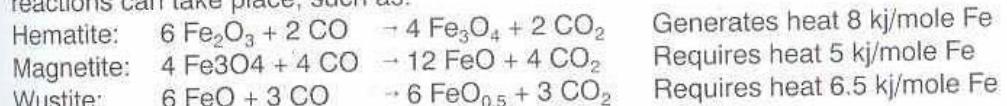
Note two important observations:

- Direct reduction uses carbon (coke) and generates extra CO-gas.
- Direct reduction costs a lot of energy.

In operational practice the direct reduction can be monitored. In many blast furnaces the direct reduction rate (the percentage of the oxygen removed from the burden by direct reduction) or the solution loss (the amount of coke used for the reaction) are calculated on line. Experienced operators are well aware that as soon as the direct reduction rate or the solution loss increases, the blast furnace starts to descend faster, the cohesive zone will come down as the coke below it is consumed. And the furnace will chill. When properly observed, chilling can be prevented, for example by using extra coal injection.

8.2.2 Gas reduction or “indirect” reduction

As soon as temperatures of the gas reduce, the CO_2 becomes stable and reduction reactions can take place, such as:



The reduction is called “gas reduction” because the oxygen is removed from the burden materials with CO gas. H_2 reacts in a similar way. In literature it is also often called “indirect” reduction, since carbon is only indirectly involved in this reaction. The reduction of the $\text{FeO}_{0.5}$ takes place via the direct reduction.

Following the burden descent from the stockline, the reduction from hematite to magnetite starts around 500°C. The reduction from magnetite to wustite takes place in the temperature zone from 600 to 900°C, while the reduction from wustite to iron takes place in the temperature region between 900 and 1,100°C. At the start of melting (1,100 to 1,150°C) $\text{FeO}_{0.5}$ is normally reached. Here FeO is used as a symbol for wustite, however the most stable composition is $\text{FeO}_{0.95}$.

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The reactions are shown in Figure 54.

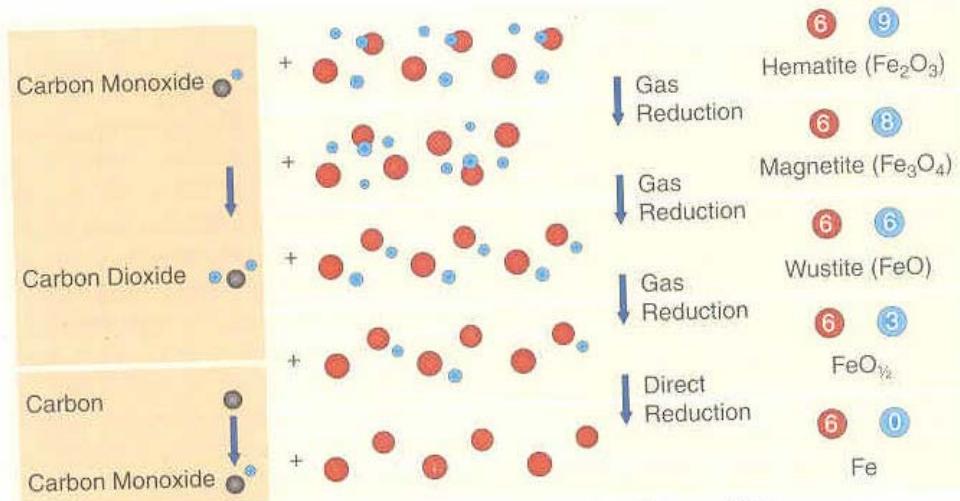


Figure 54: Overview of the reduction of iron oxides

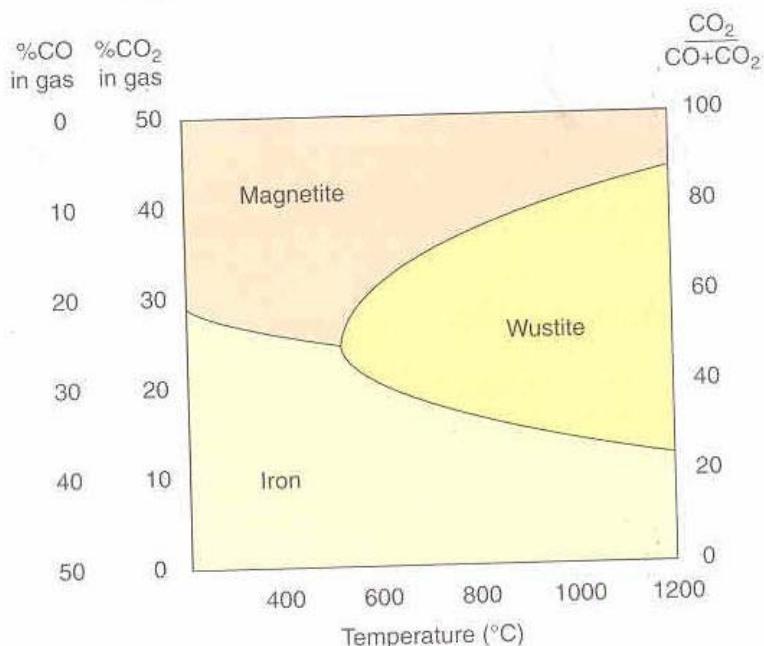


Figure 55: Schematic presentation of the relation between temperatures, CO/CO_2 gas composition and iron oxides. The drawn lines indicate equilibrium.

The equilibrium between the various iron oxides and the gas is shown in Figure 55. The figure shows at what level of temperatures and gas compositions further gas reduction of the burden is no longer possible. The reduction of wustite to iron requires the highest CO concentration, as can be seen in Figure 55, the reduction of wustite requires a gas with a relatively high

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percentage CO. Gas utilisation should be below 30%. If CO₂ is higher, wustite is no longer converted to iron. From these measurements it is clear that the reduction from wustite to iron comes close to equilibrium.

The progress of the reduction reactions in a blast furnace can be detected in two different ways:

- Burden: from quenched furnaces an overview of the progress of the reduction can be derived. An example is shown in Figure 56
- Gas: by sending gas sampling devices down into the furnace, the progress of temperature/gas composition can be derived. Figure 57 (next page) shows typical results from a gas sampling exercise. The data can be depicted in the graph of the equilibrium between gas and iron oxides. The gas normally shows a “thermal reserve zone”, that is, a zone in which the temperature does not change rapidly as well as a “chemical reserve zone”, a zone in which the chemical composition of the gas does not change. The thermal reserve zone decreases and can disappear when the furnace is pushed to high productivities.

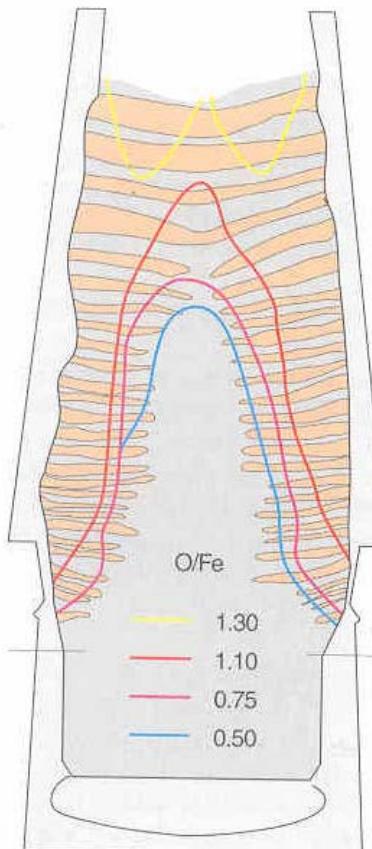


Figure 56: Reduction progress in a quenched furnace (Hirohata, after Omori, 1987, p. 8)

8 – Blast Furnace Productivity and Efficiency

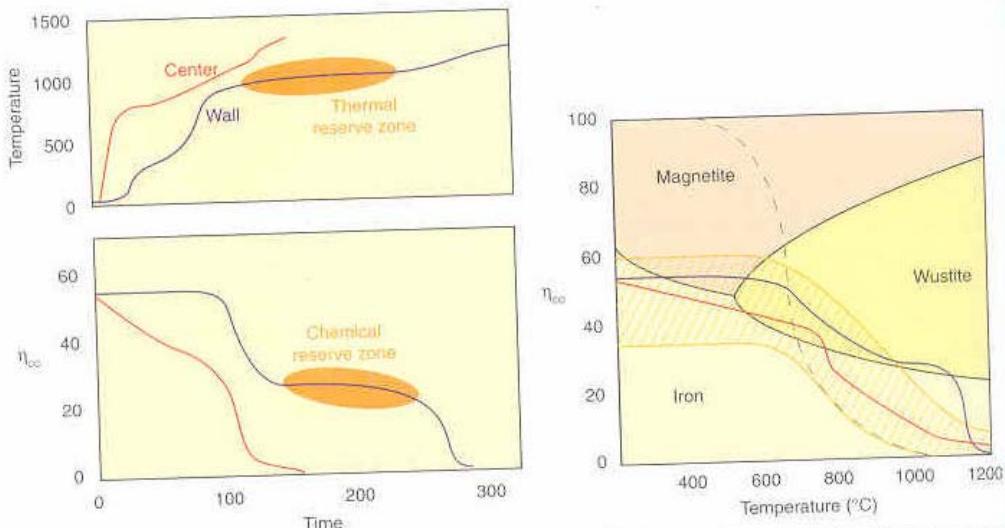
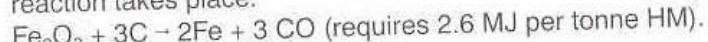


Figure 57: Gas composition in operating furnace. CO, CO₂, H₂ and temperature were measured with descending probes (Chaigneau et al, 2001). Typical measurements from various furnaces are shaded. After McMaster, 2002.

8.2.3 Gas reduction and direct reduction

The direct reduction and gas reduction reaction combine to create a very efficient process. Suppose that all oxygen is removed by direct reduction. Then, the following reaction takes place:

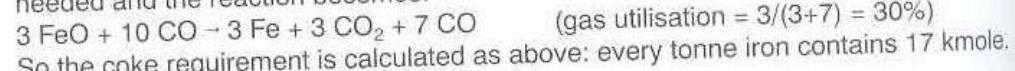


Iron contains about 945 kg Fe per tonne hot metal. Coke contains about 87.5% carbon. Atomic weights of Fe and C are 55.6 and 12 respectively. A tonne of iron contains 17 kmole (945/55.6). For every atom of iron we need 1.5 atoms of carbon, so the carbon requirement is 25.5 kmole (1.5x17), which is 306 kg carbon (25.5x12). In addition, about 45 kg carbon is dissolved in iron. In total, 351 kg carbon is used per tonne hot metal, which corresponds to 401 kg of coke. This is a very low equivalent coke rate and a blast furnace will not work, because the heat generated in this reaction is too low.

Now consider that all reduction reactions are done via the gas reduction, what coke rate is required in this situation? It is assumed that coke combustion generates the CO required. The reaction is:



We only consider the reduction of wustite since the resulting gas is powerful enough to reduce magnetite and hematite. We know from the above (Figure 55) that for gas reduction the maximum gas utilisation is 30%. To get 30% gas utilisation more CO is needed and the reaction becomes:



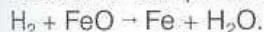
8 – Blast Furnace Productivity and Efficiency

There is a need of 10 atoms carbon per 3 atoms of Fe. So the carbon requirement is 57 kmole ($10/3 \times 17$), which corresponds to 684 kg carbon (57×12). Again, the extra 45 kg carbon in iron has to be added giving a carbon rate of 729 kg/t and a coke rate of 833 kg coke per tonne hot metal ($729/0.875$). This reaction has a poor coke rate and a high heat excess.

The conclusion of the considerations above is, that the counter-current character of the blast furnace works efficiently to reduce the reductant rate by combining direct and gas reduction reaction. Approximately 60–70% of the oxygen is removed by gas and the remaining oxygen is removed by direct reduction.

8.2.4 Hydrogen

Hydrogen is formed from moisture in the blast and injectants in the raceway. Hydrogen can act as a reducing agent to remove oxygen and form water. The reaction is comparable with that for carbon monoxide:



The major differences with the reactions for hydrogen and carbon monoxide are as follows:

- Figure 57 shows the equilibrium of the iron oxides and hydrogen. Hydrogen is more effective for the reduction at temperatures above 900°C. From measurements in the blast furnace it has been derived, that hydrogen reactions are already completed at this temperature.
- Hydrogen utilisation as measured from the top gas is normally around 40% while CO utilisation is close to 50%.

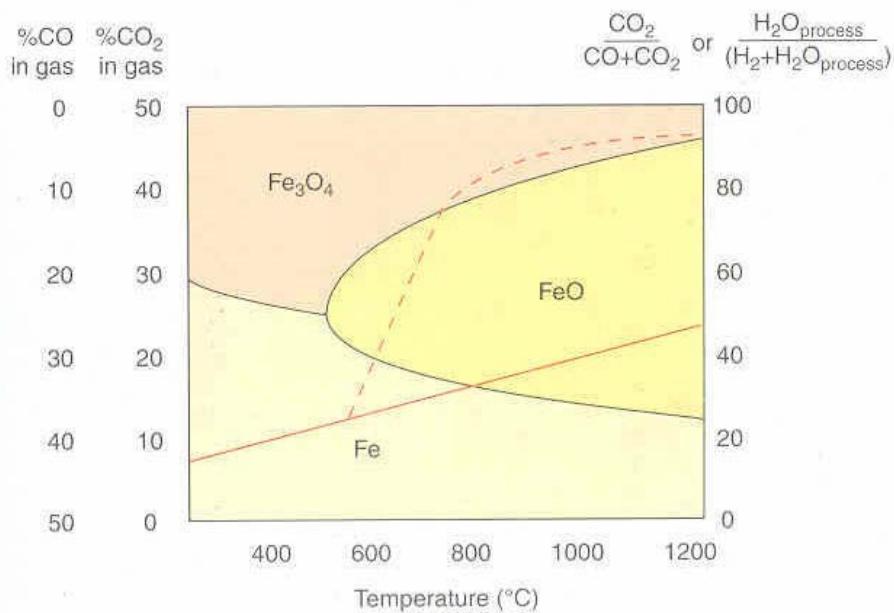


Figure 58: Equilibrium iron oxides with hydrogen

Note that the hydrogen utilisation cannot be measured. The H_2O formed in the process cannot be discriminated from the water put in the furnace with coke and burden moisture.

The hydrogen utilisation of the top gas is defined as $H_2/(H_2+H_2O_{process})$. When working at high hydrogen input (via moisture, natural gas, coal), the competition between the reduction reactions will lead to lower top gas CO_2 utilisation. The simple reasoning is, that H_2 competes with CO . All oxygen taken by H_2 is not taken by CO_2 and thus CO increases and CO_2 decreases. 1% extra H_2 in topgas will lead to 0.6% extra $H_2O_{process}$ in top gas and thus to a 0.6% lower CO_2 and a 0.6% higher CO percentage. 1% extra topgas hydrogen leads to a decrease in topgas CO -utilisation of 1.3%, e.g from 49% to 47.7%. If a more advanced model is used and the efficiency of the furnace is kept constant at the so called FeO level, a 1% increase in topgas hydrogen leads to a decrease of 0.8% in topgas CO -utilisation.

8.3 Temperature profile

The temperature profile and the chemical reactions in a blast furnace are closely related. It is summarised in Figure 59. The reduction of the oxides to wustite takes place at temperatures between 800 and 900 °C. Thereafter, in the temperature range of 900 to 1,100°C, the wustite can be further reduced indirectly without interference from the Boudouard reaction. This chemical preparation zone can take up to 50 to 60% of the height of the furnace and has a relatively constant temperature. This region is called the thermal reserve zone.

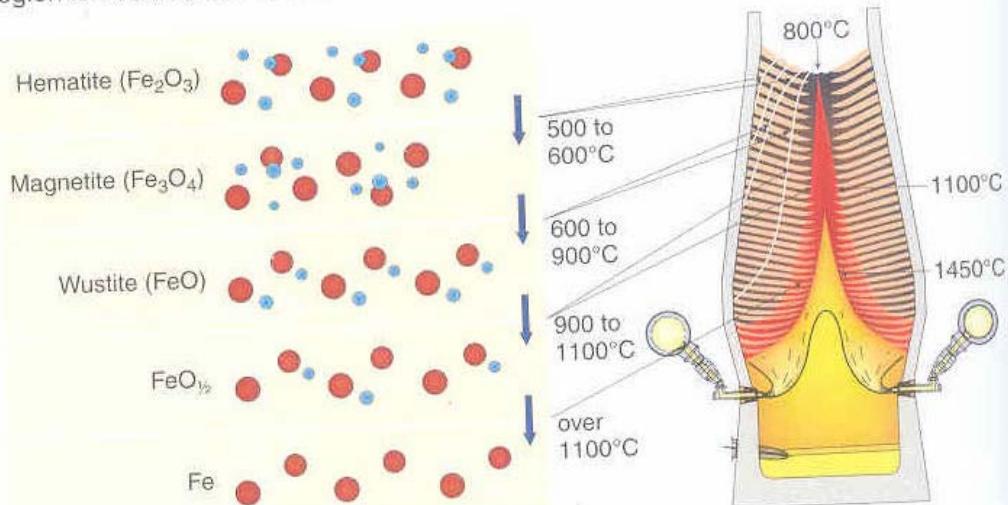


Figure 59: Progress of the reduction reactions and temperature of the burden

8.4 Circumferential symmetry and direct reduction

High performance operation of a blast furnace requires that the complete circumference of the furnace contributes equally to the process. A furnace can be divided into sectors in which every tuyere forms one sector. See Figure 30 for an example.

If all sectors do not contribute equally to the process, asymmetry in the melting zone will arise, as shown in Figure 60. Local heat shortages will drive the melting zone downwards in certain sectors and upwards in other sectors. This can result in an increase in direct reduction in some sectors. Increasing the thermal level of the entire furnace affecting its overall efficiency can only compensate for the effect and not resolve it.

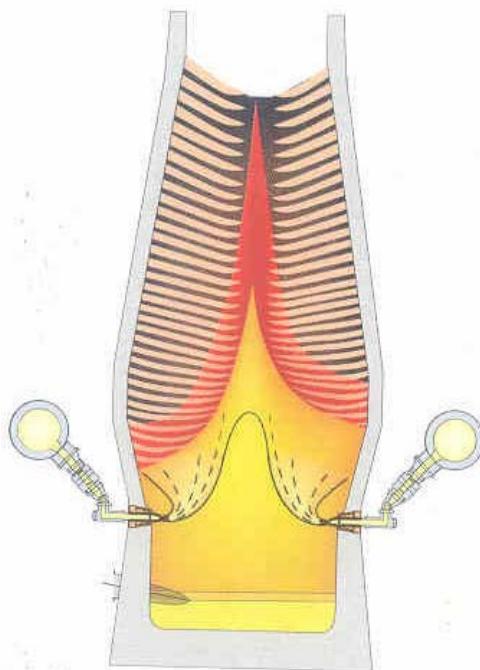


Figure 60: Asymmetric melting zone

Asymmetry in the process can arise from various sources.

- From uneven coal injection. Especially tuyeres without PCI (section 5.6).
- Blast distribution: if the blast speed is too low (under 100 m/s), tuyeres will not efficiently function as blast distributors. This can be observed especially at the tuyeres opposite the inlet between hot blast main and bustle main. Blast distribution can also be effected by plugged tuyeres (above a taphole or refractory hot spots) and slag deposits in the tuyere.
- By asymmetry of the charging. With a bell-less top this can be prevented by alternating the coke and ore top bins and by changing the rotational direction of the chute. With a double bell system it is possible to alternate the last skip in a dump. Note that the changes have to be made on a time scale smaller than the blast furnace process i.e. more frequent than every six hours.

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- Worn refractory or armouring plates at the top of the furnace.
 - Deviation of furnace centre line from vertical line. This is especially a concern in older furnaces.

Measures to correct for deviations of circumferential symmetry are available, such as removing PCI injection from specific tuyeres. However, it is preferred to eliminate the causes of the circumferential asymmetry instead of correcting for it.

Asymmetry in the gas flow can be derived from the radial heat loss distribution. In the figure below, the heat losses are measured in eight segments of the furnace over four vertical sections. Extended asymmetry can be investigated with the help of this type of data and graphs.

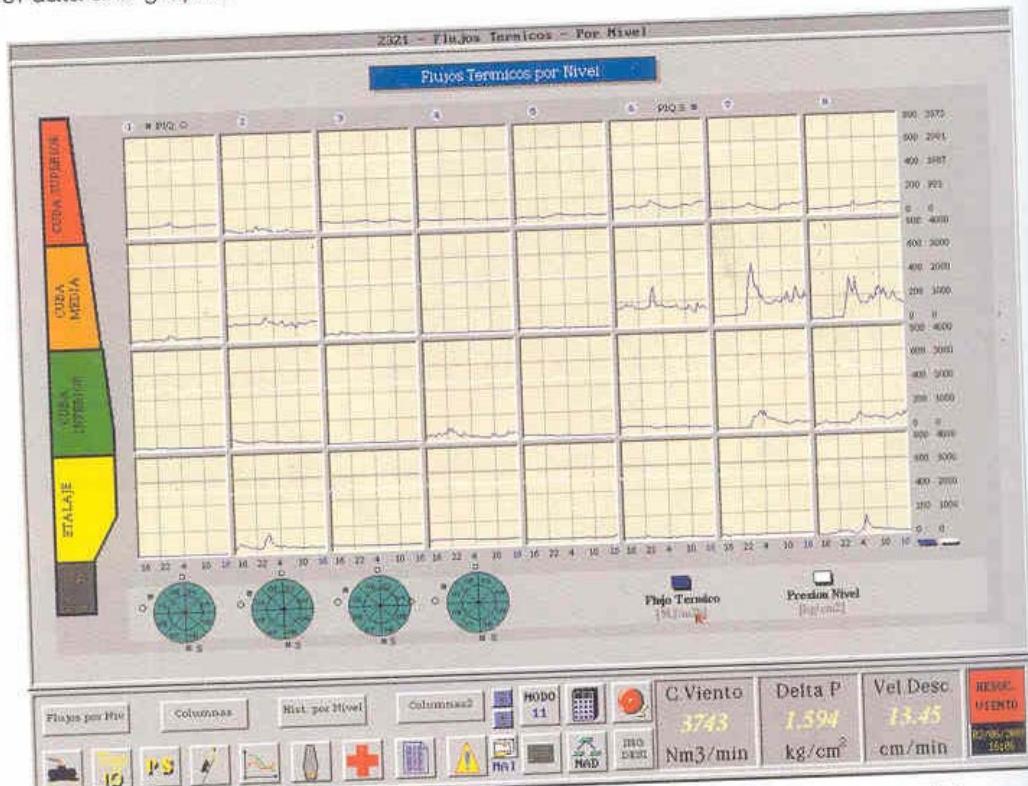


Figure 61: 24 hrs heat loss distribution (blue). Note a slight process asymmetry.
One day graph of eight sections, four levels.

Chapter 9

Hot Metal and Slag

Typical hot metal and slag compositions are given in Table 22. Hot metal leaves the furnace with a temperature typically in the range between 1,480 and 1,520 °C.

Table 22: Typical hot metal and slag composition

Hot metal		Typical	Slag	Typical	Range
Iron	Fe	94.5%	CaO	40%	34–42%
Carbon	C	4.5%	MgO	10%	6–12%
Silicon	Si	0.40%	SiO ₂	36%	28–38%
			Al ₂ O ₃	10%	8–20%
Manganese	Mn	0.30%			
Sulphur	S	0.03%	Sum	96%	
Phosphorous	P	0.07%	Sulphur	1%	

9.1 Hot metal and the steel plant

Hot metal is used for the production of steel. In a steel plant the hot metal is refined so that the (chemical) composition can be adjusted to the metallurgical requirements. The refining process is usually achieved in two steps:

- Removal of sulphur from the hot metal by means of desulphurisation. In most cases the sulphur is removed with carbide and lime (stone) or magnesium, according to:
 - $2 \text{CaO} + 2 [\text{S}] + \text{CaC}_2 \rightarrow 2 (\text{CaS}) + \text{CO}$ (gas) or $\text{Mg} + [\text{S}] \rightarrow (\text{MgS})$. (Square brackets, i.e. [S], show that material is dissolved in the hot metal. Round brackets, i.e. (CaS), show material dissolved in slag.)

9 – Hot Metal and Slag

- Removal of carbon, silicon, manganese and phosphorous. These elements react with the oxygen blown into the converter. The “affinity” for oxygen decreases in the sequence Si>Mn>C>P>Fe. In this sequence material is refined in the converter process. At the end of the refining process iron can be reoxidised, which is sometimes required to heat up the steel before casting. Si, Mn, P and FeO are removed with the slag phase, the C as CO or CO₂ in the gas phase.

The important considerations for a steel plant are:

- Consistent quality: the control of the converter process incorporates “learning”, which adjustments to the process settings are necessary on the basis of expected outcome versus the actual outcome. The more consistent the iron quality, the better the results in the steel plant.
- Hot metal silicon, manganese, titanium and temperature are important energy sources for the converter process and effect the slag formation.
- Hot metal phosphorous has a major influence on steel production process. In the blast furnace 97 to 98% of the phosphorous leaves the furnace with the hot metal.
- Hot metal sulphur is a problem because sulphur is difficult to remove in the converter process. For high grades of steel a maximum sulphur level of 0.008% is required, while the blast furnace produces hot metal with a content of 0.030% and higher. Therefore, an external desulphurisation step is often required.

9.2 Hot metal composition

The final hot metal composition is the result of a complex process of iron–slag interactions as the various elements are divided over the slag and iron phases. The dispersion of an element over the two phases depends on the slag and hot metal composition as well as temperature, as discussed below. As an illustration the typical percentages of elements entering the slag and iron phases are indicated in Table 23.

The following points should be noted:

- Silicon, titanium and sulphur are concentrated in the slag.
- Manganese is concentrated in the hot metal.
- Some of the potassium is discharged from the top.
- Nearly all the phosphorous goes to the hot metal.

Table 23: Typical distributions of selected elements over iron and slag

Element	Input kg/tHM	Output Iron		Output Slag	
		kg/tHM	%	kg/tHM	%
Silicon	46	5	11%	41	89%
Manganese	6	4.5	75%	1.5	25%
Titanium	3	0.7	23%	2.3	77%
Sulphur	3	0.3	10%	2.7	90%
Phosphorous	0.5	0.48	96%	0	0%
Potassium	0.15	0	0%	0.11	73%

9.3 Silicon reduction

Silicon, manganese and phosphorous oxides are reduced via the direct reduction reaction. Out of these three, the silicon reactions are of particular interest. The hot metal silicon is a sensitive indicator of the thermal state of the furnace, and the silicon variation can be used to analyse the consistency of the process. For these reasons the silicon reactions are discussed in more detail. The reduction of silicon takes place via three steps (Figure 62):

- Formation of gaseous SiO in the raceway. The first reduction step takes place at the very high flame temperatures of the raceway. The silicon comes from the ash of the coke (and coal). The higher the coke ash, the higher the silicon in hot metal.
- Further reduction by means of direct reduction with the iron. The SiO gas in contact with the iron can be reduced as follows: $\text{SiO} + [\text{C}] \rightarrow [\text{Si}] + \text{CO}$ (square brackets indicate solution in iron).
- The more intimate the contact between iron and gas, the higher the hot metal silicon content. The higher the height that the iron drips down, the greater is the contact between the hot gasses and the liquid metal, leading to higher hot metal temperatures. The longer contact allows more SiO gas to react with the carbon in the hot metal, leading to higher hot metal silicon content. Therefore, a high-located melting zone corresponds with high hot metal temperature and high hot metal silicon.

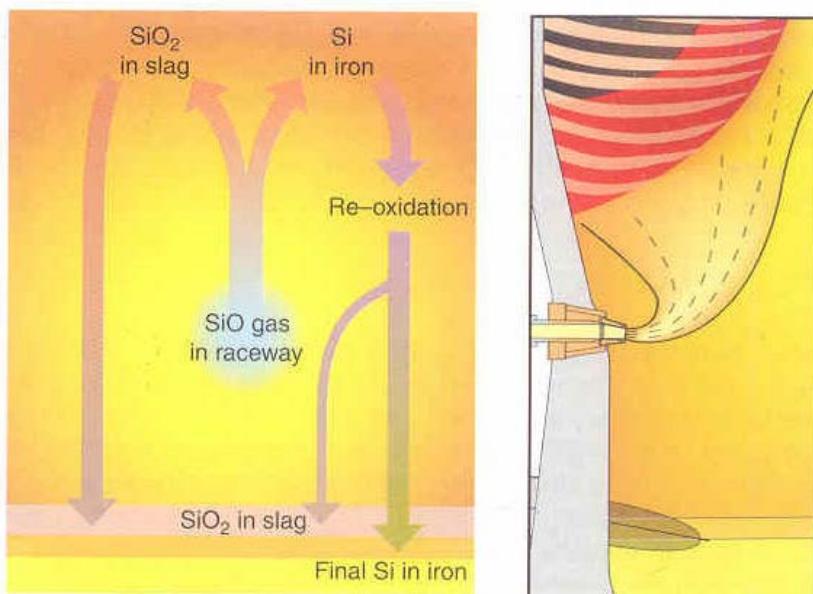


Figure 62: Reactions of silicon in the blast furnace

- The hot metal silicon is in equilibrium with the slag. Important aspects are:
 - When iron droplets descend and pass through the slag layer, the silicon can be reoxidised if FeO is present in the slag, according to:

$$[\text{Si}] + 2 (\text{FeO}) + 2 [\text{C}] \rightarrow (\text{SiO}_2) + 2 [\text{Fe}] + 2 \text{CO}$$
 - The more basic the slag (less SiO_2 in slag), the lower the hot metal silicon.

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- The hot metal formed in the centre has high silicon, while the hot metal formed at the wall has low hot metal silicon. The cast result is an average value.

Hot metal silicon and manganese are both indicators of the thermal state of the furnace. Manganese shows a quicker response on process changes due to the fact that the equilibrium with the remaining slag in the furnace is faster for manganese due to the smaller fraction of manganese in the slag.

9.4 Hot metal sulphur

The hot metal sulphur is governed by the input of sulphur, the slag composition and the thermal state of the furnace. The most important parameters are:

- Sulphur input: the sulphur input is typically 2.5 to 3.5 kg/tHM. The main sources being coke and the auxiliary reductant such as coal or oil.
- The division of sulphur between iron and slag, indicated by the (S)/[S] ratio. This ratio is very sensitive to the slag basicity and the thermal level of the furnace (hot metal temperature or hot metal silicon).
- The slag volume: the lower the slag volume per tonne hot metal, the higher the hot metal sulphur at the same (S)/[S].

Most companies have their own correlations between (S)/[S] and the slag basicity and thermal level. The correlations are derived on the basis of historical data for a blast furnace. As a basic guide: to reduce hot metal sulphur by 5%:

- reduce input by 5%.
- Increase basicity by 0.02 (basicity defined as $\text{CaO}+\text{MgO}/\text{SiO}_2$) or
- Increase hot metal silicon by 0.06%.

9.5 Slag

9.5.1 Slag composition and basicity

Slag is formed from the gangue material of the burden and the ash of the coke and auxiliary reductants. During the process primary slag develops to a final slag. Composition ranges are presented in Table 25 on the next page. Four major components make up about 96% of the slag, these being SiO_2 , MgO , CaO and Al_2O_3 . The balance is made up of components such as manganese (MnO), sulphur (S), titanium (TiO_2), potassium (K_2O), sodium (Na_2O) and phosphorous (P). These components have a tendency to lower the liquidus temperature of the slag. The definitions of basicity are given in Table 24.

Table 24: Definitions of basicity (weight percentage)

B2	CaO/SiO_2
B3	$\text{CaO}+\text{MgO}/\text{SiO}_2$
B4	$(\text{CaO}+\text{MgO})/(\text{SiO}_2+\text{Al}_2\text{O}_3)$

Table 25: Typical slag compositions

	Typical	Range
CaO	40%	34–42%
MgO	10%	6–12%
SiO ₂	36%	28–38%
Al ₂ O ₃	10%	8–20%
Total	96%	96%

9.5.2 Slag properties

Slag has much higher melting temperatures than iron. In practice it is more correct to think in temperature ranges than in melting points, as composite slags have a melting trajectory rather than a melting point. At the solidus temperature the ore burden starts melting. The liquidus temperature is the temperature at which the slag is completely molten. At temperatures below the liquidus temperature solid crystals are present. These solid crystals increase the viscosity of the slag. In our experience the behaviour of slag can be well understood on the basis of its liquidus temperature. Liquidus temperatures are presented in ternary diagrams as shown in Figure 63. These diagrams have been developed for pure components and in practice the liquidus temperatures are somewhat lower. Since in the ternary diagrams only three components can be indicated, one of the major slag components is taken as fixed, i.e. Al₂O₃ content is 10%. Diagrams at different Al₂O₃ percentages are presented in Figure 64 (next page). The typical slag composition for a blast furnace slag is also indicated (Table 24). Note that the liquidus temperature is above 1,400 °C and that the liquidus temperature increases when CaO increases (i.e. when the basicity increases).

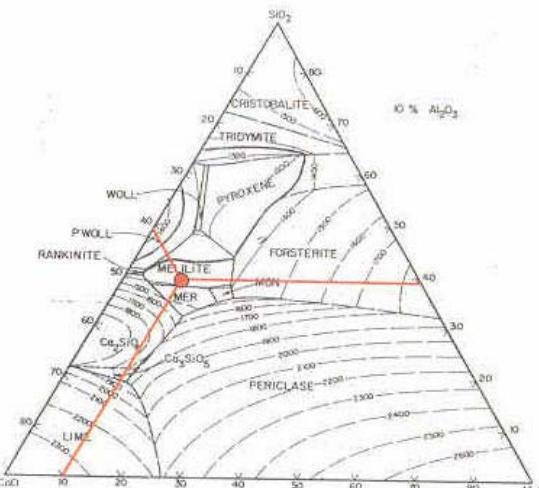


Figure 63: Phase diagram of liquidus temperatures of blast furnace slag system for 10% Al₂O₃. The slag composition CaO=40%, MgO=10% and SiO₂=36% is also indicated. To this end the components have to be recalculated from 96% to 100% of the slag. (After slag atlas, 1981.)

9 – Hot Metal and Slag

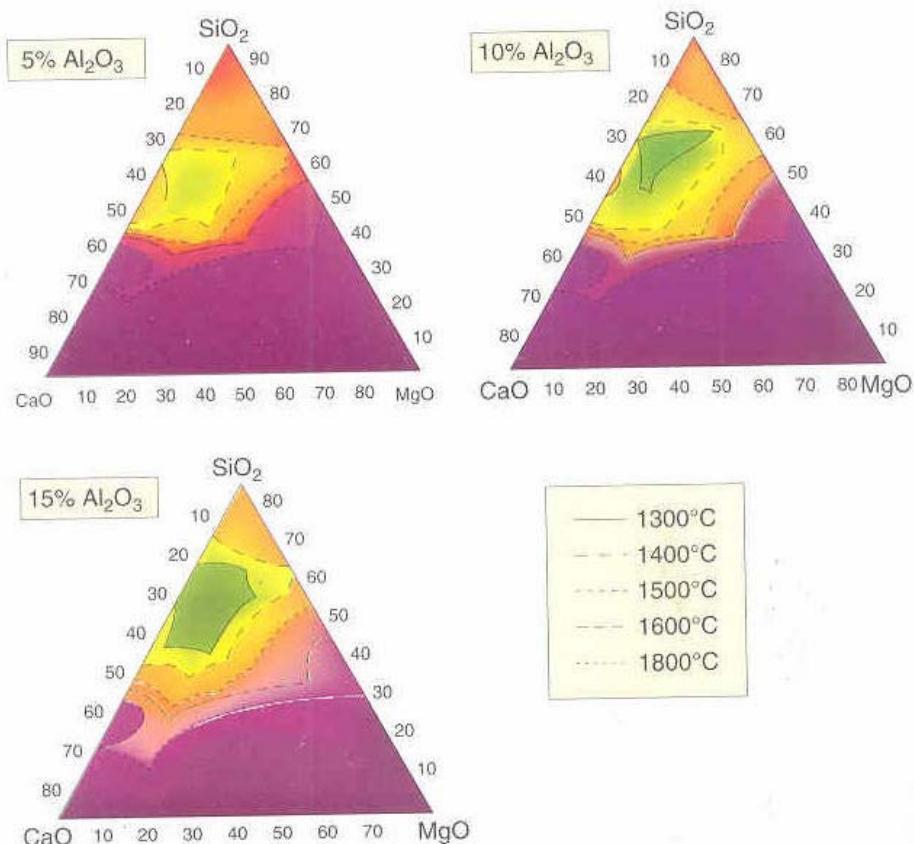


Figure 64: of slag liquidus temperatures at various Al_2O_3 levels. (After slag atlas, 1981.)

In Figure 65 (next page), the composition of the slag resulting from a burden of self fluxed sinter and pellets is indicated. The liquidus temperatures of the "pure" components give high liquidus temperatures for the slag, well above 1,500 °C. How is it possible that the material melts in the cohesive zone?

The secret behind the melting of sinter and pellets is, that the ore burden contains a lot of FeO , which lowers the melting temperature or, as mentioned earlier, lowers the liquidus temperature and solidus temperature. The primary slag, i.e. the slag formed during melting process and prior to solution of the coke ash components into the slag, is made liquid due to dissolved FeO (Figure 19). The final slag is made liquid through the solution of SiO_2 as indicated in Figure 66 (next page).

9 – Hot Metal and Slag

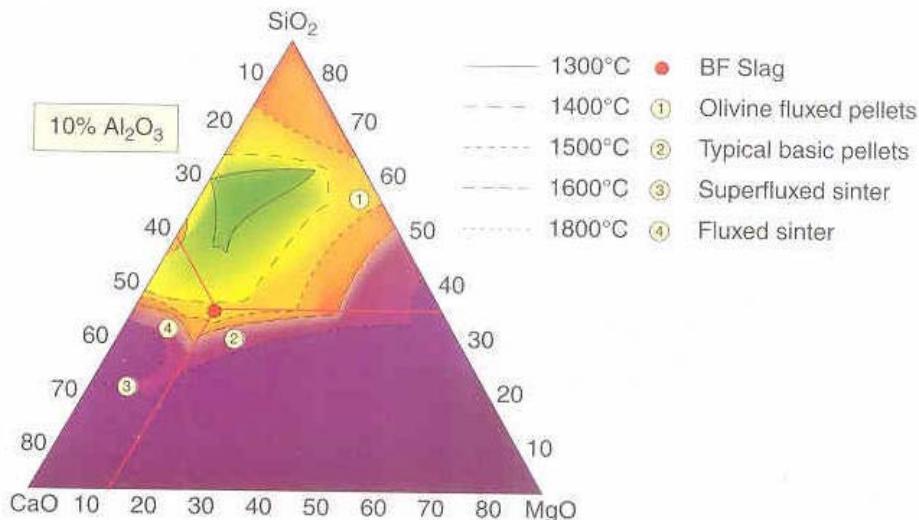


Figure 65: The slag composition of typical pellets and sinter qualities

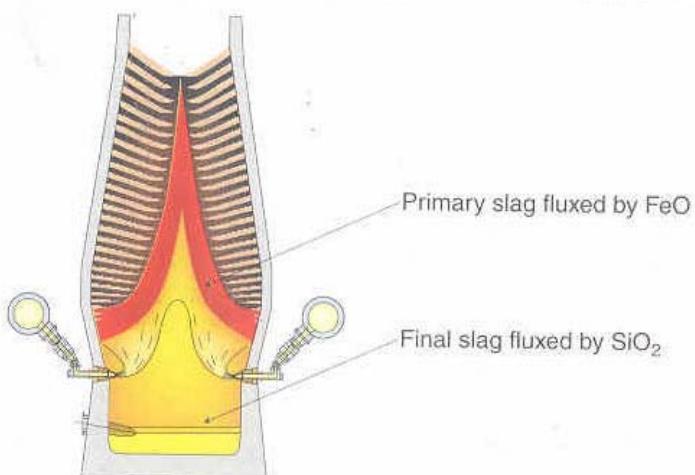


Figure 66: Slag formation

9.6 Hot metal and slag interactions: special situations

During special blast furnace situations like a blow-in or a very hot furnace the hot metal silicon can rise to very high values. Since the silicon in the hot metal is taken from the SiO_2 in the slag, the consequence is that the basicity increases. This leads to high slag liquidus temperature. In a situation with very high basicity the final slag is no longer liquid in the furnace and cannot be cast. It will remain in the furnace where it can form a ring of slag, particularly in the bosh region. Burden descent and casting will be disrupted. Therefore, for special situations where hot metal silicon is expected to be high, the slag should be designed to handle the high hot metal silicon. To this end, extra SiO_2 has to be brought into the furnace and the recommended method is the use of siliceous lump ore.

9 – Hot Metal and Slag

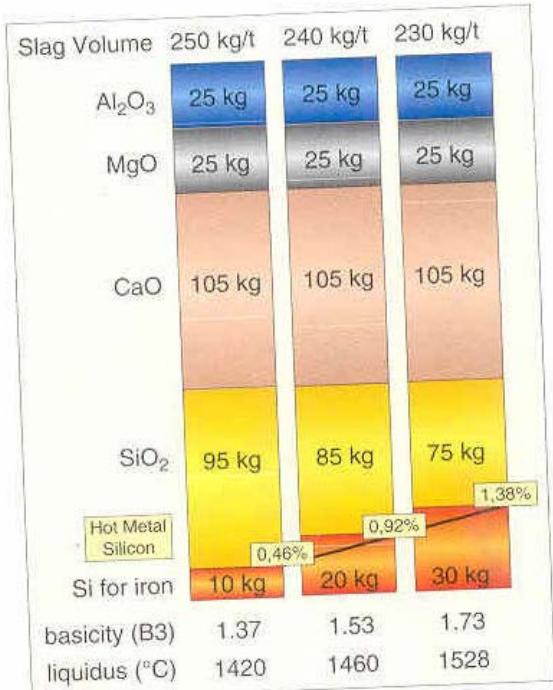


Figure 67: Slag properties under special situations

Some companies use quartzite, which is suitable to correct the basicity in normal operation however, it is not suitable for chilled situations, since the liquidus temperature of quartzite itself is very high (1,700°C). The effect of the use of a siliceous ore can also be shown in the ternary diagram in Figure 68: by working at a lower basicity, the liquidus temperature decreases along the indicated line.

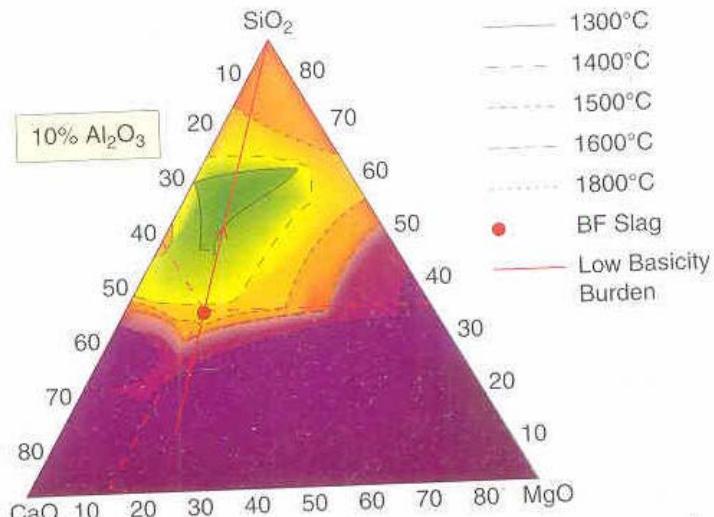


Figure 68: Effect of low basicity burden on slag liquidus temperatures

Chapter 10

Casthouse Operation

10.1 Objectives of casthouse operation

Excellent operational results from a blast furnace can only be reached if three conditions are met:

- Good burden and coke quality.
- Good gas flow control.
- Good casting of the furnace.

The present chapter deals with good casting practice.

The liquid iron and slag collect in the furnace hearth well below the tuyeres. Iron and slag do not mix: slag has a lower specific weight (2.4 t/m^3) than iron (7.2 t/m^3) and "floats" on the iron. The implication of this is that droplets of iron pass through a layer of slag. Iron and slag come close to thermal and chemical equilibrium. A schematic presentation of a hearth and a taphole are presented in Figure 69 (next page). The taphole is a refractory construction and on the inside a refractory "mushroom" consists of solidified taphole clay.

A blast furnace is tapped 8 to 14 times per day through a taphole. The average duration of a cast is 90 to 180 minutes. In this time the furnace produces a considerable part of its working volume. As shown in chapter 2, the residence time of the burden is 5 to 6 hours. Therefore, for a two hour cast about one third of the contents of the furnace is transformed into molten iron and slag.

Spraying of the liquids due to gas from the raceway escaping from the taphole indicates the end of a cast.

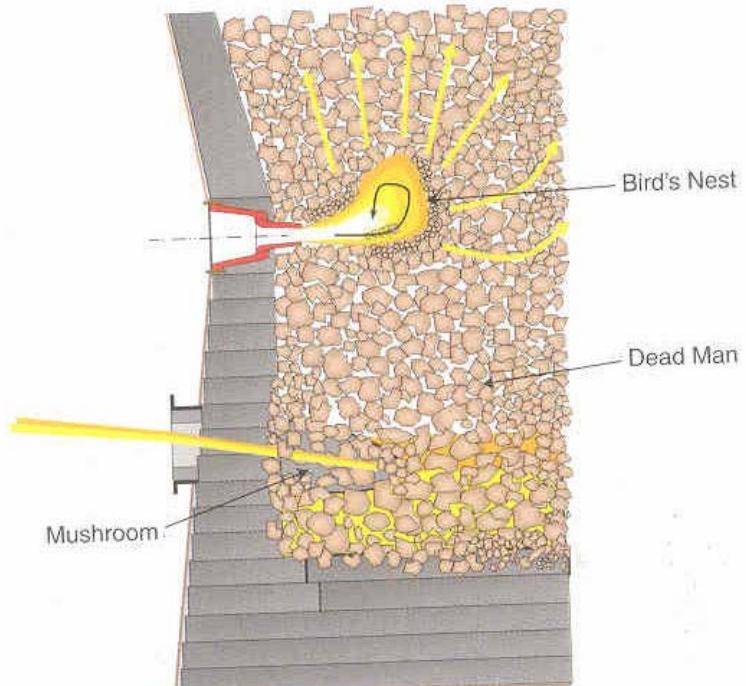


Figure 69: Hearth

The liquid level in the hearth has two major effects on the blast furnace process:

- The liquids in the hearth effect the burden descent: the higher the liquid level, the stronger the upward force from the submerged coke (Figure 35).
- If slag reaches the level of the tuyeres and cannot be drained out through the taphole, the gas flow is severely affected. This can result in poor reduction of the burden and therefore a chilling furnace.
- Slag can be blown high up in the dead man, impeding normal gas distribution.

In order to prevent the effects, the hearth liquid level has to be kept under control and preferably at a constant and low level (see De Pagter and Molenaar, 2001). The objective of casthouse operation is to get the liquids from the furnace without interfering with the blast furnace process.

Note that the storage capacity of the hearth depends on the void fraction of the coke in the hearth. The void fraction can be very low; with values of 25% void fraction possible. In static areas of coke in the hearth the void fraction can be even lower.

10.2 Casthouse layout

A modern blast furnace has at least two tapholes, with furnaces as big as 14 metres hearth diameter equipped with up to four tapholes. A schematic layout of a casthouse is shown in Figure 70, including a layout of the runner system. The iron is cast into the main iron runner, or trough. Iron and slag can be separated easily because they do not mix due to their difference in specific weight. Figure 71 shows the iron and slag flows through the main trough to a skimmer, which allows the iron to flow

10 – Casthouse Operation

through, but diverts the slag to the slag runner. The slag is then usually granulated by water or dumped into slag pits. The iron is collected into torpedoes. Two torpedoes are located at each iron runner and can be filled using a tilting runner, which allows the operator to exchange a torpedo during a cast.

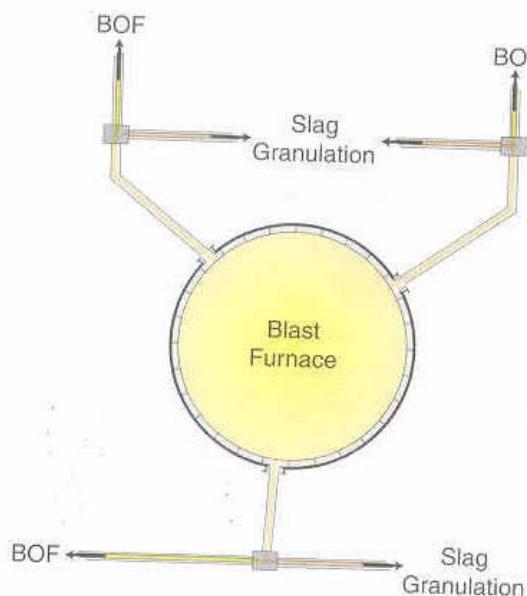


Figure 70: Example of a casthouse layout (three tapholes)

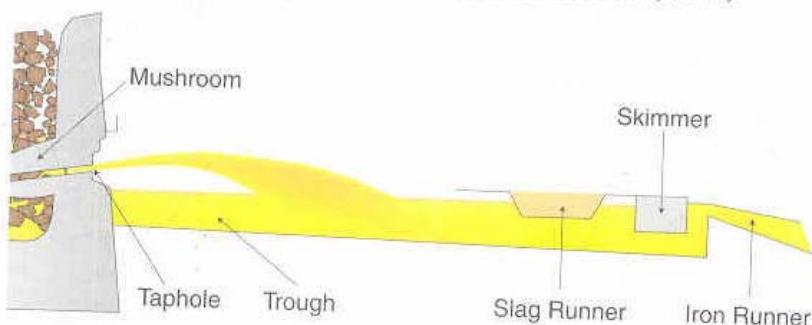


Figure 71: Iron/slag separation

10.3 Dry hearth practice

Most high productivity furnaces cast through alternating tapholes, i.e., when one taphole is plugged, the other taphole is opened. With this practise the gap time between the casts can be reduced to as low as zero, where continuous tapping becomes possible. The advantage of the method is that the liquid level in the hearth can be kept consistently at a low level and interference with the process is avoided. Be it by incident or by purpose, sometimes only one taphole is available and the liquids are cast via one taphole. In this situation there will be a gap time between the casts of 20 to 45 minutes. This time is needed to clean the taphole area and for

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curing of the taphole clay. As a consequence of 1-taphole operation the liquid level within the furnace will vary much more (section 10.5.1). Control of taphole length in this situation is more difficult. There appears to be a maximum iron production that can be cast with 1-taphole operation. It is estimated to be 6,000 t/d for an 11 metre furnace and 8000 t/d for a 14 metre furnace.

The “ideal” casthouse operation for a big, high productivity furnace is continuous, alternating casts with very similar cast times (10 casts/day) and almost continuous slag flow. Slag coverage should be more than 95% of the time the iron is cast. See Figure 72.

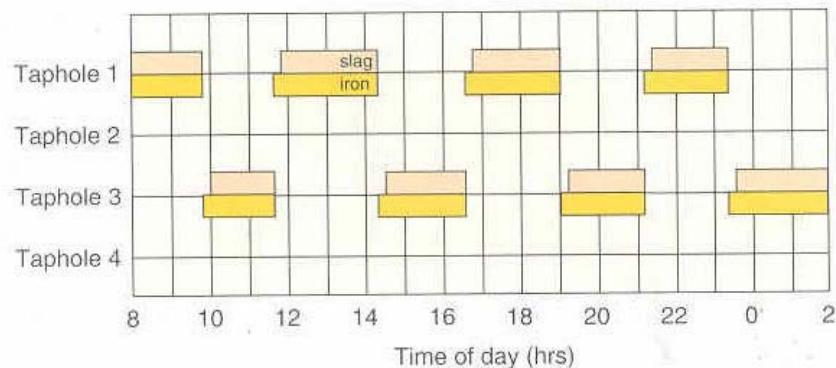


Figure 72: Casting presentation for a 4 taphole furnace, where tapholes 1 and 3 are operated alternately without gaps

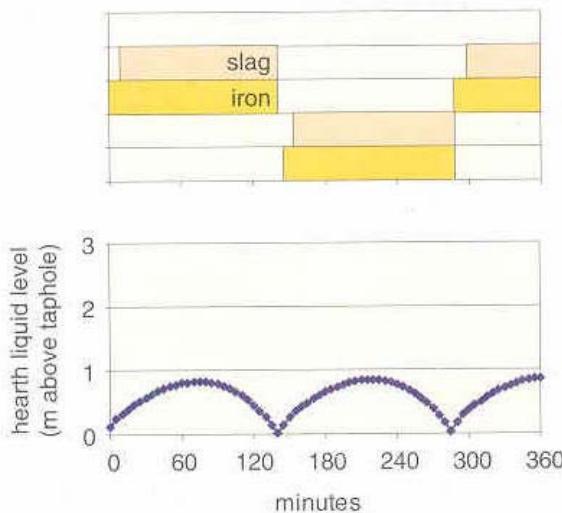


Figure 73: Hearth liquid level variation (in meter) for alternating casts.

Alternating casts and 1-taphole operation give different variations in the liquid level in the hearth, as shown in Figure 73.

10.4 Opening and plugging the taphole

10.4.1 Iron and slag flow

The iron and slag flow from a taphole are determined by the characteristics of the taphole i.e.:

- The length of the taphole, which is affected by the way of plugging and clay quality.
- The diameter of the taphole and especially the wear of the taphole over time.
- The roughness of the inner surface of the taphole.
- The pressure inside, consisting of the blast pressure and the "hydrostatic" pressure.

Initially, during a cast, the liquid flow is lower than the production rate, thus the liquid level in the hearth moves upwards. As soon as the slag is tapped, the taphole starts to wear out and the liquid flow will be higher than the production rate. At the beginning of a cast only iron might be cast. Even with good casting there is a variation in the hearth liquid level of up to one metre inside the furnace. The taphole clay quality determines the resistance to attack of slag and appropriate choice of clay has to be made. This is not discussed here.

10.4.2 Opening the taphole

Two methods for opening the taphole are applied: drilling a taphole open and soaking bar practice. With soaking bar practice a bar is put into the clay just after plugging and is pulled out 20 to 45 minutes later in order to open the taphole. The advantage is that opening the taphole is very fast and thought to be reliable. It is especially suitable for single taphole operations. Many companies have tried the soaking bar technique, but the majority uses the drilling method. The soaking bar technique has a high impact on the taphole bricks and blocks.

Good drilling has to fulfil the following requirements:

- The drill position should be reproducible and at the same position: the centre of the taphole. To this end the drill should be automatically secured on the furnace.
- Drill diameter: a selection of drill diameters should be available (40–60 mm). The diameter of the hole should be constant over its complete length. To this end the drill should be guided and the forward force, the rotating speed and hammering should be properly controlled. Wear of the drill bit should be minimised, which can be achieved by using nitrogen or nitrogen/water for purging and cooling during drilling.
- The length of the taphole: drilling directly into the hot metal bath should open the furnace. It is counterproductive to try to save a drill bit and rod by stopping drilling earlier.

In some situations drilling cannot open the taphole. In this case, manual oxygen lancing is applied. Lancing should be the last resort, since reproducibility is poor and there are risks for taphole damage and break out.

10.4.3 Plugging the taphole and taphole length

Clay quality and proper plugging are very important for the length of the taphole and for the flow rates of iron and slag. Plugging has to meet the following requirements:

- Plugging has to be done on the same position as the drill in order to avoid clay spillage. Clay spillage may lead to a "secondary" tap stream.
- The speed of the piston and the pressure on the clay has a direct effect on the proper injection of the clay into the hole.
- When using tar-bonded clay the curing time should be sufficiently long to ensure proper clay conditioning. To this end the clay gun has to be positioned and pressurised in front of the taphole for 15 to 30 minutes.
- The length of the taphole is determined by the quantity of clay injected. Therefore, more clay is injected than required to fill the taphole. The excess clay is used to form the mushroom at the inside of the taphole. The bigger the furnace, the longer the taphole i.e. a 2.5 metre taphole length for an 11 metre furnace and 3 metres for a 14 metre furnace.
- Gas tightness. After drilling the hole, the taphole clay should be sufficiently gas tight to prevent spraying of the casts. Re-pressurising the taphole clay directly after plugging might help to improve gas tightness.

10.5 Hearth liquid level and casthouse operation

10.5.1 The effect of 1-side casting and alternating casts compared

In Figure 75, the effect of the casthouse operation on hearth liquid level is compared for alternate casts and 1-side casting. Since at 1-side casting there is a gap time of 30 to 40 minutes between the casts, the hearth liquid level rises much more than in a situation with alternate casts. The calculation made for 1-side casting shows, that the liquid level might reach 2 to 3 metres above the taphole. This has also an effect on the descending speed of the burden: it is not unusual, that at the end of a cast the burden descent and thus charging rate increases. The reason is that the upward force from the liquids in the hearth decreases, see Figure 35.

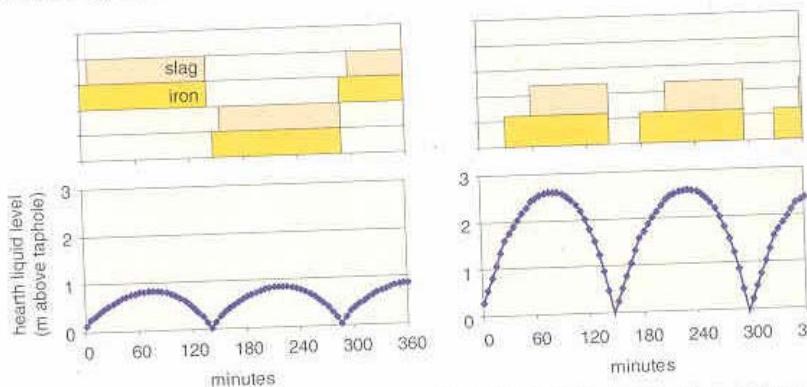


Figure 75: Liquid level in hearth compared for alternating casts (left) and 1-side casting (right)

10.5.2 Casthouse delays

The consequences of casthouse delays can be very severe.

First, if starting the cast is delayed—say delay times of 45 to 75 minutes—the burden descent is effected: the higher upward force prevents the burden descending (Figure 35). The slower burden descent will be followed by a period with faster descent, during which there will be less heat available to melt the burden and the consequence is that hot metal temperature and silicon will decrease. From an operational point of view it is important to be able to keep the casthouse always under control. It is recommended to decrease production level at an early stage to prevent more severe effects.

Second, if the casting is delayed beyond the moment that slag reaches the tuyere level, the slag will be pushed inwards and will block part of the gas flow. Consequently, if the furnace was operating with central gas flow, the gas flow pattern will be redistributed towards the wall. The moment when slag reaches the tuyeres is visible from a gradual increase of blast pressure/burden resistance or decrease of permeability index. In this situation the chilling of the furnace will be more severe than above.

The very bad situation can arise where even the iron can reach the tuyere level. If that is the case, iron will flow through the tuyeres into the blowpipes and a blowpipe will burn causing an emergency stop of the furnace.

10.6 Exercise

Suppose the cast is dry and we cannot open the next cast. Suppose we continue the production at full blast. What is the time needed to fill the hearth from taphole to tuyeres?

Use the following data:

- The furnace produces iron (6.7 t/min or 400 t/hr) and slag (100 t/hr).
- Hot metal weighs 7.2 t/m^3 , slag weighs 2.4 t/m^3 .
- Voidage is 20%.
- Hearth diameter: 14 m.
- Distance taphole–tuyeres: 4.2 m.

Chapter 11

Special Situations

11.1 Stops and start-ups

When a blast furnace in full operation is stopped, some of the processes continue. While the blast is stopped, the direct reduction reactions within the furnace continue as well as heat losses to the wall. The consequence is that the temperature of the material in the melting zone is reduced to around 1,050 to 1,100°C, which is the start of the carbon solution loss reaction. The decreasing temperature re-solidifies the melting materials. Therefore, after a stop it takes some time for the burden to start descending. The burden descent restarts as soon as the "old" melting zone is molten (Figure 76).

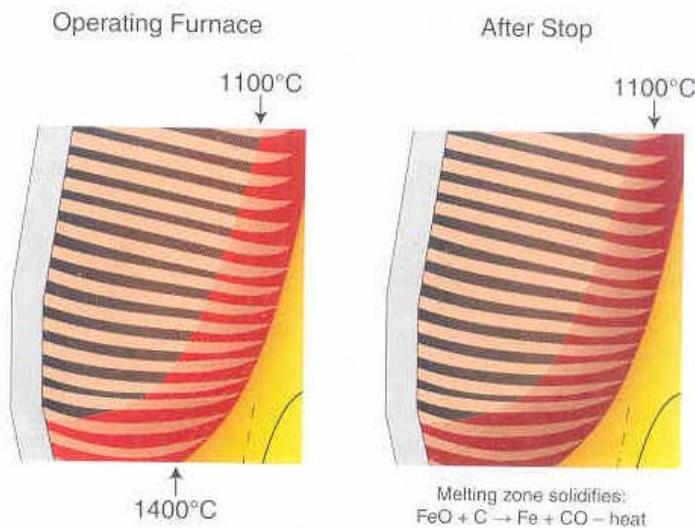


Figure 76: Solidified melting zone as consequence of a stop

The heat shortage for a stop of a furnace operating with PCI is even worse: during the stop procedure the coal injection is switched off from the furnace and during the start-up it takes time to restart the PCI. An additional reductant shortage results.

In addition, after a stop the hot metal silicon sometimes rises to very high values, especially if during the stop/start procedure the furnace is operated at a low blast volume. As shown in Figure 68, the basicity of the slag will be affected by the high hot metal silicon and might even solidify within the furnace. This results in disturbed burden descent. Heating up the slag is the only solution, which can be achieved by charging coke blanks into the furnace.

So, in order to compensate for the heat losses during a stop and the risk for high hot metal silicon, the following measures have to be applied:

- Extra reductant into the furnace. Coke as well as auxiliary reductants are possible. Additional reductant is needed for a period that the furnace is not operated on PCI.
- Design slag composition for low basicity at high hot metal silicon. Use of a siliceous lump ore is recommended.

Even if a stop is unplanned, taking these measures after the stop is worthwhile.

For a blow-in after a stop major pitfalls are:

- Too fast blow-in. The solidified melting zone will take time to melt during the start-up. If allowed time is insufficient, the pressure difference over the burden can increase too much, leading to gas escaping along the wall (high heat losses) and poor burden descent.
- Too fast restart of the PCI. Since the melting zone is solidified, there is a risk that solid agglomerates will block the hot blast through the tuyere. If this happens, the coal will still be blown into the blowpipe where it can cause blowpipe failure. It is recommended to restart coal injection only when the burden starts descending.
- Too high slag basicity.

11.2 Blow-down

Blowing down a blast furnace requires operating the furnace without simultaneous charging of the furnace. Therefore, all material charged into the furnace is exposed to the same temperatures and reduction processes as if the furnace was fully charged.

However, since the temperature of the shaft gas is not transferred to the cold charge, the off-gas temperatures increases and the gas composition changes. Since the equipment has not been designed to withstand the high top gas temperatures, the top gas temperatures are kept under control by spraying water. The water sprayed above the burden should be prevented from reaching the burden surface, either directly via descent on top of the burden or indirectly via the wall. Special atomising nozzles are required and the success of the blow-down heavily depends on proper spraying. The progress of the blow-down process can be measured from the burden level as well as from the analysis of the top gas composition. Since less and less oxygen is removed from the ore, the CO₂ percentage decreases and CO percentage increases (Figure 77, next page).

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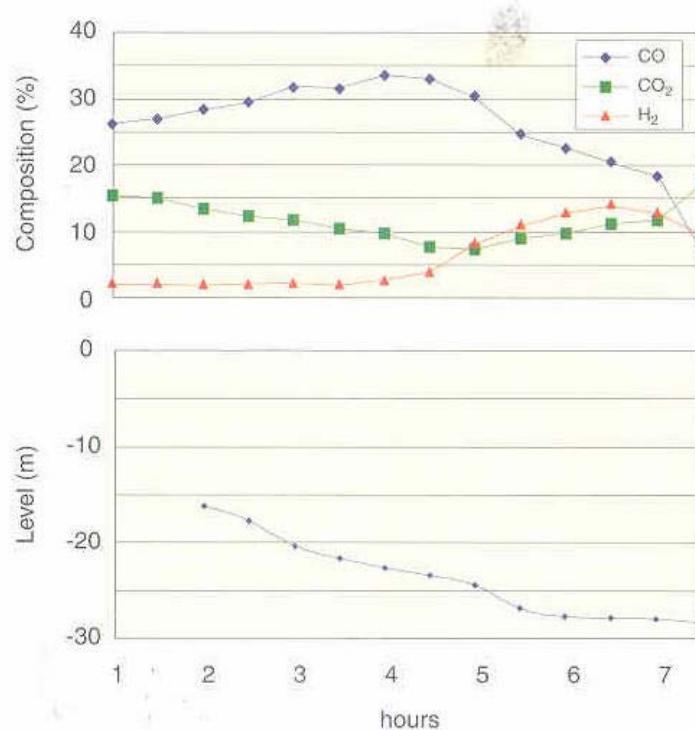


Figure 77: Typical progress of a blow-down

Moreover, generally H₂ increases as a consequence of the (unavoidable) contact of spraying water with the hot coke. At the end of the blow-down, when the level of the coke is coming close to the tuyeres, the CO₂ formed at the tuyeres has insufficient opportunity to be transformed to CO and the CO₂ percentage in the top gas increases. As soon as half of the oxygen is in CO₂ (i.e. when the CO₂ percentage equals half the CO percentage), the furnace should be isolated from the gas system.

Normally, a blow-down takes 10 to 12 hours, after a preparatory stop, to reach the tuyere level.

Prior to the blow-down the furnace contains coke in the active coke zone and dead man, and alternating layers of coke and ore in melting zone and stack zone. Since during the blow down the coke of the active coke zone and dead man will be gasified, there is coke excess in the blast furnace. During the latter stages of the blow down reduction reactions have largely stopped, so any auxiliary reductant injection can be stopped during the early stages of the blow down. The moment is indicated by the gas analysis: as soon as the CO₂ percentage starts to decrease to below 10%, then there is little iron left to reduce.

The burden level in the furnace is difficult to measure with standard stock rods. The stock rods have to be equipped with chain extensions and recalibrated for the purpose. The stock rods should be used only at intervals, since the high

temperatures above the burden may cause chain breakage. Radar level indicators can be used if reliable. Indications from the level of the burden can also be obtained from:

- The pressure taps.
- The casthouse operation i.e. the quantity of iron cast.
- Calculation of the amount of coke consumed in front of the tuyeres.

The required condition of the furnace after the blow-down depends on the purpose of the blow-down and consequent repair. Generally the walls have to be clean. Cleaning of the hearth is another important topic. If solid skulls and scabs are expected in the hearth and have to be removed prior to the blow-down, the furnace can be operated for a prolonged period on a high thermal level, relatively low PCI rate and without titanium addition. The effect of these measures is, however, uncertain.

11.3 Blow-in from new

Blowing in a furnace from new can be considered in two phases:

Phase 1: Heating-up the hearth.

Phase 2: Starting the reduction reactions and iron production.

The two phases are discussed separately below.

11.3.1 Heating up the hearth

Heat is generated by the reaction of coke carbon to CO. Coke generates 55 kJ per mole carbon, when reacting to CO, which corresponds to 3.9 MJ/t coke.

The heat requirement in the early stages of the blow-in is for the following:

- Heat coke in the hearth, dead man and active coke zone to 1500°C.
- Heat required for evaporation of moisture from the coke.
- Heat required to compensate for moisture in blast dissociating into hydrogen gas ($H_2O + C \rightarrow CO + H_2$).
- Heat to compensate for loss of heat through the wall.

A detailed analysis of the heat requirement to fill the hearth, dead man and active coke zone with coke of 1500°C indicates the following:

- Moisture in the coke can be neglected.
- The heat required filling the hearth, dead man and active coke zone with hot coke of 1500°C requires an amount of coke gasified to CO of about two-thirds of the estimated volume of the hearth/dead man/active coke zone.
- Additional heat requirement arises from the water dissociation reaction and the heat losses through the wall. For example, if 300 tonne coke is required to fill hearth, dead man and active coke zone with coke, a coke blank is required with a total weight of 600 tonne: 300 tonne to fill hearth, dead man and active coke zone with coke and 300 tonne for the generation of heat to bring the coke to 1500°C.

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- In the early stages of a blow-in, blast temperature should be maximised and blast moisture minimised.
- Heating up the hearth requires some 7 to 8 hours after the blow-in. Heat is generated from coke used at the tuyeres.

11.3.2 Starting the reduction processes

During the early stages of the blow-in while the hearth is heating up, the reduction of the iron oxides has not yet begun due to the temperatures being too low. Therefore, one has to consider the increased amount of direct reduction. The situation may become difficult if the level of direct reduction is too high, (and gas reduction is low). This situation manifests itself from:

- The gas utilisation.
- The direct reduction, as manifest from $\text{CO} + \text{CO}_2$ exceeds "normal" values.

The gas utilisation is an indication of the amount of gas reduction taking place, while the total CO and CO₂ percentage is an indication for the direct reduction. Especially the CO₂ percentage indicates if gas reduction takes place.

11.3.3 Slag formation

In general, the slag during blow-in has to be designed for high hot metal silicon. However, with the proposed method the hot metal silicon should be under control. If we continue to follow the "two-phase" blow-in approach mentioned here, during the first phase of the blow-in about 350 tonne coke is gasified in 8 hours and the slag formed comes only from the coke ash. Taking 10% ash and 30% of the ash as Al₂O₃, we get during the first 8 hours 35 tonne of a high Al₂O₃ slag. This will not cause a problem in the furnace because of the small volume. The coke ash can be diluted, e.g. by using a high siliceous ore in the coke blank. In order to dilute to 20% Al₂O₃, some 30 tonne of a siliceous ore has to be added to the 350 tonne coke blank.

11.3.4 Hot metal quality during blow-in

As soon as the hearth is heated the hot metal temperature exceeds 1400 °C. As soon as the top temperature exceeds dew-point, all excess moisture has been removed from the furnace and the process has started. There is only limited heat required for heating-up and drying of refractories, if compared with the heat requirements of the process itself. So as soon as hot metal temperature reaches 1400 °C and top temperature exceeds 90 °C, the process has to be brought back to normal operation conditions.

However, in this situation the coke rate in the furnace is still very high and the hot metal silicon will rise to 4 to 5%. The hot metal silicon can be reduced by putting a normal coke rate in the furnace. The "normal" coke rate at "all coke" operation is about 520 kg/tHM. In doing so, however, it takes considerable time to consume all excess coke, which is present in the furnace. More rapid decrease of hot metal

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silicon can be reached, if a lower coke rate is charged and auxiliary injection is used as soon as required. The injectant is switched on, as soon as the hot metal silicon decreases below 1%.

An example of such a rapid blow-in of a furnace is presented in Figure 78. At the blow-in the furnace was started-up with eight tuyeres (of 36). After opening all tuyeres, a "heavy" burden (coke rate 450 kg/tHM) was put in the furnace 50 hours after the blow-in and coal injection was put on the furnace 58 hours after the blow-in. Hot metal silicon reached the 1% mark 60 hours after the blow-in. The 4th day after the blow-in, average hot metal silicon was 0.95% and productivity was 2.1 t/m³ WV/d.

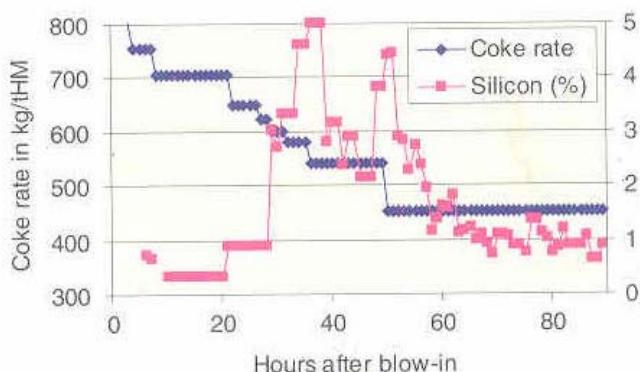


Figure 78: Charged coke rate and hot metal silicon after blow-in

Annexes

Annex I Further reading

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Annex III Answers to exercises

1.4 Raw material flows

10,000 t/d gives about 3.5 million tonne per year (at an availability of 96%). So total world production of 575.7 million ton can be produced in 165 furnaces. With three of these furnaces per site, there would be a worldwide need for 55 of these large ironmaking sites, each producing 10.5 million tonne.

The resulting annual flow of materials would be:

- Steel output: $0.9 \times 10.5 = 9.45$ million tonnes steel products per year.
- Pellets required $1.6 \times 10.5 = 16.8$ million tonnes.
- Coal required:
 - For cokemaking: $10.5 \times 0.3 / 0.75 = 4.2$ million tonnes
 - For PCI: $10.5 \times 0.2 = 2.1$ million tonnes

2.3.1 How much blast oxygen is used per tonne hot metal?

Oxygen from the blast volume amounts to $0.256 \times 6,500 \text{ m}^3 \text{ STP/min} = 1,664 \text{ m}^3 \text{ STP oxygen/min}$. The production rate is $10,000 / (24 \times 60) = 6.94 \text{ tHM/min}$.
So the oxygen use is $1,664 / 6.94 = 240 \text{ m}^3 \text{ STP blast oxygen/tHM}$.

2.3.2 How often are the furnace contents replaced?

To produce a tonne of hot metal, the furnace is charged with:

- 300 kg coke: 0.64 m^3 (300/470)
- 1,580 kg sinter/pellets: 0.88 m^3 (1,580/1,800)
- Total per tonne of hot metal: 1.52 m^3

Production is 10,000 tonne per day, which is $10,000 \times 1.52 \text{ m}^3 = 15,200 \text{ m}^3$ per day.
This material can be contained in the working volume of the furnace, with exception of the volume used for the active coke zone. So the contents of the furnace are refreshed 4.6 times per day ($15,200 / (3,800 - 500)$). This means the burden charged at the top reaches the tuyeres in 5.2 hours.

2.3.3 How many layers of ore are in the furnace at any moment?

The number of ore layers depends on the layer thickness or the weight of one layer in the burden. It can vary from furnace to furnace and depends on the type of burden used so there is a large variety of appropriate burden thicknesses. A typical range is 90–95 tonne of burden per layer. A layer contains 94.8 tonne, so about 60 tonne hot metal. In 5.2 hours, the furnace produces 2,167 tonne, which corresponds to 36 layers of ore ($2,167 / 60$). In our example, taking a throat diameter of 10 m, the ore layer is 67 cm and the coke layer is an average of 49 cm at the throat.

Annexes

2.3.4 What happens to the carbon of the coke and coal?

One tonne of HM requires:

- 300 kg coke, C content 87%: 261 kg C
- 200 kg coal, C content 78%: 160 kg C
- Total carbon: 417 kg C

About 45 kg carbon dissolves in the hot metal. The balance leaves the furnace through the top, which is $417 - 45 = 372$ kg. It leaves the furnace as CO and CO_2 .

2.3.5 How much top gas do we get?

Production per minute is $10,000 / (24 \times 60) = 6.94$ tonne HM/min. For a tonne hot metal $6,500 / 6.94 = 936$ m³ STP blast is used, which weighs 1,218 kg. The balance is presented in Table 26. The weight of the top gas is the known inputs minus the output of iron and slag. Total top gas weight is twice the weight of the iron!

Table 26: Weight balance example

Input	In kg/tHM	Output	In kg/tHM
Blast	1,218	Iron	1,000
Coke	300	Slag	240
Coal	200	Top gas	2,058
Burden	1,580		
	3,298		3,298

The raceway gas volume is $8,164$ m³ STP/min ($6,500 \times (1 + 0.256)$), and the top gas volume is $9,987$ m³ STP/min ($2,058 / 1.43 = 1,439$ m³/tHM). So the gas from direct reduction reactions is approximately $1,800$ m³ STP/min ($9,987 - 8,164$).

2.3.6 Estimate how long the gas remains in the furnace

The blast volume is $6,500$ m³ STP/min with 25.6% oxygen. Since for every unit of oxygen two units of CO are produced, the raceway gas amounts to $6,500 \times (1 + 0.256) = 8,164$ m³ STP. This gas has a higher temperature (decreasing from some $2,200^\circ\text{C}$ to 125°C top gas temperature), the furnace is operated at a higher pressure (say 4.8 bar, absolute at the tuyeres and 3 bar, absolute at the top) and extra gas is formed by the direct reduction reaction (see exercise 2.3.5). If all these effects are neglected, the exercise is straightforward:

Suppose the void fraction in the burden is 30%, then the open volume in the furnace is $(3,800 - 100) \times 0.30 = 1,100$ m³ STP, through which $8,164$ m³ STP gas is blown per minute. So the residence time of the gas is $(1,100 / 8,164) \times 60 = 8$ seconds.

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It is possible to make the corrections mentioned above. Take an average temperature of the gas of 900°C and an average pressure of 4 bar, and then the effects are:

- Increase in residence time owing to higher pressure: $4/1 = 4$ times longer.
- Decrease in residence time owing to higher temperature $273/(273+900) = 0.23$ times shorter.
- Decrease in residence time due to extra gas from direct reduction is $8,164/9,987 = 0.82$ times shorter.
- In total, the residence time is shorter by a factor of 0.75 ($4 \times 0.23 \times 0.82$), so the corrected residence time is $8 \times 0.75 = 6$ seconds.

2.3.7 If you get so much top gas, is there a strong wind in the furnace?

No, at the tuyeres there are high wind velocities (over 200 m/sec), but top gas volume is about 9,970 m³ STP/min. Over the diameter of the throat, at a gas temperature of 120°C and a top pressure of 2 bar, top gas velocity is 1,0 m/sec: on the Beaufort scale this corresponds to a wind velocity of 1. Through the voids the velocity is about 3 m/s. Note, that in the centre the velocity can be much higher, so that even fluidisation limits can be reached (See 7.4).

10.6 Hearth liquid level

The production of liquid is:

- Iron: $6.7 \text{ t/min} = 400 \text{ t/hr} = 400/7.2 \text{ m}^3/\text{hr} = 89 \text{ m}^3/\text{hr}$.
- Slag: $0.25 \times 6.7 \text{ t/min} = 100 \text{ t/hr} = 100/2.4 \text{ m}^3/\text{hr} = 42 \text{ m}^3/\text{hr}$

So in total, the liquid formed per hour (no casting) amounts to 131 m³/hr.

The amount of liquid that can be stored in the hearth:

- The hearth surface is $3.14/4 \times 14 \times 14 = 154 \text{ m}^2$.
- Voidage is 20% and height 4.2 meter. Between taphole and tuyeres we can store $0.2 \times 4.2 \times 154 = 129 \text{ m}^3$ STP of liquid, which is sufficient for 1 hour of production without casting!

Annexes

Annex IV Rules of Thumb

Table 27: Rules of thumb for daily operation of the blast furnace process, a typical example

	Unit	Change	Coke Rate Adj. (kg/t)
Si	%	+ 0.1	+ 4
Moisture	g/m ³ STP	+ 10	+ 6
Top pressure	bar	+ 0.1	- 1.2
Coal	kg/t	+ 10	- 9
Oil	kg/t	+ 10	- 11
Oxygen	%	+ 1	+ 1
Blast temperature	°C	+100	- 9
Slag	kg/t	+ 10	+ 0.5
Cooling losses	GJ/hr	+ 10	+ 1.2
Gas Utilization	%	+ 1	- 7

Table 28: Rules of thumb for daily operation of the blast furnace process (constant blast volume)

	Unit	Change	Flame temp. (°C)	Top temp. (°C)
Blast temp.	°C	+100	+ 65	- 15
Coal	kg/t	+ 10	- 30	+ 9
Oxygen	%	+ 1	+ 45	- 15
Moisture	g/m ³ STP	+ 10	- 50	+ 9

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