

**Modern
Blast Furnace
Ironmaking**
an introduction

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Third Edition, 2015

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Preface

The third edition of “Modern Blast Furnace Ironmaking” was prepared by an international staff of experts in blast furnace ironmaking. We gratefully acknowledge the contributions of Mr. Cor van der Vliet and Mr. Hisko Toxopeus, who co-authored the first two editions of the book and agreed to hand over future editions of the book to a next generation. Hisko Toxopeus asked to see the revised edition, but he passed away before publication.

The objective of the book is to share our insights that optimization of the blast furnace is not only based on “best practice transfer”, but also requires conceptual understanding of why a measure works in some cases and not in other cases. In other words, operational improvement is not only based on know-how, but also on know-why.

Compared to the second edition we have included gas injection as well as coal-gas co-injection, expert systems, more elaborate descriptions of alkali cycles, the deadman and operational challenges.

We are indebted to many colleagues we have worked with. We are grateful to Jennifer Wise-Alexander and Tim Vander, who find the major part of their contributions to the second edition in chapters 10 and 4 respectively. Jim Plooij shared with us his insights and benchmarking data. Edo Engel did the editing and developed the colorful and insight-giving illustrations.

We learn by sharing our knowledge. We wish the same to our readers.

Maarten Geerdes, Rénard Chaigneau, Ivan Kurunov,
Oscar Lingiardi and John Ricketts

February 2015

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Contents

Preface	v
Contents	vi
List of Symbols and Abbreviations	x
Chapter I Introduction of the Blast Furnace Process	1
1.1 Global steel consumption	1
1.2 Short description of the process	2
1.3 The layer structure within a furnace	5
1.4 The equipment	7
1.5 Book overview	12
Chapter II The Blast Furnace: Contents and Gas Flow	15
2.1 The generation of gas and gas flow through the burden	15
2.2 Furnace efficiency	20
2.3 Removal of liquids	20
2.4 An example of gas flow and contents of a blast furnace	21
Chapter III The Ferrous Burden: Sinter, Pellets, Lump Ore	25
3.1 Introduction	25
3.2 Iron ore	26
3.3 Quality demands for the blast furnace burden	28
3.4 Sinter	32
3.5 Pellets	35
3.6 Lump ore	39
3.7 Metallics charge and briquettes	39
3.8 Interaction of burden components	41
3.9 Chemical control of the burden	41
Chapter IV Coke	43
4.1 Introduction: function of coke in the blast furnace	43
4.2 Coal blends for cokemaking	45
4.3 Coke quality concept	46
4.4 Coke size distribution	50
4.5 Strength of coke	51
4.6 Coke deadman	54
4.7 Overview of international quality parameters	57

Chapter V	Injection of Coal, Oil and Gas	59
5.1	Properties of coal, oil and gas	59
5.2	Coal injection	62
5.3	Natural gas injection	70
5.4	Coal–gas co-injection	73
5.5	Complete combustion of injectants	75
Chapter VI	Burden Calculation and Mass Balances	77
6.1	Introduction	77
6.2	Burden calculation: starting points	77
6.3	An example of a burden calculation	78
6.4	Process calculations: a simplified mass balance	79
6.5	Estimation of direct reduction	83
Chapter VII	The Process: Burden Descent and Gas Flow Control	85
7.1	Burden descent: where is voidage created?	85
7.2	Burden descent: system of vertical forces	87
7.3	Gas flow in the blast furnace	89
7.4	Fluidization and channelling	96
7.5	Burden distribution	96
7.6	Coke layer	101
7.7	Ore layer thickness	103
7.8	Blast furnace instrumentation	106
7.9	Blast furnace daily operational control	106
Chapter VIII	Blast Furnace Productivity and Efficiency	109
8.1	Productivity	110
8.2	Efficiency	111
8.3	Following the gas in the furnace	113
Chapter IX	Hot Metal and Slag	127
9.1	Formation of hot metal and slag	127
9.2	Hot metal as cast from the furnace	132
9.3	Hot metal and the steel plant	137
9.4	Slag	138
Chapter X	Casthouse Operation	145
10.1	Objectives	145
10.2	Liquid iron and slag in the hearth	145
10.3	Removal of liquids through the taphole	147
10.4	Typical casting regimes	148
10.5	Taphole drill and clay gun	150
10.6	Hearth liquid level	151
10.7	Delayed casting	153
10.8	No slag casting	154
10.9	One-side casting	156
10.10	Not dry casts	157

10.11	Defining a dry hearth	159
10.12	Oxygen lancing	160
10.13	Cast data recording	160
Chapter XI	Operational Practices and Challenges	161
11.1	The burden	161
11.2	Burden descent	166
11.3	Recirculation of alkali and zinc	169
11.4	Circumferential symmetry	172
11.5	Tuyeres	176
11.6	Stops and starts	180
11.7	Casthouse challenges	187
11.8	Greenhouse gas emissions	189
Annex I	Glossary	191
Annex II	Further reading	193
Annex III	Starting point for calculation examples	194
Annex IV	Rules of thumb	195
Annex V	Coal types used for coke making	196
Annex VI	Coke quality tests	198
Annex VII	Expert systems and models	202
Annex VIII	Rist diagram	208
Annex IX	References	213
Index		215

List of Symbols and Abbreviations

B2, B3, B4	basicity, ratio of two, three or four components
CRI	coke reactivity index
CSR	coke strength after reaction
daN	Decanewton (approximately 1.02 kgf)
HGI	hard grove index
HMS	harmonic mean size
HOSIM	hoogovens simulatie (blast furnace simulation)
HV	high volatile
ISO	International Organisation for Standardization
JIS	Japanese Industrial Standard
LV	low volatile
PCI	pulverized coal injection
RAFT	raceway adiabatic flame temperature
RR	replacement ratio
Standard Coke	coke with 87.5 % carbon
STP	standard temperature and pressure
tHM	tonne hot metal
VDEh	Verein Deutscher Eisenhüttenleute
VM	volatile matter

I *Introduction of the Blast Furnace Process*

1.1 Global steel consumption

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 (fuel) sources and sinter, pellets and lump ore 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 is the primary source for worldwide steel production: about 70% of the steel consumed is produced via this route, as shown in Figure 1.1.

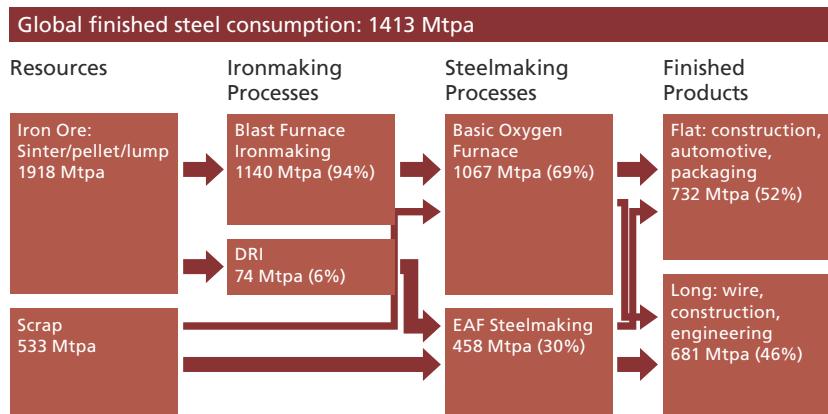


Figure 1.1 Steelmaking routes and raw materials (Steelconsult International, based on data in the IISI Steel Statistical Yearbook and World Steel in Figures, 2012 — Corex and Open Hearth omitted)

From a geographic perspective, the world has observed major changes in steel consumption. Figure 1.2 shows the regional steel consumption in 2012. Steel consumption in China amounted to 646 million tons in 2012, while in 2000 it was 124 million tons. World production capacity has increased even more, since China has become a major steel exporter.

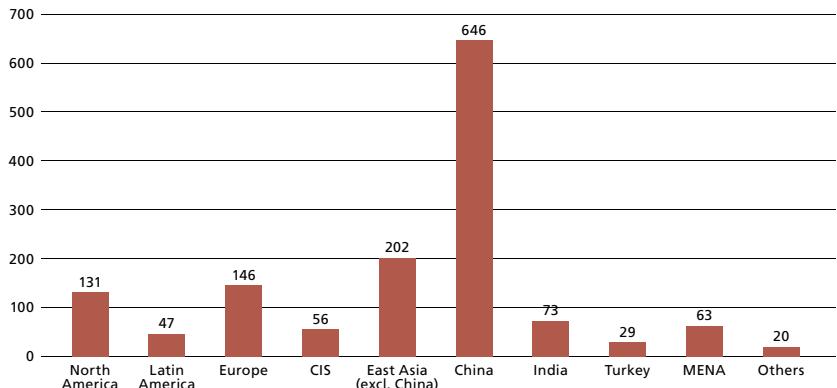


Figure 1.2 Geographical distribution of steel consumption

The present book discusses blast furnace ironmaking. 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.

In the blast furnace process, iron ore in the form of lump ore, sinter and pellets is transformed into hot metal consuming “reducing agents” (coke, coal and natural gas). 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.

In the introductory chapter, we discuss the basics of the process and the internal layer structure in the furnace as well as some major aspects of furnace design, like the profile of the blast furnace and the cooling systems.

1.2 Short description of the process

The inputs and outputs of the furnace are given in Figure 1.3. A blast furnace is filled with alternating layers of coke and iron ore-containing burden. Hot blast is compressed air that is blown into the blast furnace via tuyeres. A tuyere is a cooled copper conical nozzle, numbering up to 12 in smaller furnaces, and up to 42 in bigger furnaces, through which pre-heated air (1000–1300 °C) is blown into the furnace.

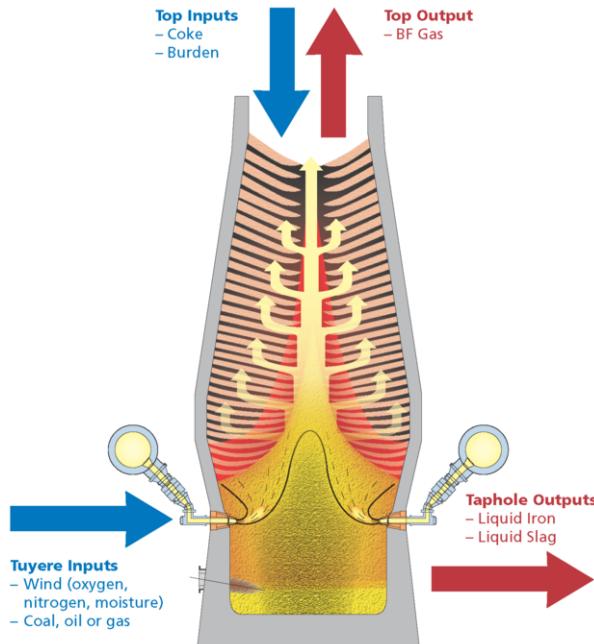


Figure 1.3 Input and output of a blast furnace

This hot blast gasifies coke and other carbon-based materials injected via the tuyeres, mostly coal, natural gas and/or oil. In this process, the oxygen in the blast is transformed into gaseous carbon monoxide. The resulting gas has a high flame temperature of between 1900 and 2300 °C. Coke in front of the tuyeres is consumed, thus creating voidage. The driving forces in the blast furnace are illustrated in Figure 1.4 on the next page.

The very hot gas ascends through the furnace, carrying out a number of vital functions as listed below:

- Heating up the coke in the bosh/belly area.
- Melting the iron ore in the burden, creating voidage.
- Heating up the material in the shaft zone of the furnace.
- Removing oxygen of the iron ore burden by chemical reactions.
- Upon melting, the iron ore produces hot metal and slag, which drips down through the coke into 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 carburization.

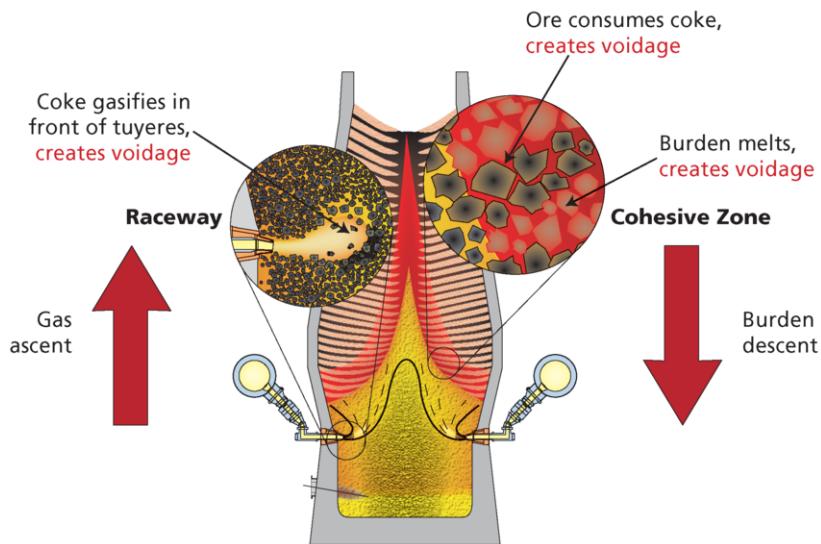


Figure 1.4 The driving force of a blast furnace: the counter-current process creates voidage at the indicated areas causing the burden to descend

A typical example of the temperature profile in the blast furnace is shown in Figure 1.5. It is shown that the softening/melting zone is located in an area where temperatures are between 1000 and 1350 °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 VI.

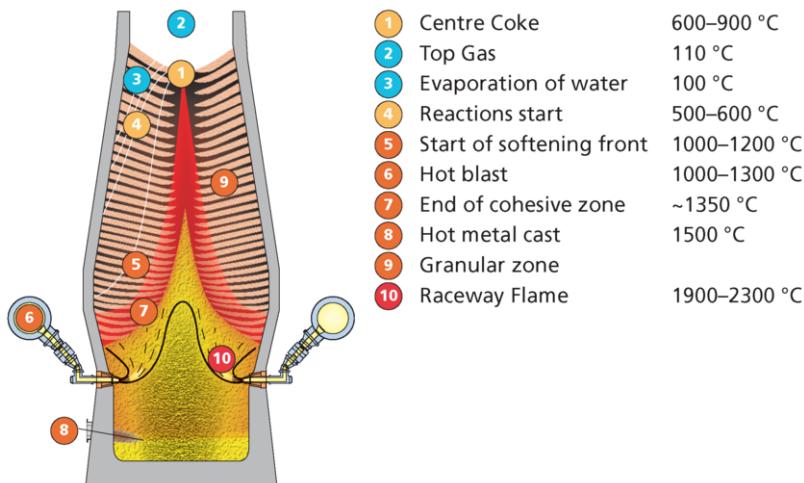


Figure 1.5 Temperature profile in a blast furnace (typical example)

1.3 The layer structure within a furnace

Ore and coke are charged in discrete layers at the top of the furnace and processed more downwards. The structure of layers is maintained throughout the furnace until all ferrous burden materials have been molten. This is derived 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 as well as a solidified cohesive zone are presented in Figures 1.6a and 1.6b.

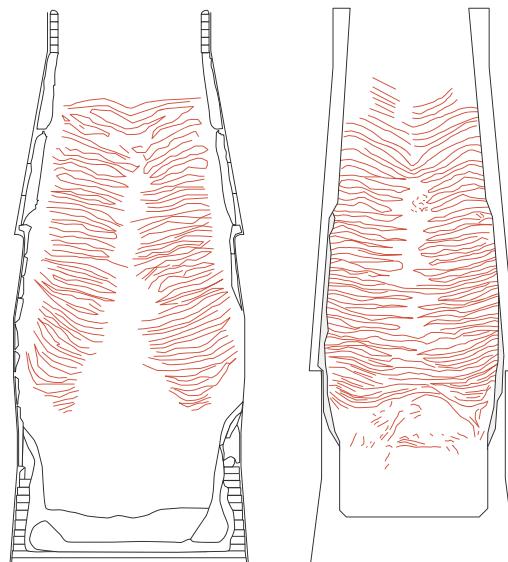


Figure 1.6a Dissections of quenched blast furnaces Kakogawa 1 and Tsurumi
(Based on Omori et al, 1987)



Figure 1.6b Cohesive zone after blow-down

The quenched blast furnace clearly shows 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.

As indicated in Figure 1.7, 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 or cohesive zone.
- An area where there is only coke and liquid iron and slag, called the “active coke” or dripping zone.
- The deadman or “inactive coke zone”, which is a stable pile of coke in the hearth, extending into the bosh of the furnace.

A blast furnace has the shape of two truncated cones that are joined at their widest points. The sections are, from the top down:

- The throat, where the burden surface is located.
- The stack, where the ores are heated and reduction reactions start.
- The bosh parallel or belly.
- The bosh, where the reduction is completed and the ores are melted down.
- The hearth, where the molten material is collected and cast via the taphole. The liquid iron and slag are collected in the voids between coke particles in the hearth. The solid coke is pushed down to the bottom of the furnace by the weight of the charge.

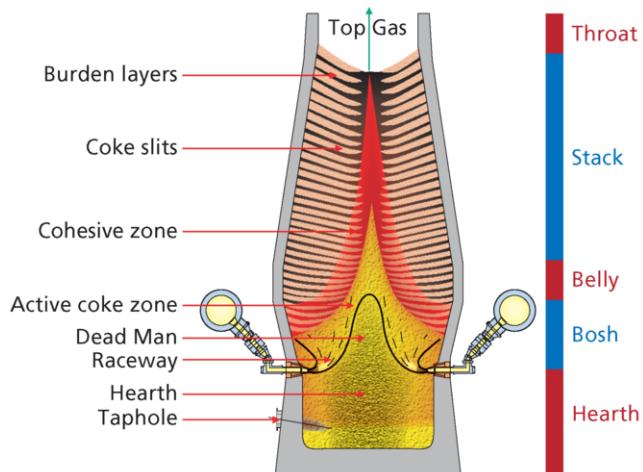


Figure 1.7 The zones in the blast furnace

1.4 The equipment

1.4.1 Equipment overview

An overview of the major equipment is shown in Figure 1.8. These include:

- Hot Blast Stoves. Air preheated to temperatures between 1000 and 1300 °C is produced in the hot blast stoves and is delivered to the furnace via a hot blast main, bustle pipe, tuyere stocks and finally through the tuyeres. The hot blast reacts with coke and injectants. The high speed gas forms the area known as the raceway in front of the tuyeres.
- Stock house. The burden materials and coke are delivered to a stock house. The materials are stored, screened and then weighed 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 skip cars or via a conveyor belt, where they are charged into the furnace in separate layers of ore and coke.
- Gas cleaning. The top gas leaves the furnace via uptakes and a downcomer. The hot top gas contains a substantial quantity of fine particles, so it is required to remove these particles and cool the gas in a gas cleaning system consisting of a dust catcher or cyclone, a scrubber for wet cleaning of the gas and finally a demister for removing the water.
- Casthouse. The liquid iron and slag collect in the hearth of the furnace, from which they are tapped via the taphole into the casthouse and to transport ladles. Depending on the size of the furnace, there may be one to five tapholes and one or two casthouses.
- Slag granulation. The molten slag may be quenched with water to form granulated slag, which is used for cement manufacturing.

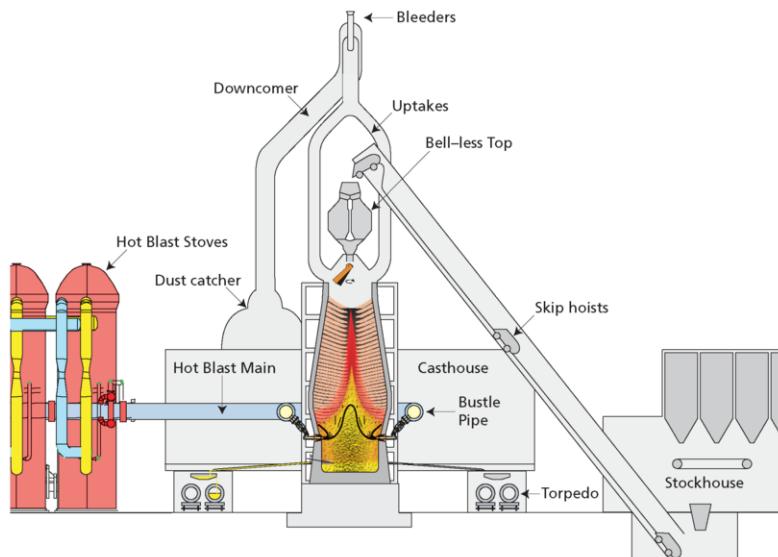


Figure 1.8 Blast furnace general arrangement

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

- The most common system is the bell-less top, which allows burden distribution with a rotating and tilting chute.
- The double bell system was the predecessor of the bell less top. It is often equipped with a movable throat armour.
- Newly developed systems, such as the “Totem-top” and “No–bell top”.

Examples of various types are schematically shown in Figure 1.9.

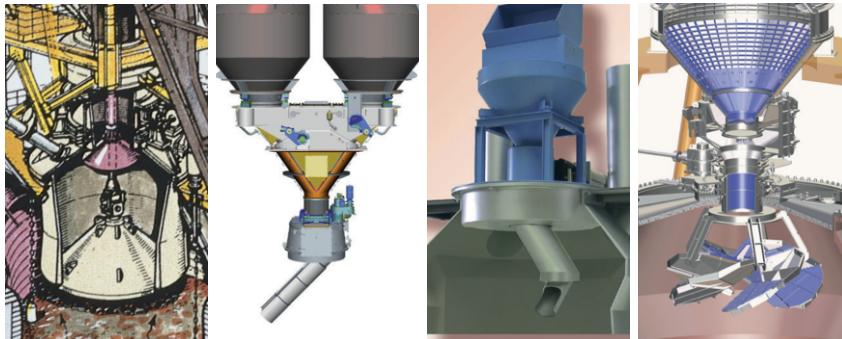


Figure 1.9 Selection of blast furnace top charging systems, from left to right
 Bell-type charging (“McKee Top”)
 Chute-type charging (“Paul Wurth Bell-Less Top”)
 Double Chute-type charging (“Zimmermann & Jansen No-Bell Top”)
 Rotor-type charging (“Totem Top”)
 Pictures taken from company websites.

1.4.2 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 section 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 compensator allowing for expansion (thermal as well as from the pressure) is below the lintel for the lintel furnace, while in the freestanding furnace it is located at the top, as indicated in Figure 1.10.

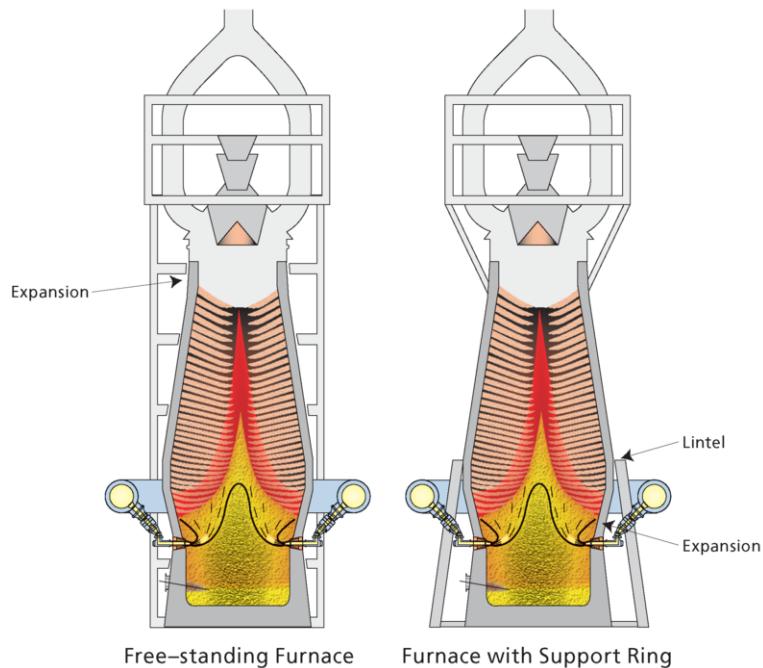


Figure 1.10 Blast furnace constructions

1.4.3 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 hearth diameters between 14 and 15 meters and were producing 3 to 4 million tonnes per year.

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 10–15 % of the ferrous burden or lower. The reductants used developed as well: from all coke operation to the use of injectants through the tuyeres. In the 1960s, oil was the prevalent tuyere injectant, while since the early 1980s coal injection is used extensively. Presently, about 30–40 % of the earlier coke requirements have been replaced by injection of coal and sometimes oil or natural gas. As a result of a new technology to access natural gas deposits, the cost of natural gas has decreased in some areas of the world and now natural gas has become 20–25% of the total reductant. Currently, the co-injection of coal and natural gas is being optimized in the blast furnace.

The size of a blast furnace is often expressed in terms of 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 1.11.

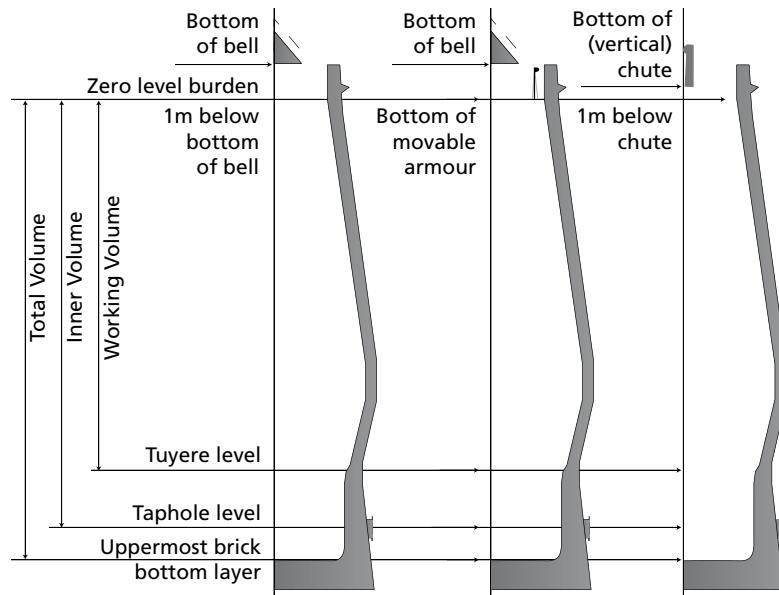


Figure 1.11 Definitions of working volume and inner volume

Presently, very big furnaces reach production levels of 12,000 t/d or more. E.g. the Oita blast furnace No. 2 (operated by Nippon Steel & Sumitomo Metal Corporation) has a hearth diameter of 15.6 meter and a production capacity of 13,500 t/d. In Europe, the ThyssenKrupp Steel Schwelgern No. 2 furnace has a hearth diameter of 14.9 m and a daily production of 12,000 t/d. In Russia, NLMK Blast Furnace No. 7 (13.1 meter hearth diameter) reached an average production of 12,500 tHM/d. In 2013, POSCO commissioned their Gwangyang No. 1 blast furnace with an inner volume of 5800 m³ and a rated capacity of 5.65 million ton per year (around 16,000 tHM/d). It is the largest furnace at the time.

1.4.4 Blast furnace profile

There is no scientific, theoretically justified practice of calculating the blast furnace profile. The dimensions are estimated based on empirical formulas and past experience, which takes different operating conditions, like the type of burden materials, into account.

The design of a blast furnace starts with the production target for the furnace. Taking into account the quality of the burden, a productivity target as tHM (ton Hot Metal) per m³ working volume per 24 hours can be derived from benchmarking with comparable furnaces. Then the working volume can be derived. Hot metal production, together with the expected slag volume, can be

used to calculate the production of the total liquid volume, which is required to define the hearth diameter and tuyere-to-taphole distance.

As soon as these primary dimensions are known, the next step is to derive the total furnace dimensions, such as the height from tuyere to stockline and to bottom, the number of tuyeres, the bosh dimensions, stack dimensions, throat dimensions and the number of tapholes (Figure 1.12). Of special importance are the slopes or angles of the bosh and shaft, which vary from furnace to furnace in the range of 81–86 degrees and 76–82 degrees respectively.

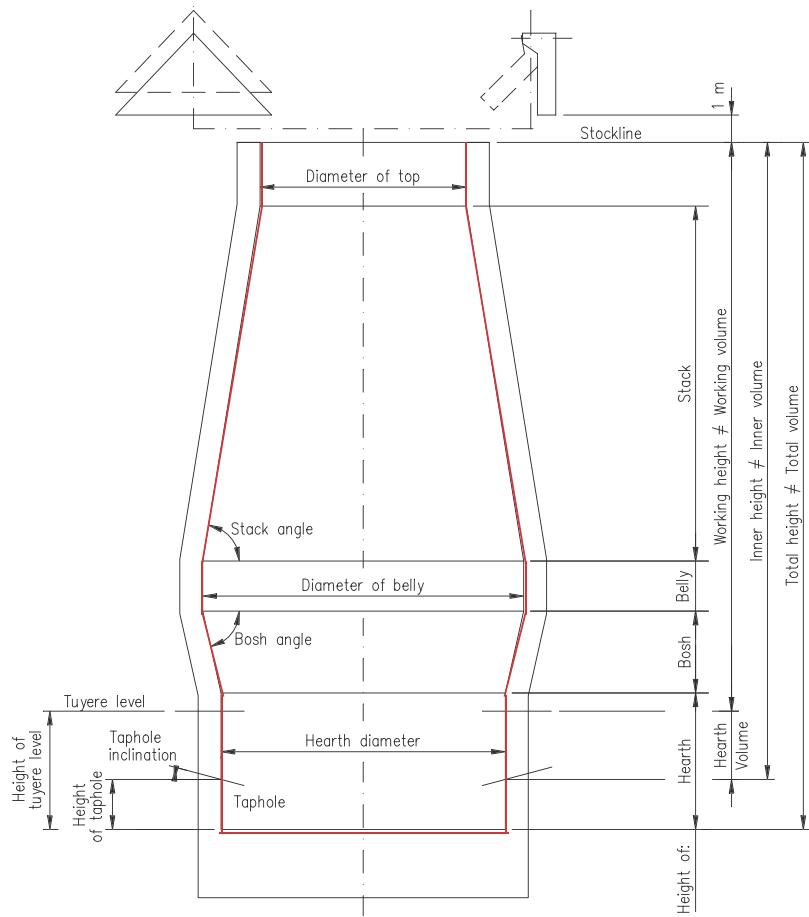


Figure 1.12 Major dimensions of a blast furnace

1.4.5 Cooling systems

In order to maintain the integrity of the steel shell of the furnace and because of the high temperatures, the inside of the shell is protected by a water-cooled refractory lining. An extensive discussion of cooling systems and specifications is beyond the scope of this book.

There are two main shell cooling systems in use: stave cooling and horizontal cooling plates, as shown in Figure 1.13 for the bosh of a furnace. Cooling plates are water-cooled, typically produced from copper castings, with a relatively small distance between plate locations and an insert depth of 40–50 cm. When leaking, they can be replaced from outside the steel shell during a blast furnace stop within a few hours.

Staves are vertical cooling elements, which can be made from cast iron, steel or copper, covered with refractories on the hot face. The insert depth is smaller, which leads to an increase in working volume of a blast furnace when cooling plates are replaced by staves.

The advantage of staves is that when the burden descends, the material can follow the wall rather easily. This is especially an advantage in the stack. The advantage of cooling plates is that the cooling plates can work as an anchor for freezing a protective skull, which is difficult with staves. So especially in the lower part of the furnace cooling plates are more forgiving than staves. Therefore, hybrid systems are presently being developed and installed.

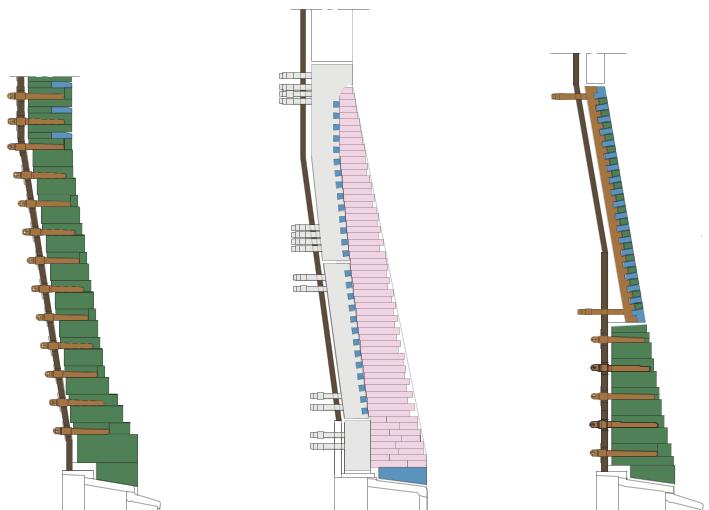


Figure 1.13 Cooling systems: plate coolers, staves and hybrid

1.5 Book overview

Blast furnace ironmaking can be discussed from three 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 “operator’s view”, with the aim to understand what is going on inside the blast furnace. To this end, the principles of the process are discussed (Chapter II), followed by the demands on burden quality (Chapter III) and coke and auxiliary reductants (Chapters IV and V). Simplified calculations of burden and top gas are made (Chapter VI). The control of the process is discussed in Chapter VII: burden descent and gas flow control. The issues pertaining to understanding the blast furnace productivity and efficiency are presented in Chapter VIII. Subsequently, hot metal and slag quality (Chapter IX), casthouse operation (Chapter X) and special operational conditions like stops and starts, high moisture input or high amounts of fines charged into the furnace (Chapter XI) are discussed.

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II *The Blast Furnace: Contents and Gas Flow*

2.1 **The generation of gas and gas flow through the burden**

The blast furnace process starts when pre-heated air, or “hot blast” is blown into the blast furnace via the tuyeres at a temperature of up to 1250 °C. The hot blast gasifies the reductant that is in front of the tuyere, which is either coke or another fuel that has been injected into the furnace through the tuyeres. This gasification generates a very hot flame and is visible through the peepsites as the “raceway”. At the same time, the oxygen in the blast is transformed into gaseous carbon monoxide (CO), and hydrogen (H₂) is generated. These gases can reduce the oxygen content of the burden, hence the name “reductant”. The hot flame generates the heat required for melting the iron ore (Figure 2.1a). The raceway is physically present in operation.

In Figure 2.1b, the remains of the raceway are shown, after a blast furnace has been taken out of operation.

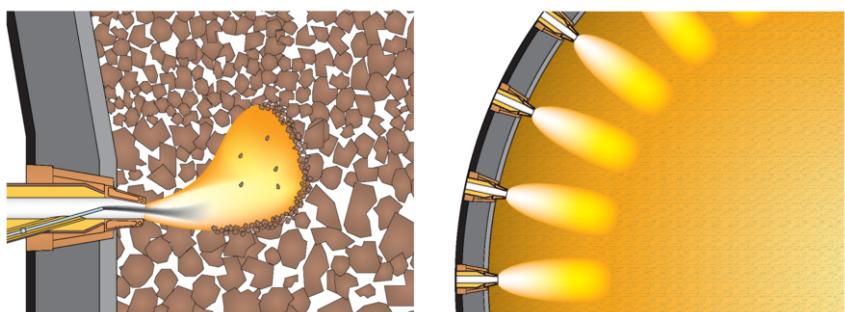


Figure 2.1a The raceway, horizontal and vertical sections



Figure 2.1b The raceway after a blowdown and nitrogen quench.

The blast furnace is a counter-current reactor (Figure 2.2). 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 (21 to 25 m from tuyeres to burden surface), the gas temperature drops from a flame temperature of 1900–2300 °C to a top gas temperature of 100–150 °C.

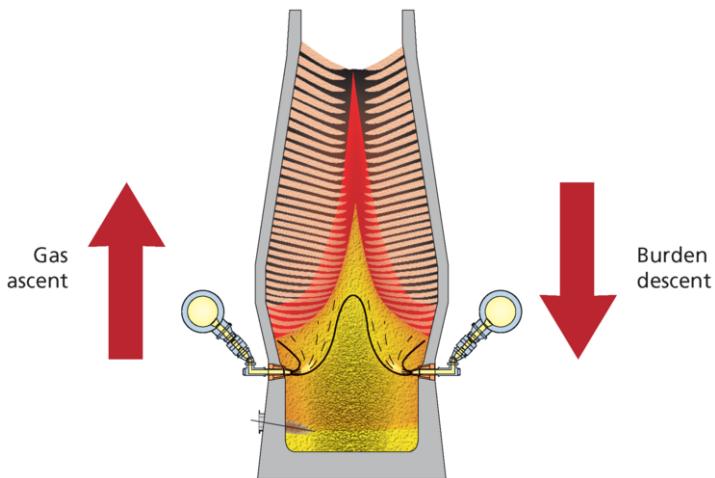


Figure 2.2 The blast furnace as a counter-current reactor

The process starts with the hot blast through the tuyeres, which reacts with the coke and coal or natural gas injection in the raceway (Figure 2.1). Gasification of the coke creates hot gas, which is able to melt the ore burden. Consumption

of coke and melting of the ore burden creates voidage inside the furnace, which is filled with descending burden and coke. Moreover, gasification of coke and coal generates carbon monoxide (CO) gas. For every molecule of oxygen, two molecules of carbon monoxide are formed: every cubic meter (m³ STP) of oxygen will generate 2 m³ STP of CO. So if the blast has 74 % of nitrogen and 26 % of oxygen, the bosh gas will consist of 59 % (i.e. 74/(74+2x26)) Nitrogen and 41 % CO gas. For the time being, we neglect the effect of hydrogen (see section 5.1) In addition, a huge amount of heat is generated in the raceway from the combustion of coke and injectants (coal, oil, natural gas). The heat leads to a high flame temperature, which generally is in the range of 1900–2300 °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 higher up in the furnace. The gas flowing through a blast furnace filled with coke is presented in Figure 2.3. To the experienced blast furnace operator, the furnace filled with only coke may seem like 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.

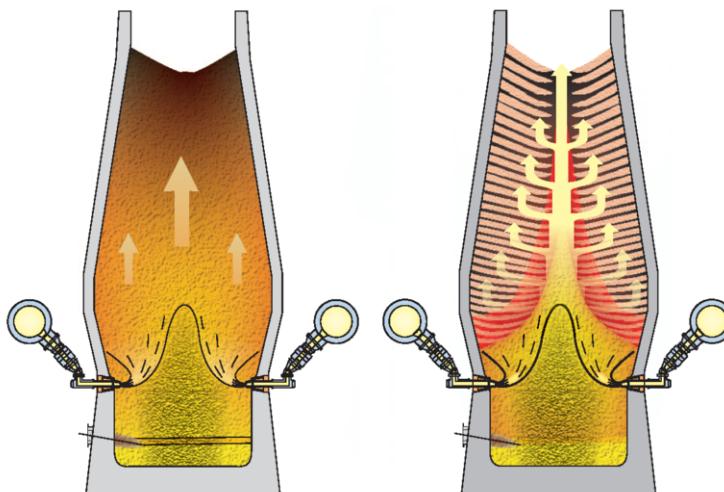


Figure 2.3 Gas flow in a furnace filled with coke only (left) and in a furnace filled with alternating layers of coke and ore (right).

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 into layers. It is important to note that the permeability of coke is much higher than that of the ore (see also Figure 7.7). 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 blast furnace is typically 45 to 55 mm, while the average size of sinter is 10 to 20 mm and that of pellets is 6 to 15 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 1500 °C, the iron temperature during casting, is transferred to the layered burden and coke, causing the metallic portion to melt and be heated. In the temperature range from 1000 to 1350 °C the burden will soften and stick together rather than melt. Below 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 part of the furnace.

The gas has now cooled to about 1000 °C and additional gas has been generated. Since the direct reduction reaction requires a lot of energy, the efficiency of the furnace is largely dependent on the amount of oxygen removed from the burden materials before reaching this 1000 °C temperature.

In summary:

- Heat is transferred from the gas to the ore burden, which melts and softens (over 1100 °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 1000 °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 is vaporized in the “drying zone” and 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 diagram of the removal of oxygen from the ore burden is shown in Figure 2.4.

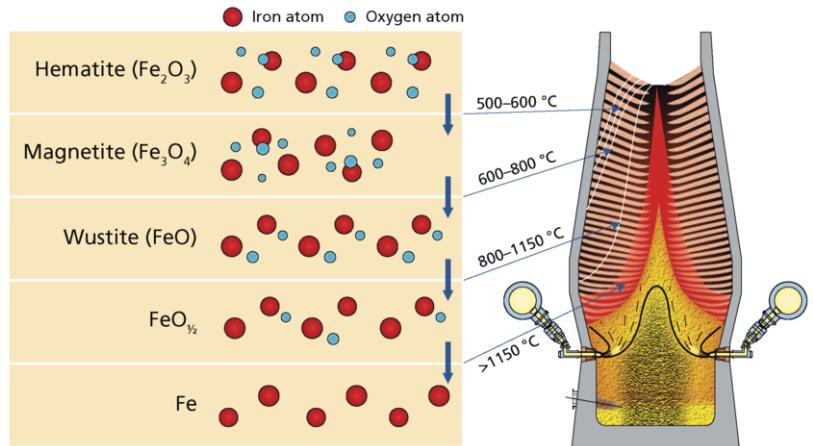


Figure 2.4 Schematic presentation of reduction of iron oxides and temperature

The first step is the reduction of hematite (Fe_2O_3) to magnetite (Fe_3O_4). The reduction reaction generates energy, so it helps 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 (LTD). Several tests are available to quantify the effects (see Chapter III). The second step in reduction is magnetite (Fe_3O_4) to wustite (FeO). This reduction reaction consumes energy.

Further down in the furnace, the temperature of the burden increases gradually until the burden starts to soften and melt in the cohesive zone. The molten iron and slag drip through the coke layer below the cohesive zone and 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. 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. 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 (coke plus injectants containing carbon) rate per tonne hot metal, is continuously monitored through measurement of the chemical composition of the gas exiting the furnace top known as “top gas”. The efficiency is expressed as the gas utilization formula, which indicates the percentage of CO gas utilized for reduction of the ferrous burden.

$$\eta_{CO} = \frac{CO_2}{(CO + CO_2)}$$

In addition, at modern furnaces, the gas composition over the radius is measured frequently. 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 across the radius. The wall zone is especially important 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 heat losses via the cooling system.

The top gas analysis gives a reasonably accurate indication of the efficiency of the furnace. When comparing different furnaces, one should realize that the hydrogen also takes part in the reduction process (section 8.2.4).

The gas utilization also depends on the amount of oxygen that must be removed. Since pellets have about 1.5 atoms of oxygen per atom of Fe (Fe_2O_3) and sinter has about 1.45 (mix of Fe_2O_3 and Fe_3O_4), the top gas utilization will be lower when using sinter. It can be calculated as about 2.5 % difference in top gas utilization, when comparing an all pellet burden with an all sinter burden.

2.3 Removal of liquids

The liquid hot metal and slag have to be removed from the furnace. This is done by making a hole (taphole) of 40–60 mm diameter and a length of 2.5–3.5 m in the furnace. Hot metal and slag separate in two liquid phases. The hot metal is more dense (around 7.2 t/m³) than slag (around 2.3 t/m³). As a consequence, the hot metal generally comes out of the taphole first and later hot metal and slag are drained together. Finally, when the slag level is below the taphole elevation, some of the gas generated in the raceway blows out of the taphole indicating the end of the cast. An operator should be aware, that the volumes of slag and iron are quite similar: the hot metal volume and slag volume produced are about equal at a slag volume of 320 kg/tHM. At higher slag rates, there is a higher

volume of slag to be cast than hot metal. Moreover, since slag is more viscous and its viscosity also more variable than that of hot metal, it is more difficult to remove slag from the furnace than hot metal.

2.4 An example of gas flow and contents of a blast furnace

The contents of a blast furnace and the gas flow can be derived from operational results. How long do the burden and gas reside within the furnace? In this book, we use a consistent set of calculation examples with the same starting points. Details of the starting points are given in Annex 3.

For the content of a blast furnace, a large, high productivity blast furnace with a 14 m hearth diameter is being used. It has a daily production of 10,000 tHM at a coke rate of 300 kg/tHM and a coal injection rate of 200 kg/tHM. Yield losses are neglected. Additional data are given in Table 2.1.

	Consumption	Specific weight	Carbon content
Ore burden	1600 kg/tHM	1800 kg/m ³	
Coke	300 kg/tHM	470 kg/m ³	87.0 %
Coal	200 kg/tHM		80.0 %
Blast Volume	6500 m ³ STP/min		
O ₂ in blast	26.0 %		
Working volume	3800 m ³		
Throat diameter	10 m		
Hearth diameter	14 m		
A charge contains	96.0 t ore burden		
A ton hot metal contains	945 kg Fe 45 kg carbon		
Voidage in shaft	30 %		

Table 2.1 Data for calculation of blast furnace contents
1 tonne hot metal contains 945 kg Fe = 945/55.6 = 17.0 kmol

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

Oxygen from the blast volume amounts to $0.26 \times 6500 \text{ m}^3 \text{ STP/min} = 1690 \text{ m}^3 \text{ STP oxygen/min}$. The production rate is $10,000/(24 \times 60) = 6.94 \text{ tHM/min}$. So the oxygen use is $1690/6.94 = 243 \text{ m}^3 \text{ STP blast oxygen/tHM}$.

2.4.2 How often is the ferrous burden replaced?

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

- 300 kg coke: $0.64 \text{ m}^3 (300/470)$ volume
- 1600 kg sinter/pellets: $0.89 \text{ m}^3 (1600/1800)$ volume
- Total per tonne of hot metal: 1.53 m^3 volume

Production is 10,000 tonne per day, which is $10,000 \times 1.53 \text{ m}^3 = 15,300 \text{ m}^3$ volume per day. This material can be contained in the working volume of the furnace minus the unused volume in the throat (1 m burden level) minus the volume used for the active coke zone. The working volume is 3800 m³, the unused volume at the top (1 m below 0-level of stockline) is about 79 m³ and we estimate the active coke zone as 3 meters above the tuyere level, corresponding with 461 m³. So the burden is processed in a volume of $3.800 - 79 - 462 = 3260 \text{ m}^3$. Not taking any compression into account, the contents of the furnace are refreshed 4.7 times per day ($15,300/3260$). This means the burden charged at the top reaches the tuyeres in 5.1 hours.

2.4.3 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. A layer contains 96 tonnes, corresponding with 60 tHM. In 5.1 hours, the furnace produces 2135 tonnes, which corresponds with 36 layers of ore ($2135/60$). In our example, taking a throat diameter of 10 m, the ore layer is 68 cm and the coke layer is an average of 49 cm at the throat.

2.4.4 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 80 %: 160 kg C
- Total carbon: 421 kg C

About 45 kg carbon dissolves in the hot metal. The balance leaves the furnace through the top gas, which is $421 - 45 = 376 \text{ kg}$. It leaves the furnace as CO and CO₂.

2.4.5 How is this top gas CO and CO₂ produced?

All oxygen input in the tuyeres gasifies carbon to CO. The amount of oxygen entering the furnace comes mainly from the wind (2.4.1: 243 m³/tHM) and some additional oxygen comes from moisture and coal, when calculated as m³ STP it is 17 m³, totalling to 260 m³ STP O₂. 1 molecule O₂ generates 2 molecules CO. So the carbon gasified in front of the tuyeres is $260 \times 2 \times 12 / 22.4 = 279 \text{ kg carbon per tonne HM}$. The top gas contains (see 2.4.4) 376 kg, the balance, 97 kg carbon, comes from direct reduction reactions.

2.4.6 Estimate how long the gas remains in the furnace

The blast volume is $6500 \text{ m}^3 \text{ STP/min}$ with 26% oxygen. Since for every m^3 of oxygen, two m^3 of CO is produced, the raceway gas amounts to $6500x(1+0.26)=8190 \text{ m}^3 \text{ STP/min}$. This gas has a higher temperature (decreasing from some 2200°C to 125°C top gas temperature), the furnace is operated at a higher pressure (assume 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 2.4.5) and finally, it contains hydrogen as well. 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 the working volume (3800 m^3) minus the part that is not used at the top, estimated 79 m^3 corresponding to $(3800-79)x0,3 = 1116 \text{ m}^3$, through which $8190 \text{ m}^3 \text{ STP}$ gas is blown per minute. So the residence time of the gas is $(1116/8190)x60 = 8 \text{ seconds}$.

It is possible to make the corrections mentioned above. Assume an average temperature of the gas of 900°C and an average pressure of 4 bar, 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 and hydrogen is $8190/10,387= 0.79$ times shorter. Extra gas from direct reduction is the gas coming from 97 kg carbon per ton is $97x6.94x22.4/12 (\text{t}/\text{min}) = 1.257 \text{ m}^3/\text{min}$, extra hydrogen is $940 \text{ m}^3/\text{min}$, in total $8190+1257+940=10,387 \text{ m}^3/\text{min}$
- In total, the residence time is shorter by a factor of 0.73 ($4x0.23x0.79$), so the corrected residence time is $8x0.73 = 6 \text{ seconds}$.

2.4.7 If there is so much top gas, is there a strong "wind" the furnace?

No, at the tuyeres there are high blast velocities (over 200 m/sec), but top gas volume is about $10,387 \text{ m}^3 \text{ STP/min}$. 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/s . This corresponds with Beaufort 1, which is a very light breeze. Through the voids, the velocity is about 3 m/s . Note, that in the center, the velocity can be higher, so that even fluidisation limits can be reached (section 7.4).

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III *The Ferrous Burden: Sinter, Pellets, Lump Ore*

3.1 Introduction

In the early days of commercial ironmaking, blast furnaces were often located close to iron ore mines. In those days, blast furnaces were using local ore and charcoal, later replaced by coke. In the most industrial areas of that time, the 19th century, many blast furnaces were operating in Germany, Great Britain and the United States. After the application of the steam engine for ships and transportation, the center of industrial activity moved from the ore bodies to the major rivers, such as the river Rhine, and later from the rivers to the coastal ports with deep sea harbours. This trend, supported by seaborne trade of higher quality ores may appear clear at present, but has only a recent history. In 1960, there were sixty operating blast furnaces in Belgium and Luxembourg. Since 2009, only two are operating, both have the favorable 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, frequently obtained after a physical beneficiation or enrichment process of the ore at the mine, consequently creating more fine material as opposed to a rich and good lump ore. These fine ores are too impermeable to gas flow to be charged directly to the blast furnace and hence sintering and pelletizing as an agglomeration process is favored. Sinter and pellets are used in combination with lump ore as the ore burden. Briquetting of in-plant revert materials is an option as recycling of some of these waste materials through a sinter plant becomes more restricted or a prerequisite in absence of a sinter plant. Boosting blast furnace productivity through charging of metallic iron (scrap, DRI or HBI) can finally make up the total ferrous burden, while reducing greenhouse gas emissions.

A good blast furnace burden consists, for the major part, of sinter and/or pellets and can be topped off with sized lump ore (Figure 3.1 on the next page). Sinter burdens are prevalent in Europe and Asia, while pellet burdens are used more commonly in North America and Scandinavia. Many companies use sinter as well as pellets, although the ratios vary widely.



Figure 3.1 *Burden materials*

Lump ores are becoming increasingly scarce and generally have poorer properties as a blast furnace burden. For this reason, it is used mainly as a lower cost replacement for pellets. For high productivity, low coke rate blast furnace operation, the maximum lump ore rate is around 10 to 15 %. The achievable rate depends on lump ore quality and the successful use of higher percentages is documented. The present chapter deals with the ferrous burden quality.

3.2 Iron ore

Iron is the fourth most abundant element in the earth crust, making up approximately 5 % of the total. However, mining of iron (as an oxide) is only economically viable where substantial concentration has occurred, and only then can it be referred to as iron ore. More than 3 billion years ago, through the generation of Banded Iron Formation, the first concentration occurred. The conventional concept is that several billion years ago the banded iron layers were formed at the bottom of ancient seas as the result of an increase in oxygen in the water to form insoluble iron oxides which precipitated out of the water, alternating with mud, which later formed cherts and silicate layers.



Figure 3.2 *Banded Iron Formation (National Museum of Mineralogy and Geology, Dresden, picture by André Karwath, file from the Wikimedia Commons)*

Subsequently, leaching out of the cherts and silicates resulted in a concentration of the iron oxide and through further geological processes such as (de)hydration, inversion leaching, deformation and sedimentation, a wide variety of iron ore deposits have been created all around the world. These total over 300 billion tonnes at an average Fe content of 47 %.

A minor fraction of these deposits are currently commercially mined as iron ore with Fe contents ranging from below 30 % to up to 64 % (pure iron oxide as hematite (Fe_2O_3) contains 70 % Fe). As mentioned before, an efficient blast furnace process requires a rich Fe burden, preferably in excess of 58 % Fe. Within the conventional mining process of drilling, blasting and crushing, a wide size range of particles is obtained. Through screening, lump ore and sinter fines are separated. However, an increasing part of the iron ore needs further beneficiation and processing prior to becoming a usable material for the blast furnace. A vast amount of equipment has been developed to economically upgrade the iron ore to a suitable product. These processes will not be described here, but most of them are based on liberating the iron oxide from the gangue minerals and then making use of the differences in density, magnetic properties or surface properties between these minerals to separate them physically. Sometimes, large amounts of quartz (SiO_2) need to be removed, or minor amounts of impurities (such as phosphorus in the mineral apatite). Depending on the specific requirements, these processes can be easily achieved, or they can be difficult or even impossible.

These processes result in a wide variety of beneficiated iron ores with varying grades, impurities and sizes available on the market. Silica content can vary between 0.6% to above 10% and phosphorus from below 0.005% to above 1%. Similar variations apply for other components such as the oxides of aluminium, calcium, magnesium, manganese, titanium and alkalis. With tighter environmental control over the whole process chain, tramp elements at minute levels are starting to play a more dominant role. From sulphur, zinc and copper to mercury, arsenic and vanadium. The importance of these elements greatly depends on the applied process and process conditions, environmental measures and local legislation of where the ores are to be used.

Due to the physical beneficiation, more and finer iron ores have been generated and though the designation is not consistent, the ore in the size range for sintering (indicative between 6 mm and 150 μm) is called sinter feed, while smaller and more narrowly sized ore is denoted concentrate and finally pellet feed which is, as an indication, below 150 μm and suitable mainly for pelletizing. The original lump ore is, as an indication, between 8 and 40 mm. Most common ferrous ores are hematite (Fe_2O_3), magnetite (Fe_3O_4) and goethite ($\text{Fe}_2\text{O}_3 \cdot n\text{H}_2\text{O}$)

3.3 Quality demands for the blast furnace burden

The demands for the blast furnace burden extend to the chemical composition and the physical durability of the burden materials. The chemical composition must be such, that after the reduction and melting processes, the desired iron and slag compositions are produced, and this will be determined by the chemical composition of all the materials charged in the furnace. The physical and metallurgical aspects of the quality demands are related to the properties in both the cold and the hot state and to ensure that the gas can flow through the burden and react with the ferrous components. Both aspects are discussed in depth in this chapter.

3.3.1 Generation of fines, reducibility, softening and melting

In the shaft zone of the blast furnace, the permeability of the burden is determined by the sizing of the burden components and specifically the amount of fines (see Figure 3.3). Fines may be defined as the fraction of the material smaller than 5 mm, since the burden components have a general range of 5 to 25 mm. If there are too many fines and especially fines below 5 mm, the void fraction used for the transport of the reduction gas will be reduced and will affect the bulk gas flow through the burden (Hartig et al, 2000). The furnace contains fines from two sources: those that are charged into the furnace and those that are generated by the process.

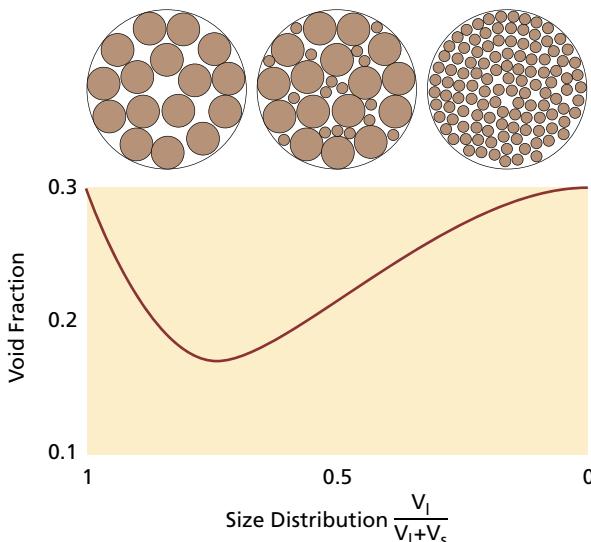


Figure 3.3 Permeability for gas flow depends on void fraction, which depends on the ratio of smaller and larger particles. Example of two types of spherical particles, large (V_l) and small (V_s). The x-axis gives the fraction of the large particles: $V_l/(V_l+V_s)$. The ratio $V_l/(V_l+V_s)$ itself highly influences the absolute void fraction. This example assumes a ratio between two sizes of particles; in reality always a size range will exist.

Fines in the burden and coke can be controlled by proper screening in the stockhouse, just before charging to the blast furnace. The aim should be less than 3% below 5 mm after screening in the stockhouse. Measurement of the percentage of fines after screening in the stockhouse can give an indication of whether or not excessive fines are charged into the furnace. Material from the stockyard will have varying levels of fines and moisture and thus screening efficiency will be affected accordingly. In some cases, iron ore is pre-screened in the stockyard to remove fines that cannot be removed in the stockhouse.

Once in the furnace, 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 or reduction-degradation. The reduction-disintegration depends on the strength of the bonds between the particles of ore fines in sinter and lump ore. Generally speaking, the reduction disintegration is dependent on:

- The FeO percentage, especially in the sinter. The more magnetite (Fe_3O_4 , which corresponds with $\text{FeO} \cdot \text{Fe}_2\text{O}_3$) is present, the stronger the sinter and the less reduction disintegration can take place at low temperature caused by the change in crystal structure from hematite to magnetite. In the sinter process, the FeO percentage in the sinter can be increased by cooling sinter with air that is low in oxygen. In an operating plant, the FeO in the sinter can be increased by adding more fuel (coke breeze) to the sinter blend.
- The chemical and mineralogical composition of the lump ore and sinter. For sinter, basicity, Al_2O_3 and MgO content play an important role. Higher basicity improves sinter strength while higher alumina and magnesia can have a negative impact on sinter strength.
- The heating and reduction rate in the furnace. The slower the progress of heating and reduction, the higher the reduction-disintegration of sinter and lump ore.
- The amount of hydrogen in the reducing gas. More hydrogen in the reducing gas leads to lower reduction-disintegration.

A major requirement for the blast furnace ore burden is to limit the quantity 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 mm aperture 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 slower warm-up and hence result in a higher level of reduction-disintegration, making this negative process self-sustaining.

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 and the permeability of the layers, which determines the blast furnace internal gas flow. This is discussed in depth in the later blast furnace chapters. The intrinsic reducibility of the burden components will be of less importance as long as the gas flow within the furnace does not allow sufficient gas for the reactions to take place.

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 and the difference between softening and melting temperatures should be as narrow as possible, so that they do not impede gas flow while they are still high up in the stack. Melting properties of burden materials are determined by the slag composition. Melting of acid pellets and lump ore starts at temperatures of 1000 to 1100 °C, while fluxed pellets and basic sinter generally starts melting at higher temperatures. See also section 9.1 on how iron ore melts.

3.3.2 Ore burden quality tests

Ore burden material is characterized 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 characterize the degradation of ore burden materials during transport and handling.
 - Reduction-disintegration, which characterizes the effect of the first reduction step and is relevant in the stack zone of the furnace.
 - Reducibility, which characterizes the capability of the material to improve the extent of reduction reactions
 - Softening and melting properties, which are important for the formation of the cohesive and melting zone in the furnace.

Most of the tests are ISO standardized; some of the tests are generic for sinter, pellets and lump but some are specific. For pellets for example, a cold compression strength and a swelling test are applicable. The latter is to ensure that the volume increase during reduction does not exceed a set maximum. Specifically lump ores are tested for decrepitation: fracturing of the lump due to thermal decomposition of crystalline water during the initial stage of heating near the top of the blast furnace.

Table 3.1 summarizes these characteristics with an indicative optimum range. A short description of generic tests used for characterization of materials is given below, with the objective to help understand the terminology.

	What is measured?	Results	Optimum Range		Reference
			Sinter	Pellets	
Mean Size	Size distribution	Average size, mm % 6.3–16 mm % < 0.5 mm	< 2 %	> 95 % < 2 %	ISO 4701
Cold Strength	Size distribution after tumbling Compression	% > 6.3 mm % < 0.5 mm daN/p	> 70 %	> 90 % < 5 % > 150	ISO 3271 ISO 4700
Strength after reduction <i>LTD (Low Temp. Disintegration)</i>	Size distribution after reduction and tumbling	% > 6.3 mm % < 3.15 mm % < 0.5 mm	< 20 %	> 80 % < 10 %	ISO 4696
Reducibility	Weight decrease during reduction	%/min	> 0.7 %	> 0.5 %	ISO 4695

Table 3.1 Characterization of ore burden

3.3.2.1 Tests for cold strength

Cold strength is mostly characterized 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 is measured. The size distribution after tumbling is determined and used as a quality indicator (Figure 3.4). This process simulates the physical transport and handling of all ferrous burden components. Pellets are also tested for Cold Compression Strength to guarantee intrinsic strength during handling and processing inside the blast furnace.

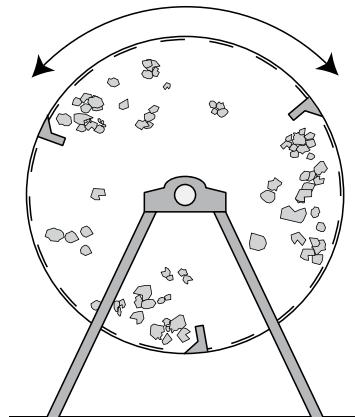


Figure 3.4 Principle of tumbler test

3.3.2.2 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 and tumbled and the amount of fines is measured. The quoted result is the percentage of particles smaller than 3.15 mm.

Reducibility is measured with two different ISO tests at a temperature of 900 or 950 °C in a gas containing CO. During the test period, the weight of the sample is measured continuously. The weight loss is contributed to oxygen loss and either a figure for reducibility (dO/dt) or final reduction degree after 180 minutes is obtained.

Sometimes, non-ISO standardized tests are applied with the aim to simulate the actual blast furnace conditions. HOSIM is such an example of a blast furnace simulation test where the sample is reduced to the endpoint of gas-reduction in a furnace. After the test, the sample is tumbled. The results are the reducibility defined by the time required to reduce the sample to the endpoint of gas reduction, and the reduction-disintegration is represented by the percentage of fines (under 3.15 mm) after tumbling.

Although all these tests are relevant for the upper part of the blast furnace process, the ISO tests are excellent to give an idea of burden quality, but the more advanced simulation tests gives a more realistic description of the effects of that burden in the blast furnace. If test conditions are continued to higher temperatures and the direct reduction is included (i.e. simulation of the lower part of the furnace), tests are referred to as softening and melting tests. Pressure drop over the ferrous burden sample, softening and melting trajectories and the amount of material dripped out of the sample characterize the burden.

3.4 Sinter

3.4.1 Sinter quality

Sinter is made in three different types: acid (CaO/SiO_2 ratio below 1.0), fluxed (CaO/SiO_2 ratio between 1.0 and 2.5) and super-fluxed (CaO/SiO_2 ratio over 2.5) sinter. Fluxed sinter is the most common type. Since sinter properties vary considerably with the blend type and chemical composition, only some qualitative remarks can be made.

The sinter quality is defined by:

- Size distribution: sinter size ranges from 5 to 40 mm with a 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 the generated fines. Sinter from the stock yard may have 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, for example by using a dedicated bin in the stockhouse to stock sinter or pre-screened in the stockyard before going to the stockhouse.
- Cold strength: normally measured with the tumble test. The more energy is used in the sinter process, and the more gangue components such as silica, the stronger the sinter. The cold strength influences the sinter plant productivity because a low cold strength results in a high fines recycle rate.

- Reduction-disintegration (reduction-degradation index or RDI used in Japan) 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. For an operator, this is the fastest way to improve sinter quality, at cost. However, reducibility properties are adversely affected.

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 3.5, there is primary and secondary hematite in the sinter. Particularly the latter causes reduction-disintegration, since it is more easily reduced in the upper part of the furnace than primary hematite (see Figure 3.5).

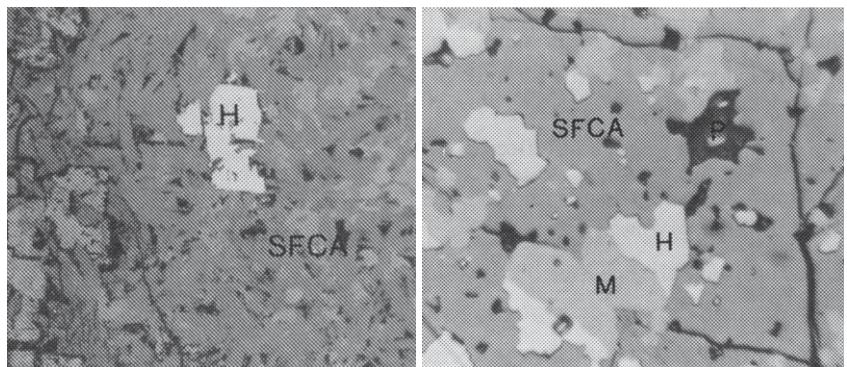


Figure 3.5 Cracking of calcium ferrites (SFCA) due to reduction of hematite (left) into magnetite (M). Pores appear black. (Chaigneau, 1994)

The higher the secondary hematite percentage in the sinter, the more the sinter is prone to reduction-disintegration effects. This can also be said in reverse, that is, there is a 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 to the sinter blend, which is normally done in the form of coke breeze. 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. To illustrate this by example: a high fraction of magnetite in the sinter blend will give sinter with a high (primary) magnetite fraction. Moreover, in the presence of sufficient SiO_2 fayalite structures ($2\text{FeO} \cdot \text{SiO}_2$) can

be formed. These structures are chemically very stable and can only be reduced at high temperatures by direct reduction reactions. Alternatively, in the presence of MgO, spinel structures containing large amounts of FeO can be formed. These spinel structures are relatively easy to reduce. Finally, sinter that has been formed at high temperatures (acid sinter), will contain glass-like structures where the FeO is relatively difficult to reduce.

It is possible to suppress the formation of secondary hematite by cooling the sinter with air-gas mix with a reduced oxygen percentage (12 to 14%). This results in a relatively high FeO content of the sinter, 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, as less oxygen has been removed from the ore burden, giving an economic advantage.

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, using limestone as CaCO_3 or dolomite as $\text{Ca.Mg}(\text{CO}_3)_2$. The carbonates are decomposed on the sinter strand, requiring a large energy input. However, the melts containing substantial amounts of CaO have low liquidus temperatures, such as 1100 °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 using CaO than with MgO. And generally, making sinter with CaO can be done at a lower temperature. But sinter with high MgO is more resistant against reduction-disintegration. MgO content can be increased by adding olivine or serpentine to the sinter blend. However, there is an upper limit for MgO content where sinter strength is negatively impacted. Again all these relationships and their limits are plant-specific, based on raw materials and sinter equipment.

For the final result of the produced sinter, it is important to note that the sinter blend prior to sintering is far from homogeneous. It contains various types of material and locally there are widely varying compositions and sizes 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 microscale, where the sintering takes place. Types 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 specific disadvantages for good sinter quality. This makes optimization of sinter quality a plant-specific technological challenge.

In the above sections, the importance of reduction-disintegration of sinter is stressed. The lower the reduction-disintegration, the poorer the reducibility of the sinter. Needle-like structures of calcium 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 layers in the furnace and impedes proper further reduction of the iron oxides in the blast furnace.

3.5 Pellets

3.5.1 Pellet quality

With the correct chemical composition and induration conditions, pellets can be easily transported from mine to blast furnace. Pellets can be stocked and generally remain intact in the blast furnace. Therefore, when judging pellets, the main issues are:

- Cold strength, measured as compression strength and the fines generated through tumbling. Low figures for compression and tumble index indicate bad or lean firing, which results in a pellet with a hard outer shell but an unfired core.
- The reduction–disintegration properties. These properties are less of a concern with pellets than with sinter and lump ore.
- The 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 softening and melting. Acid pellets tend to melt at lower temperatures than fluxed sinter.

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

- Acid pellets (CaO/SiO_2 ratio below 0.5).
- Basic or Fluxed pellets using limestone or dolomite as additives (CaO/SiO_2 ratio 0.9–1.3).
- Olivine pellets using olivine as the fluxing additive.

Typical properties of the three types of pellets are shown in Table 3.2.

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

Pellet Type	Fe %	SiO_2 %	CaO %	MgO %	Compression (daN/pellet)
Acid	66–67	2–5	< 0.5	0.1–0.6	>270
Basic	63–66	1.5–4	0.8–1.1	0.1–1.5	>240
Olivine	64–67	2–4	< 0.5	1.3–1.8	>200

Table 3.2 Overview pellet properties (daN = Decanewton, around 1.02 kgf)

Acid pellets are strong, but have moderate metallurgical properties. They have good compression strength (over 250 daN/pellet), but relatively poor reducibility. The softening and melting temperatures are low compared to that of fluxed pellets or sinter. In addition, acid pellets are very sensitive to the CaO content with respect to swelling. When the CaO/SiO₂ ratio is over 0.25, some pellets have a strong tendency to swell, which might jeopardize proper blast furnace operation.

Basic and fluxed pellets have good metallurgical properties for blast furnace operation. By adding limestone or dolomite 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 % lower when producing basic pellets compared with production of acid pellets. However, the fluxed pellet reducibility, softening temperature and melting temperature are higher than that of acid pellets.

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

3.5.1.1 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 Fe₃O₄, the cold compression strength drops to 45–50 daN for acid pellets and to 35–45 daN for olivine pellets. Therefore, a little lower average compression strength has no drawback for the blast furnace process.

Though it is sometimes claimed that pellets need to be strong to support the burden on top, actual calculations show that the theoretical strength of a pellet needs to be minimum: with a ferrous burden density of 1800 kg/m³, the weight per cm² (which is roughly the cross sectional surface area of a single pellet of 11 mm) is 0.18 kg. A column of 20 m ferrous burden imposes a static ‘pressure’ of 3.6 kg on this single pellet only. Factors like burden movement, blast pressure counteracting the downward pressure, bridging and coke will offset this calculated pressure and make it dynamic. However, the average low compression strength can occasionally be the consequence of an increased percentage of very weak pellets (below 60 kg/pellet). That fraction is a good indicator for the pelletizing/induration process: the more pellets that collapse at low compression, 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 become stronger.

3.5.1.2 Swelling

As mentioned above, pellets, in contrast to sinter and lump ore, can have the tendency to swell during reduction. Generally, a volume increase of 20 % or over, measured according to ISO 4698, is seen as critical. The effect, however, depends on the percentage of pellets used in the burden. Swelling occurs during the transformation of wustite into iron, but like any transformation, this is a balance between iron nucleation and growth of these nuclei. During swelling, limited nucleation occurs and these nuclei grow like needles, causing a volume increase which is seen as swelling, see Figure 3.6. These needles, called whiskers, are difficult to observe; a microscopic image of the phenomenon is shown in Figure 3.7. Under certain conditions, for example in the presence of alkalis in the blast furnace, the swelling can become excessive and a cauliflower-like structure develops. This coincides with a low compression strength of this structure, with the opportunity to generate fines.



Figure 3.6 Balance between iron nucleation and nuclei growth. Limited swelling accompanied by the formation of an iron shell (left). Limited iron nucleation followed by strong needle growth of the nuclei with as a result excessive swelling of the pellet (right).

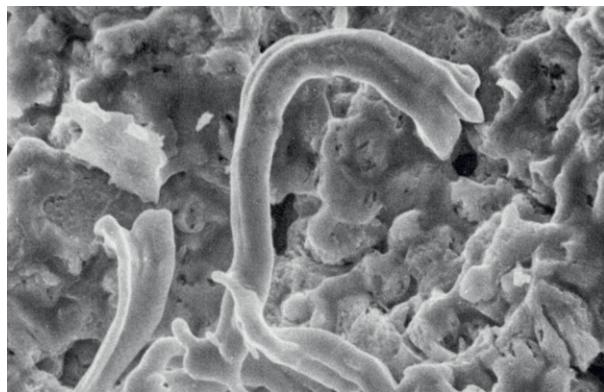


Figure 3.7 Whisker formation

Main factors influencing pellet swelling are basicity and gangue (non–iron compounds) content. Figure 3.8 on the next page shows how swelling depends on pellet basicity. Pellets with a basicity between 0.2 and 0.7 are more prone to swelling.

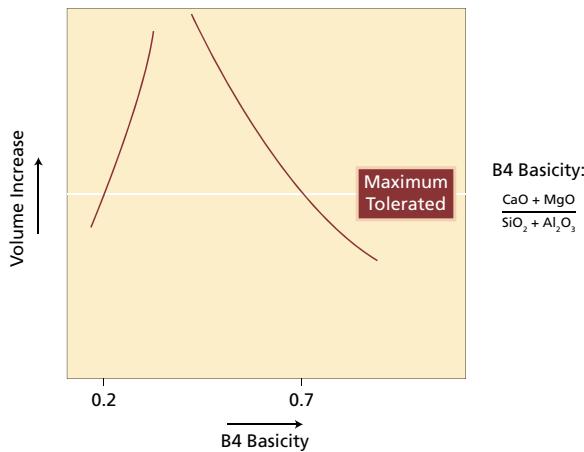


Figure 3.8 Graph showing volume increase effect of pellet swelling with increasing basicity of the pellet.

Swelling is mitigated by proper induration. In the blast furnace, local process conditions like temperature and gas composition greatly influence the swelling behavior. At higher reduction degrees, also swollen pellets will shrink. The Japanese measure this shrinkage in a Contraction Test and have empirical data showing that pellet contraction of over 10 % in a large blast furnace has a negative impact on productivity and reductant rate. As the phenomenon of swelling is well known, it is normally under control with commercially available pellets, but always requires a check because it could have a severe impact on the regularity of the blast furnace process.

An overview of pellet properties is shown in Table 3.3

	What is measured?	Results	Acceptable Range	Reference
Mean Size	Size distribution	% 6.3–16 mm % < 6.3 mm	> 95 % < 2 %	ISO 4701
Cold Strength	Compression Strength Tumbling Strength and Abrasion	Average kg/p % < 60 kg/p % > 6.3 mm % < 0.5 mm	> 150 daN/p < 5 % > 90 % < 5 %	ISO 4700 ISO 3271
LTB (Low Temp. Breakdown)	Size distribution after static reduction and tumbling	% > 6.3 mm	> 80 %	ISO 4696
Reducibility	Weight decrease during reduction	%/min $(dR/dt)_{40}$	> 0.5 %/min	ISO 4695

Table 3.3 Characterization of pellets

3.6 Lump ore

Lump ores are natural iron-rich materials, which are used directly from the mines after the crushing and screening operations. Because the lump ores are screened out at the mines, the mines generally produce lump ore as well as (sinter) fines. Major lump ore deposits are present in Australia (Pilbara region), South America (Carajas and Iron Ore Quadrangle), and South Africa (Sishen). In many other places, limited amounts of lump ores are produced. Lump ores are becoming more and more scarce.

The lump ores are priced lower than pellets. For this reason, 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 physical and metallurgical properties. Generally speaking, in comparison with pellets, lump ores:

- Show some decrepitation due to evaporating moisture in the upper stack of the furnace. This can be tested with the decrepitation test.
- Generate more fines during transport and handling.
- Have poorer reduction-degradation properties and may have poorer reducibility properties.
- Have a lower melting temperature.
- Have greater diversity in physical properties due to being naturally occurring.

Lump ore is used in an appropriate size fraction, such as 8–40 mm. For blast furnace operation at high productivity and high coal injection levels, lump ore is not the preferred burden material. As lump ore is a natural material, properties can differ from type to type. Certain types of lump ores can compete favourably with sinter, and for example Ternium Siderar blast furnaces in Argentina have operated successfully with up to 40 % of a Brazilian lump ore in the burden at high furnace productivity.

3.7 Metallics charge and briquettes

Two other components with ferrous content can be charged to the blast furnace:

- Metallics like Hot Briquetted Iron (HBI, Figure 3.9 on the next page), Direct Reduced Iron (DRI) and steel scrap to increase the productivity and efficiency of the blast furnace whilst limiting the greenhouse gas exposure.
- In-plant revert as briquettes. Especially if an on-site sinter plant is not available, valuable revert materials containing iron, manganese, carbon and fluxes can be charged with the burden. The phosphorous level is the limiting factor in many cases.



Figure 3.9 Hot Briquetted Iron (HBI)

Direct reduced iron (DRI) is seen more and more as a feedstock for the blast furnace. After hot briquetting in the form of HBI, it can be charged with the other burden components and with this HBI addition, the hot metal production is increased for situations where a mill is short on hot metal. In this way, advantage can be taken over charging these metallics directly into the converter. Typically, HBI contains over 90 % Fe and has a metallization degree higher than 90 %. Since the metallics only need to be melted and hardly reduced, a coke rate decrease per ton of hot metal is achieved and this helps lower the carbon dioxide generation within an integrated steel plant

Table 3.4 gives typical compositions of HBI in comparison with typical sinter and pellets.

	Fe	FeO	Met.Fe	SiO ₂	CaO	Al ₂ O ₃	MgO	C
HBI	92	15	80	2.0	0.9	0.8	0.3	1
Sinter	58	7	0	4.5	9.2	1.7	1.1	0
Pellet	65	0	0	4.5	1.0	0.4	0.4	0

Table 3.4 Typical chemical composition of HBI versus sinter and pellets

In our experience, if 100 kg HBI/DRI/Steel Scrap is used per tHM, the productivity increases with 4–6 % and the coke rate can be decreased by around 30 kg/tHM.

Cold briquetting of in-plant reverts such as blast furnace dust, BOF sludge and mill scale is a way to recycle these materials. The preferred way is to recycle them through the sinter process. In North America, some sinter plants only operate on these waste materials, whilst other locations lack such sinter capacity and need to landfill them, sell them or recycle them via the briquetting route. These cold-bonded briquettes often have cement as a bonding agent resulting in high strength cold properties but poorer reduction disintegration characteristics. However, a small furnace in India operates on 100 % briquettes, manufactured from sludges, top dust and iron ore fines (20 %).

3.8 Interaction of burden components

The results of burden tests on the total burden can differ greatly from results on sinter, pellets and lump ore alone. An example is given in Figure 3.10. A relatively poor quality of lump ore is blended with good sinter. It is shown that the behaviour of the blend is better than expected from the arithmetic mean of the data. Generally speaking, blending of materials dilutes the disadvantages of a certain material. Therefore, the blast furnace burden components have to be properly blended when charged into the furnace.

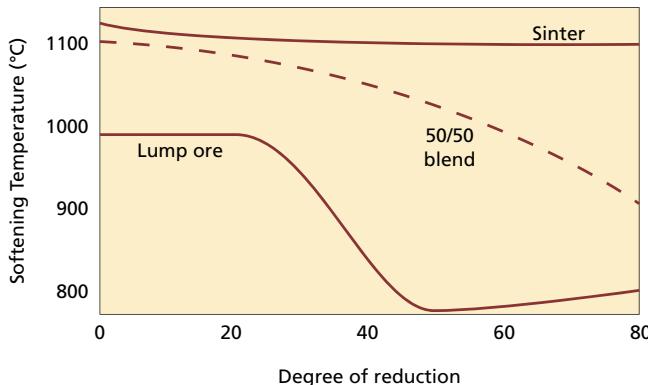


Figure 3.10 Softening temperature of a 50/50 blend of sinter and lump ore (example taken from Singh et al, 1984)

3.9 Chemical control of the burden

Though the aim of a blast furnace is to make iron, the importance of all other components coming with the burden as discussed in section 3.2 cannot be neglected and a certain burden composition is required to achieve a balance between these components. The vast majority will be tapped as slag during a hot metal tap, though some will partition between slag and hot metal, such as silica, manganese and sulphur and others will completely revert to the hot metal like phosphorus.

Silica, together with alumina, in quantity represents the bulk of the components originating from the ore gangue and coal and coke ash. They need to be fluxed with CaO and some MgO to achieve the desired slag chemistry. A low slag volume is desired, but depends on the quality of the raw materials available. Beneficiation of the raw materials to the desired low gangue and ash levels at the mine site is not always possible or only at increasing cost and yield losses. Furthermore, some gangue is required to attain a proper sinter quality. Operators will not go below a 3.5 % silica level in their sinter, with 4.5–5.5 % being more common. With pellets, a good quality can still be obtained with low gangue levels and consequently blast furnaces operating with 100 % pellets can achieve the lowest slag volumes per tonne hot metal.

As an example: what does it mean when the burden contains 1 % more silica (SiO_2)? Since we charge about 1600 kg per ton, 1 % additional silica means 16 kg of silica contribution which mainly ends up in the slag. In the simple case that we correct with lime only and have a CaO/SiO_2 ratio of 1, then we have to add also 16 kg of CaO , for which we need about double the amount in limestone, about 32 kg, which has to be processed in the sinter plant and consumes coke breeze (30 kg per tonne of limestone). So the slag volume increases with 32 kg/tHM, coke breeze consumption in the sinter process increases with 1 kg/tHM and coke consumption in the blast furnace increases with 1.6 kg/tHM while blast furnace productivity will decrease.

The amount of alumina in slag is generally limited to levels below 20 %, though some operators have learned to exceed these boundaries. If the burden composition does not allow for lower alumina levels, the only solution is to increase the total slag volume with a silica component and dilute the high level of alumina to acceptable levels. In this case, the slag volume is dictated by alumina rather than based on silica.

When a desirable burden composition has been established, fine-tuning can be achieved by charging small amounts of high siliceous ore, quartzite, or directly charge fluxes such as limestone or dolomite to the burden.

IV Coke

4.1 Introduction: function of coke in the blast furnace

Coke is basically a strong, non-melting material which forms lumps based on a structure of carbonaceous material glued together internally (Figure 4.1).



Figure 4.1 Coke

The average size of the coke particles is much larger than that of the ore burden materials and the coke will remain in a solid state throughout the blast furnace process, even at the high temperatures present in the hearth of the furnace (1500 °C and more).

For blast furnace ironmaking, the most important functions of coke are:

- To provide 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 (above 2000 °C), which is of particular importance in the hearth and melting and softening zone. Below the melting zone, coke is the only solid material in the blast furnace, so the total weight of the blast furnace content is supported by the coke structure. The coke bed has to be permeable, so that slag and iron can flow downward to accumulate in the hearth and flow to the taphole.
- To generate heat to melt the burden.
- To generate reducing gases to remove the oxygen attached to iron in the metallic burden.

- To provide the carbon for carburization of the hot metal (also called dissolution of carbon).
- To act as a filter for soot and dust.

The permanent efforts aimed at reducing the costs of ironmaking have led to an increasing portion of substitute reduction materials for coke, which has mainly been coal, injected through the tuyeres and recently natural gas in North America in particular. Nowadays, blast furnaces with total coal injection rates in excess of 200 kg/tHM are operated with coke consumptions of less than 300 kg/tHM. At these high coal injection rates, coke is subjected to more vigorous conditions in the blast furnace. Dissection of furnaces taken out of operation and probing and sampling through the tuyeres of furnaces in operation have allowed the assessment of the extent of coke degradation in the furnace. Coke degradation is controlled by the properties of feed coke, i.e. mechanical stabilization, resistance to chemical attack (solution loss, alkalis, and graphitization) and by the blast furnace operating conditions. At high coal injection rates, the amount of coke present in the furnace decreases and the remaining coke is subjected to more vigorous mechanical and chemical conditions: increased mechanical load as the ore/coke ratio becomes higher; increased residence time at high temperatures; increased solution loss reaction (CO_2 , liquid oxides); and alkali attack. More severe coke degradation during its descent from the furnace stockline into the hearth can therefore be expected at high coal rates.

However, high coal injection rates can also affect the direct reduction reactions.

1. Coal injection increases hydrogen content and at elevated temperatures (800–1100 °C), hydrogen is a very effective agent in gas reduction of iron oxides.
2. The unburnt soot (also known as char) remaining after the raceway is more reactive than coke and used for direct reduction in preference of coke.
3. The alkali cycle is reduced as a consequence of the elimination of alkali through the hot furnace centre.

Therefore, at high coal injection rates, the attack of coke by direct reduction reactions may also decrease. This is beneficial for coke integrity in the lower part of the furnace.

In Figure 4.2, we show the residence time of coke in various zones. In the “drying” zone, before the burden starts melting, the coke is exposed to the blast furnace gas flow for 3–6 hours. In the cohesive zone and the active coke zone, the coke is present for 2–4 hours and is subjected to high temperature and aggressive attack by chemical reaction (direct reduction reactions, alkali). In the hearth, the coke residence time is much longer, especially in the more central part called “deadman”. There, the residence time is from a week to two months. As soon as coke reaches the area below the tuyeres, there is no further or very little “chemical” consumption of coke, nor is there a large dissolution of carbon in the hot metal.

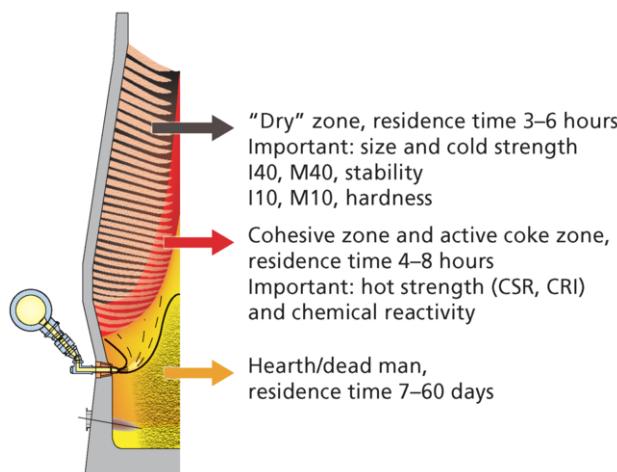


Figure 4.2 Residence time and quality parameters in various zones in the blast furnace

In this chapter, we will discuss coke quality parameters, test methods, degradation processes of the coke in the blast furnace, and finally the range of coke qualities targeted by blast furnaces that are currently operating or are aiming to operate at the highest production levels, so are more demanding in terms of coke quality.

4.2 Coal blends for cokemaking

The coal selected to make coke is the most important variable that controls the coke properties. The rank and type of coal selected impacts coke strength while coal chemistry largely determines coke chemistry. In general, bituminous coals are selected for blending to make blast furnace coke of high strength with acceptable reactivity and at competitive cost. For the conventional recovery coking process, the blend must contract or shrink sufficiently for easy removal from the oven and oven wall pressure must be acceptable to prevent coke oven damage. For the heat recovery method of cokemaking, these constraints are not valid, which leads to an increase in the number of usable coal types in this type of process. Good coking coals are expensive and therefore composing a coal blend that gives good coke and is low cost is a specialist's work. An introduction to coal selection for cokemaking is given in Annex 5.

Table 4.1 shows the typical chemical composition of coke that may be considered to be of good quality.

Typical Coke Analysis		% (db)
Coke Analysis	Fixed Carbon	87–92
	Nitrogen	1.2–1.5
	Ash	8–11
	Sulphur	0.6–0.8
	Volatile Matter (for well carbonised coke)	0.2–0.5
Ash Analysis	Silica SiO_2	52.0
	Alumina Al_2O_3	31.0
	Iron Fe	7.0
	Lime CaO	2.5
	Potassium K_2O	1.8
	Magnesia MgO	1.2
	Sodium Na_2O	0.7
	Phosphorous P	0.3
	Manganese Mn	0.1

Table 4.1 Coke chemistry for a typically acceptable coke quality grade

Ash directly replaces carbon, so higher ash content results in higher slag rates and lower energy values. The increased amount of slag requires energy to melt and more fluxes to provide a liquid slag. Ash, sulphur, phosphorus, alkalis and zinc can be best controlled by careful selection of all coal, coke and burden materials. The financial repercussions of ash, sulphur and phosphorus may be assessed by value-in-use calculations for PCI coal, coking coal blends and burden materials. Alkalis and zinc should remain below certain threshold levels (section 6.2).

4.3 Coke quality concept

Now the questions are how to characterize coke quality and how to define and measure the coke properties. In other words, how to establish a target for coke manufacturing based on determined coke properties in line with the needs of the blast furnace process. From the above discussion, the following parameters should be considered to limit the coke degradation and maintain suitable coke behavior in the blast furnace, especially at high coal injection rates.

Qualitatively, the coke should:

- Be made up of large, stabilized particles within a narrow size distribution band.
- Have a high resistance against volume breakage.
- Have a high resistance against abrasion.
- Have a high resistance against chemical attack (CO_2 , alkali).

- Have a high residual strength after chemical attack.
- Have sufficient carburization properties (the dissolution of carbon in hot metal).

A more detailed description of coke quality tests is added in Annex 6.

4.3.1 Coke degradation mechanisms in the blast furnace

The basic concepts of coke degradation in the blast furnace, according to the interconnected thermal, physical, and chemical conditions coke is subjected to in the furnace are described in Figure 4.3.

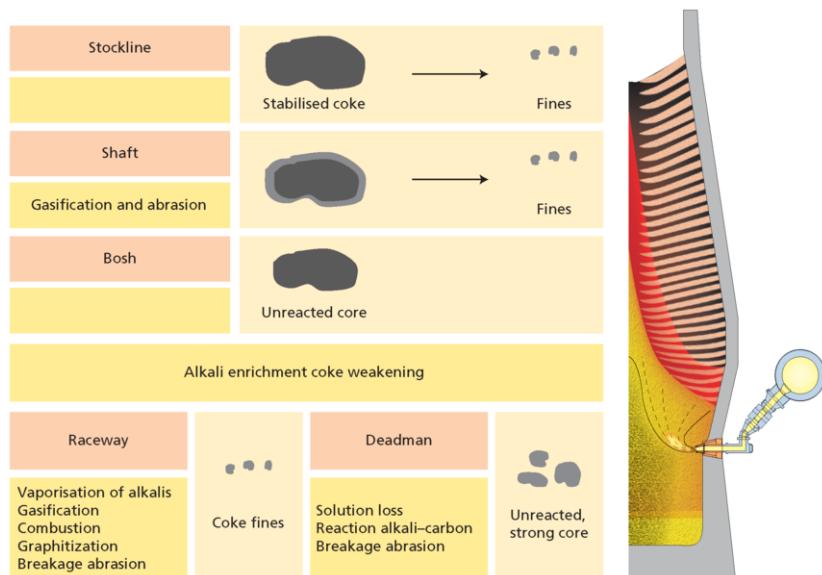


Figure 4.3 Basic concepts of coke degradation in a blast furnace

At the stockline, the coke is generally well-stabilized due to handling to get it to the furnace top. The effect of gasification on strength is controlled by the mechanisms of the heterogeneous reaction. In general, diffusion is the limiting step and the reaction is located at the surface of the lumps, the core remaining quite unaffected. As gasification and abrasion proceed simultaneously, a peeling of coke particles occurs (3–5 mm size reduction), leaving an exposed, unreacted core and fines.

Beyond gasification, coke reacts with alkali vapors when passing through the alkali circulating zone and the structure is penetrated by alkalis. This reaction reduces the strength of the coke, making it more susceptible to size reduction by breakage from mechanical action. Coke that has already been weakened, arriving in the high temperature zone of the raceway, loses its alkalis by gasification. High temperature, mechanical action and graphitization bring about severe degradation, decrease of size and formation of fines.

The coke travelling to the deadman is exposed to moderate temperatures, high alkalis during long periods of time along with additional reactions (reduction of slag, carburization) that mostly affect the surface of the coke lumps. Deadman coke, sampled by core drilling corresponds more or less with the unreacted core of the initial lumps and it is not surprising that it exhibits similar strength to that of the coke that is charged at the top.

4.3.2 Degradation of coke during its descent in the blast furnace

To discuss the phenomena leading to coke degradation during descent in the blast furnace we use Figure 4.4, representing the different zones of the process, the relevant process conditions and the evolution of the coke size under these conditions.

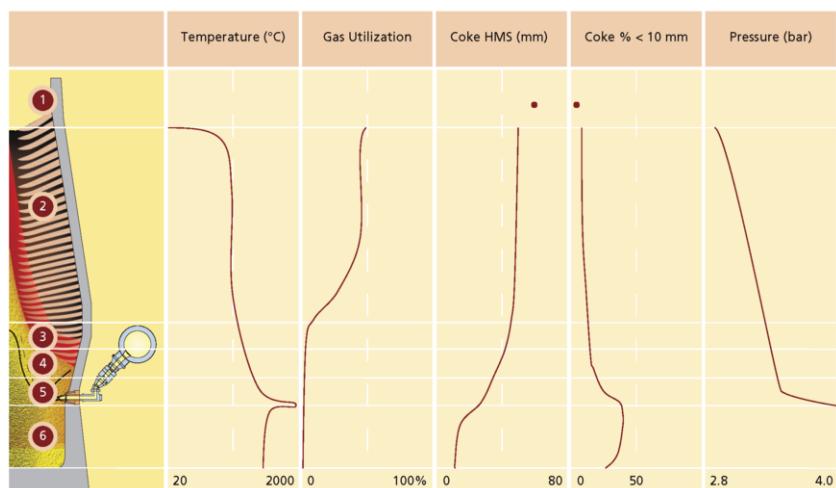


Figure 4.4 Development of coke size under the conditions that are present in the blast furnace throughout the journey from the top to the bottom of the furnace.

- Charging zone: Due to the fall of the coke onto the stockline, some breakage and abrasion will occur during charging.
- Granular zone: In this region the coke and ore remain as discrete particles within their separate layers. Drying occurs and recirculating elements such as zinc, sulphur and alkalis deposit on the burden materials as they descend to the bottom of the granular zone. From a temperature of 900 °C, coke starts to react with CO₂, continuing to do so as the temperature increases to over 1000 °C. In this zone, coke degradation (mostly abrasion) occurs due to mechanical load and mild gasification.
- Cohesive zone: This zone starts where ore agglomerates begin to soften and deform, creating a mass of agglomerate particles sticking together. This mass is barely permeable and the rising gas can only pass through the remaining coke layers. Coke gasification with CO₂ becomes significant due to increased

reaction rates at the higher temperature level (1000–1300 °C). The contact between the softened or molten materials and the coke lumps becomes more intensive, leading to increased mechanical wear on the outer surface of the coke particle. The residence time within the cohesive zone is rather short (30 to 60 minutes) depending on productivity and softening properties of the agglomerates.

4. Active Coke or Dripping zone: This is a packed bed of coke, through which liquid iron and slag percolate towards the furnace hearth. The coke particles play an active role in further reducing the remaining iron oxides and increasing the carbon content of the iron through dissolution of carbon from the coke into the iron. The bulk of the coke arriving in this zone (also referred to as bosh coke) flows towards the raceway region. The remaining part will move into the deadman (also called inactive coke zone). The residence time estimates vary from 4 to 8 hours. The temperature increases gradually from 1200 to 1500 °C.
5. Raceway: Hot blast containing oxygen is introduced through the tuyeres. The kinetic energy of the blast creates a raceway (cavity) in front of each tuyere. Coke particles circulate at very high velocity in this semi-void region while being gasified together with injectants such as coal, oil and natural gas. A part of the coke and injected reductants is not burnt completely. Soot is produced during injection of coal and natural gas. Soot and dust are transported upwards by the gas stream. They cover coke particles and react later following solution loss reaction. They decrease the reactivity of coke and cause an increase in apparent viscosity of liquid phases. The temperature increases rapidly to over 2000 °C due to the exothermic oxidation of coke and injectants. Coke fines and injectant fines that are generated in the raceway either completely gasify or get blown out of the raceway into the coke bed. Coke and coal fines may accumulate directly behind the raceway, forming an almost impermeable zone called the bird's nest. Observations of the raceway were made in blast furnaces in operation by inserting an endoscope through a tuyere. These observations showed that in this zone, the coke is subjected to very severe conditions.
6. The Hearth: Since the rate of coke consumption is the highest in the ring of the raceway, an almost stagnant zone (not directly feeding the raceway) develops in the furnace centre. This zone is called the deadman, and is thought to have a conical shape and a relatively dense skin structure. Molten iron and slag accumulates throughout the structure before being tapped through the tapholes. Tracer experiments in a German furnace gave values in the range of 10 to 14 days, but in literature also residence times of 60 days are mentioned for the deadman coke.

The above mentioned processes are summarized in Table 4.2 on the next page.

Blast Furnace Zone	Function of Coke	Coke Degradation Mechanism	Coke Requirements
Charging Zone		– Impact Stress – Abrasion	– Size Distribution – Resistance to Breakage – Abrasion Resistance
Granular Zone	– Gas permeability	– Alkali Deposition – Mechanical Stress – Abrasion	– Size & Stability – Mechanical Strength – Abrasion Resistance
Cohesive Zone	– Burden support – Gas permeability – Iron and slag drainage	– Gasification by CO ₂ – Abrasion	– Size Distribution – Low Reactivity to CO ₂ – High Strength after Abrasion
Active Zone	– Burden support – Gas permeability – Iron and slag drainage	– Gasification by CO ₂ – Abrasion – Alkali attack and ash reactions	– Size Distribution – Low Reactivity to CO ₂ – Abrasion Resistance
Raceway Zone	– Generation of CO	– Combustion – Thermal Shock – Graphitisation – Impact Stress and Abrasion	– Strength against Thermal Shock and Mechanical Stress – Abrasion Resistance
Hearth Zone	– Burden support – Iron and slag drainage – Carburisation of iron	– Graphitisation – Dissolution into hot metal – Mechanical Stress	– Size Distribution – Mechanical Strength – Abrasion Resistance – Carbon Solution

Table 4.2 Coke functions, degradation mechanisms and requirements

4.4 Coke size distribution

The shape of the coke particles and the size distribution of the particles are the decisive factors for the permeability of the coke bed, for ascending gas as well as for the descending liquids. Research has shown that the harmonic mean size (HMS) of the coke mass gives the highest correlation with the resistance to flow of gas passing through the coke bed. HMS is the size of uniform size balls with the same total surface as the original coke size mixture.

The lowest flow resistance is obtained when large coke is being used of high uniformity. Fines in particular have a strong decreasing effect on the harmonic mean size, which increases the bulk resistance of the coke. Excellent blast furnace operations are reported with screening at 24 mm (square screen openings) but there are also plants where screening at up to 40 mm is preferred.

Once the bulk coke has been classified by screening and crushing (see also Figure 4.4), the aim is to have a resulting coke with a high mechanical strength under the blast furnace conditions. This is to prevent an excessive formation of coke fines during its descent in the blast furnace.

4.5 Strength of coke

4.5.1 Coke particle size during the blast furnace process

During carbonization in a coke oven, fissures in the coke are generated due to stresses that arise from the differential contraction rates in adjacent layers of coke, which are at different temperatures. Typically, they are longitudinal, i.e. perpendicular to the oven walls. Additionally, many transverse fissures are formed during pushing. These fissures determine the size distribution of the produced coke by breakage along their lines during subsequent handling. But not all the fissures lead to breakage at this early stage, and a number of them remain in the coke particles. The initial coke size distribution is a function of the coal blend and the coking conditions. A significant number of internal fissures remain present and cause further degradation under mechanical loads during transport and charging of the blast furnace. This process of coke degradation is called stabilization. Stabilization lowers the mean size of the coke, but the resulting particles are less prone to further breakage. For blast furnace performance, it is not only important to have large, stabilized and narrow size distribution coke charged into the furnace, but it is even more important to have the same qualities present during its descent through the furnace. With mechanical handling, coke particles will degrade due to breakage and abrasion. Breakage is the degradation of coke by impact due to fissures already present in the coke. Abrasion is the degradation of the surface by relatively low impact processes (rolling and sliding against other particles or the furnace walls). It is one of the main mechanical processes for decreasing the coke size below the stock line, next to breakage in the raceway area. Abrasion causes the formation of fines, which may hamper blast furnace permeability.

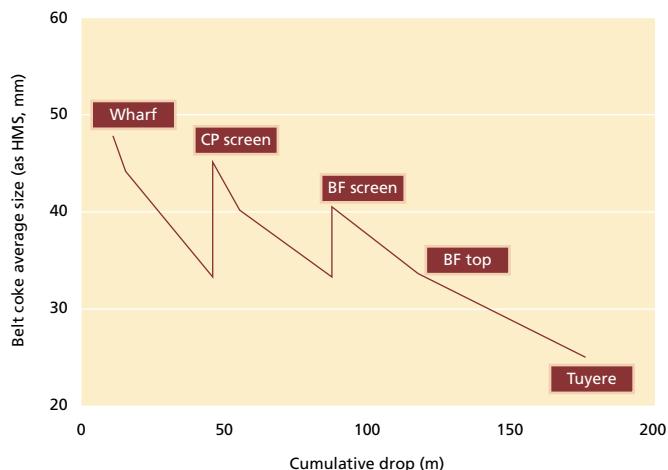


Figure 4.5 Development of Harmonic Mean Size after mechanical handling in the form of drops between conveyors and screens.

The resistance to abrasion will deteriorate in the blast furnace, due to reactions such as graphitization, gasification and carburization of the iron. Graphitization results in a more crystalline form of carbon in the coke that is more brittle. In Figure 4.5, the typical development of the HMS of coke from the coke wharf to the tuyeres is presented.

In the presented transport route, the coke is screened at 35 mm (square) at the coke plant and at 24 mm (square) at the blast furnace. The increase in HMS of the sample after screening is due to the removal of the undersized coke from the batch.

4.5.2 Coke strength simulation tests

Although it is known that coke degrades more rapidly at high temperatures, there is no test in practical use that is performed at high temperatures. Not only because of the complexity and high cost but also since it has been proven that coke with poor low temperature strength also exhibits poor strength at high temperatures. Therefore, most tests in practical use are done at ambient temperature.

Coke strength is traditionally measured by empirical tumble indices. During mechanical handling, coke size degradation takes place by two independent processes, those being breakage into smaller lumps along fissures and cracks still present in the lumps, and abrasion at the coke surface resulting in small particles (below 10 mm). It is common to measure a ‘strength’ index related to degradation by volume breakage (for example, I_{40} , M_{40} , stability index) and an ‘abrasion’ index (for example, I_{10} , M_{10} , D^{150}_{15}). These empirical indices cannot be directly related to fundamental coke properties.

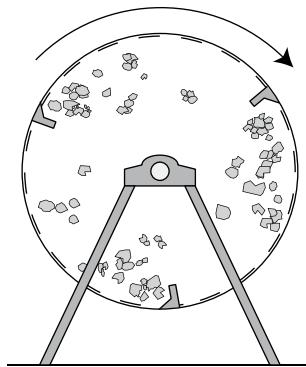


Figure 4.6 Schematic showing the motion of coke in a tumble test

Figure 4.6 shows a schematic representation of particle motion in a tumble drum. As lifter bars inside the drum rotate, a portion of the coke is lifted off the walls of the drum. Some of the coke rolls off the lifter before it reaches the horizontal plane. The coke that is not picked up slips and rolls against the

bottom of the drum. The coke that is lifted past the horizontal is dropped over a fairly narrow angular range as the lifter approaches the vertical plane. This coke impacts with the bottom of the drum. Tests have shown, that there is a relationship between the degradation of coke in a drum test and that after a number of drops. This makes it possible to translate the effect on coke size after a number of drops, in meters, into a number of rotations in a drum, and vice-versa.

4.5.3 Coke hot strength and chemical reactivity

Chemical reactivity

Besides a high mechanical strength, coke should have a high resistance against chemical attack. There are two measurements for coke after the reaction with CO: the CRI (Coke Reactivity Index) and the CSR (Coke Strength after Reaction).

Coke Reactivity Index

Reactivity of coke can be tested in numerous ways, but by far the most common way to determine the coke reactivity is the Nippon Steel Chemical Reactivity Index (CRI). With this test, coke of a certain size is put under a 100 % CO₂ atmosphere at 1100 °C. The percentage of coke that is gasified after 120 minutes gives the CRI value. The more reactive the coke, the higher the mass loss will be. Reactivity of the coke is mainly determined by the chemical composition of the parent coal blend, because ash components act as catalysts for the reaction of C with CO₂.

Coke Strength after Reaction

Due to the loss of mass whilst under attack by CO₂, the surface layer of the coke particles get very porous and the mechanical strength against abrasion drops rapidly. To measure this effect, the reactivity test is normally followed by a tumbler test to determine the residual coke strength. The percentage of particles that remain larger than 10 mm after 600 rotations is called the 'Coke Strength after Reaction' or CSR index. For most coke produced, there exists a strong correlation between CRI and CSR. As CRI decreases, CSR increases.

Before CRI and CSR were developed, a series of relatively expensive tests were carried out under various research projects that involved partially gasifying the coke in its original particle size under realistic blast furnace conditions before subjecting it to the standard drum test. While the results of this costly research work showed exactly how the coke in the blast furnace was subjected to chemical attack, it provided no better information on coke quality than the more simple methods of determining CRI and CSR. These two parameters are now generally adopted by the cokemaking industry as the most important parameters for determining coke quality, especially on large blast furnaces with high productivity and high tuyere injectant rates.

4.6 Coke deadman

The coke “deadman” is a cone-shaped coke layer, located below the cohesive zone and active coke zone (Figure 4.7). The coke in this zone comes from the coke charged into the center of the furnace. This coke is not or to only a very limited extent exposed to attack by carbon dioxide and consequently, its strength properties are not degraded. It more or less maintains its size as charged, only affected by slight abrasion on its way down through the furnace. Therefore, coke lumps in the deadman are bigger than those in coke layers more towards the wall. Some companies use stronger and/or larger coke in the center to enhance deadman permeability.

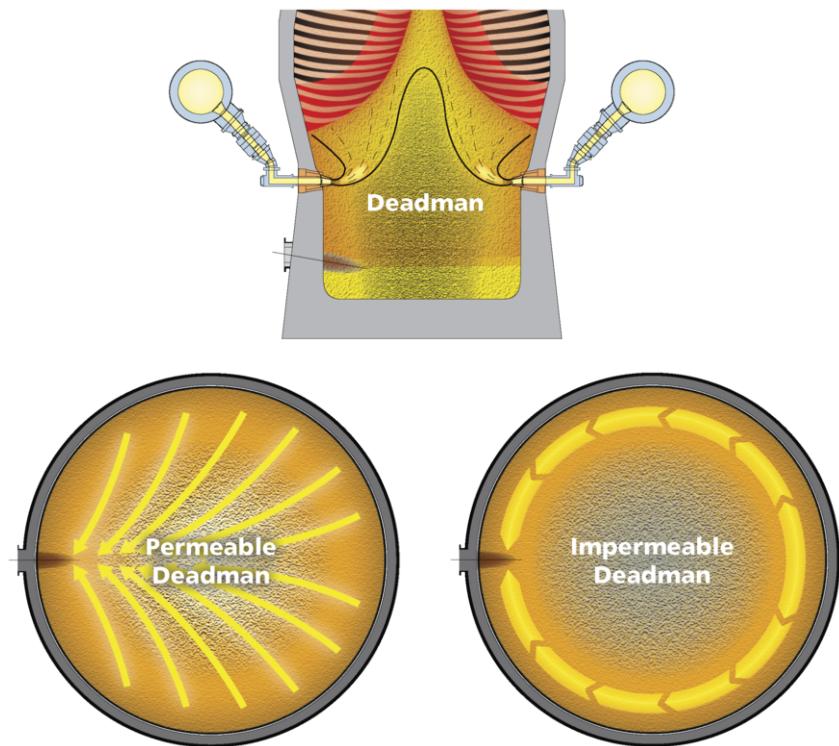


Figure 4.7 The coke deadman and the effect of a permeable deadman and impermeable deadman on hot metal flow lines towards the taphole

The main requirement for the deadman is, that it has sufficient porosity to ensure:

- From the level of the tuyeres upwards: unrestricted passage of gas from raceway.
- From the level of the tuyeres downwards: good permeability for liquids on its way to the hearth.
- Within the hearth, below the taphole level: the deadman is resting on the bottom and its permeability for hot metal and slag flowing towards the taphole determines the flow pattern of the liquids.

Figure 4.7 shows two patterns of hot metal flow in the hearth: flow through the deadman (left) and flow around the deadman (right). When the flow through the deadman is blocked, the flow will take place along the wall of the furnace. This will lead to increased wear of the hearth refractory lining and the hearth refractory reaches its end of campaign more quickly.

Drainage capacity of the deadman can be quantified using the Deadman Cleanliness Index (DCI). The DCI is based on the observation that in case of poor contact between the liquids and the coke, the carbon content of the hot metal deviates from the calculated equilibrium value (Nightingale, 2000; Bonte et al, 2005). An empirical formula is available, taking into account hot metal temperature (T_{hm}) and composition and slag basicity ($B = \text{CaO}/\text{SiO}_2$):

$$\text{DCI} = 2T_{hm} - 121[\text{Si}] - 128[\text{P}] - 156[\text{S}] + 11[\text{Mn}] - 389 [\text{C}] - 190B - 690.$$

For furnaces having volumes from 2000 to 4300 m³, using natural gas injection and PCI, DCI varies within the range of 150–250 with the higher values resulting in better drainage capacity of the dead man.

The coke lumps in the hearth are slowly consumed. When the burden melts, a “primary melt” or “primary slag” is formed consisting of a large percentage of FeO and the gangue of the burden materials. From this melting material, hot metal separates. In the contact between the melt and the deadman coke, a chemical reaction takes place between the carbon of the coke on the one hand and hot metal and slag on the other hand.

- Coke reacts with iron oxide compounds which have the lowest melting temperatures, like calcium–iron olivines ($n\text{FeO}\cdot\{2-n\}\text{CaO}$) and fayalite (Fe_2SiO_4) generating CO in the process and forming the basis of the “primary” slag.
- Coke allows for desulphurization of the hot metal:
$$[\text{FeS}] + (\text{CaO}) + \text{C} \rightarrow [\text{Fe}] + (\text{CaS}) + \text{CO}$$
- Dissolution of carbon in hot metal:
$$3[\text{Fe}] + \text{C} \rightarrow [\text{Fe}_3\text{C}]$$

As a result of carbon consumption and dissolution of coke ash in slag, coke lumps gradually decrease in size and are completely consumed. There is a slow and continuous replacement of coke in the deadman, which is supplemented with fresh coke descending from the top. Residence time of coke in the hearth is estimated as about 2–3 weeks and depends on the blast furnace volume and performance.

The inactivation of the deadman (also called “dirty deadman”) can be identified from at least the following symptoms (Raipala, 2003):

1. Decrease of central temperature in the hearth bottom.
2. Hearth wall temperature increase.
3. Oxygen potential increase in slag (FeO and MnO in slag), resulting in poorer desulphurization of the hot metal.

4. Hot metal carbon decrease.
5. Slag tapping ratio decrease.
6. Hot metal temperature is higher than usual with the same energy consumption and hot metal composition.
7. Shortening of the race way can be observed.
8. Shortening of the tap hole length.

Causes of inactivation of the deadman can be attributed to:

1. Coke properties: coke particles are exposed to mechanical stress in the stack and in the lower part of the blast furnace. Large and strong coke results in larger particle size and less fines in the hearth. Coke lumps are weakened in the stack by solution loss reaction ($C + CO_2 \rightarrow 2 CO$) and the impact on various coke qualities depends on its hot strength such as CSR/CRI.
2. Water leakage: Small leakages (tuyeres, cooling plates, stave circuits), especially when they remain undetected, may lead to an inactive deadman when water does not evaporate but seeks its way to cooler parts in the furnace bottom.
3. Maintenance stops (shutdowns): when the hot metal below the taphole level solidifies, it has a large mass and good thermal conductivity to the cooled bottom. Excessive hearth cooling systems increases the risk of inactivation during blast furnace stops.
4. Low production rate: when the production rate is low, the residence time of coke in contact with CO_2 and alkalis becomes longer and the coke properties weaken. Heat losses through the hearth lining are relatively constant at the beginning of the slow production period. Heat transport to the deadman slows down, and a risk of solidification at the bottom increases.
5. Hearth cooling: hearth cooling itself is an essential factor in problems with an inactive deadman. Because the clogged or dirty deadman is maintained by solidified matter, it requires cooling to keep the temperature low.

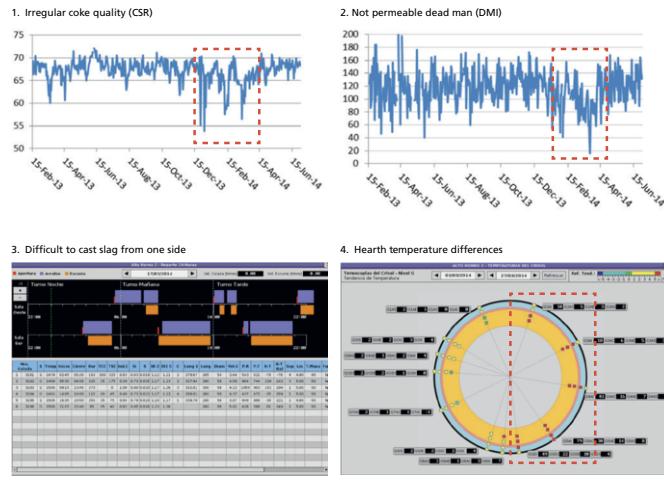


Figure 4.8 Poorer coke quality and effect on DCI, showing increased sidewall temperatures and poorer casting on 1 taphole

Measures to activate a deadman include :

1. Increase the coke quality (no small coke in the burden).
2. Large lump size of centrally charged coke (over 60 mm).
3. Increase coke rate and reduce PCI rate.
4. Reduction of hearth bottom cooling.

An example of deadman inactivation is shown in Figure 4.8, where a period with poor coke quality coincided with low values for the DCI. The effects were also visible in poorer casting behavior as well as higher sidewall temperatures.

4.7 Overview of international quality parameters

Figure 4.2 and Table 4.3 give an overview of typical coke quality parameters and their generally accepted levels for a ‘good’ coke quality. Although not complete, the values given in the table represent coke qualities that have assisted in securing excellent blast furnace results over a long period.

We have to stress, however, that blast furnace operation is very much influenced by coke variability: the gas flow in the furnace can only be held consistent if the layer build-up is consistent and if day-to-day consistency of the coke is very good. There are, however, no international standards or criteria for day-to-day consistency.

	What is measured?	Results	Accept. Range	Best	Reference
Mean Size	Size Distribution	AMS mm HMS mm % < 40 mm % < 10 mm	40–60 35–50 < 25 < 2		
Cold Strength	Size Distribution after Tumbling	I_{40} % > 40 mm I_{10} % < 10 mm M_{40} % > 40 mm M_{10} % < 10 mm	> 45 < 20 > 80 < 7	60 16 87 5.5	Irsid Test Micum Test
	Stability at Wharf Stab. at Stockh. Hardness	% > 1" % > 1" % > 1/4"	> 58 > 60 > 70		ASTM Test
Strength after reaction	CSR	% > 9.52 mm	> 58	70	Nippon Steel Test
Reactivity	CRI	% weight loss	< 29	22	Nippon Steel Test

Table 4.3 Acceptable range for coke quality parameters (for tests, see Annex IV)

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V *Injection of Coal, Oil and Gas*

The coke rate of a modern, big blast furnace operating with only coke as a reductant (fuel) is typically 500–520 kg/tHM. The coke rate can be decreased by injection of coal, oil and natural gas (and other hydrocarbons), as was suggested already back in 1831 by John Samuel Davis. Use of natural gas injection started in 1957 in the Ukraine and the technology spread quite quickly especially in the USSR. Starting in the 1960s, fuel oil injection was implemented, while industrial coal injection started at Amanda furnace in the USA in the 1960s. The technology of coal injection spread quite quickly, especially from the 1980s. The recent low cost of natural gas has led to the application of co-injection of coal and natural gas especially in North America.

Since a major part of the hot metal cost is accountable to coke, use of injectants is economically attractive. Coke rates can be decreased to about 400 kg/tHM by injecting natural gas, about 375 kg/tHM by injecting oil and to 320 kg/tHM and below by injecting coal. The investments for the application of coal injection are considerable, while those for gas or oil injection are low.

The apparent versatility of various type of injectants comes from the fact that in the raceway, the temperatures are so high, that the injectants and moisture are all converted to carbon monoxide and hydrogen within milliseconds after injection, and that the blast furnace process is not influenced by where the hydrogen and carbon monoxide are coming from.

5.1 **Properties of coal, oil and gas**

The energy inputs and outputs of the blast furnace are schematically shown in Figure 5.1 on the next page. The major sources for energy in the furnace are the coke and injectants (coal, gas, oil) and the sensible heat of the hot blast. The major part of the energy is used to drive the change from iron oxides to iron and the other chemical reactions. The remaining energy leaves the furnace through the top gas, as sensible heat of iron and slag and as heat losses to the cooling system.

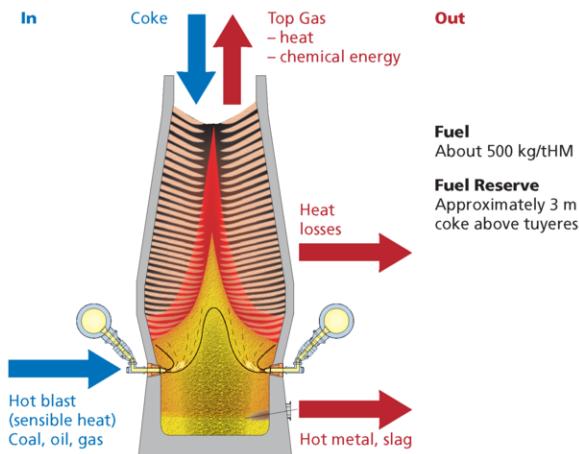


Figure 5.1 Schematic overview of energy inputs and outputs

The differences between the various injectants are based on their chemical compositions. Oil has no ash and when using oil, the carbon–hydrogen bonds in the oil are replaced by the carbon–oxygen bonds of carbon monoxide and less heat is generated than when using coal. This is even more extreme when using natural gas (mostly methane, CH₄), which has more or less twice the hydrogen content of oil and four times the hydrogen content of coal, so injection of natural gas generates even less heat in the raceway.

In the lower part of the furnace, in and below the cohesive zone, there is a need for energy to:

- Heat up and melt the material from where it starts softening, about 1100 °C, to casting temperature of 1500 °C.
- Remove the remaining oxygen from the ore burden and other hot metal components like silicon (see section 8.3.3 for direct reduction).

The largest amount of energy (more than 70%) is used in “Direct Reduction”. However, the amount of direct reduction that has to take place also depends on the amount of hydrogen generated at the tuyeres since hydrogen is very efficient in removing oxygen from the iron at high temperatures (900–1100 °C). As a consequence, the most important properties of injectants are the amount of heat generated when gasified to carbon monoxide (Table 5.1) and the hydrogen content of the gas (for more details see section 5.3 on gas injection). Another effect is that the gas volume generated at the tuyeres increases, when more hydrogen is present. This is shown as the gas volume generated by gasifying 1 kg of injectant in Table 5.1.

	MJ/kg	as % of coke	gas (m³ STP/kg)	C content (%)	H content (%)
coke	8.6	100	1.7	87.5	0.2
coal	6.7	78	2.1	80.0	5.0
natural gas	2.3	26	4.1	73.0	24.0
oil	6.5	76	2.8	86.6	11.6

Table 5.1 Heat of incomplete combustion to CO and H₂ and raceway gas volume by gasifying 1 kg injectant

Coal generates much more heat: when burning coal with pure cold oxygen, it generates a Flame Temperature of around 2000 °C, while doing the same with natural gas generates a Flame Temperature of around 850 °C.

	T _{flame} (°C)	H ₂ in gas (%)	CO in gas (%)
coal	-2200	25	75
natural gas	850	66	33

Table 5.2 Flame temperature and gas composition when pure, cold oxygen gasifies coal and gas to CO and H₂

The thermal and gas volume effects of incomplete combustion of different injectants differ from each other. A typical example is shown in Table 5.3. The table is based on the amount of injectant needed to replace 10 kg of coke. It shows the larger effect of natural gas on flame temperature. Precise results depend on composition of the injectants as well as process conditions.

	coal	nat. gas	oil
required to replace 10 kg coke (kg)	11.4	9.7	9.0
T _{flame} effect (°C)	-30	-70	-46
composition			
C content (%)	80.0	73.0	86.6
H content (%)	5.0	24.0	11.0
O content (%)	8.0	0.6	
ash content (%)	5.0		

Table 5.3 Replacement ratio of 10 kg coke with different injectants; furnace operating at 10,000 tHM/d at 26% O₂ in hot blast

The consequence is that the lowest coke rates can be reached with coal. Table 5.4 on the next page shows the worldwide experiences with different injectants, based on about 70 % of the maximum reached. Good, but not excellent levels of injection are around 170 kg/tHM for coal, around 90 kg/tHM for oil and around 80 kg/tHM for gas. Another effect is that when using natural gas, higher oxygen enrichment of the hot blast is required to keep the flame temperature within the required working area.

Base scenarios		Coal HV	Fuel Oil	Natural Gas
Coke rate (87,5 % C)	kg/tHM	324	379	408
Coal	kg/tHM	170	0	0
Oil	kg/tHM	0	90	0
Natural Gas	kg/tHM	0	0	80
Oxygen	m ³ STP/tHM	92	60	85
BF Gas Export	MJ/tHM	3695	3384	3976
BF Gas Calorific Value	MJ/m ³ STP	3920	3609	3966

Table 5.4 Typical average coke rates using coal, oil and gas injection.

5.2 Coal injection

5.2.1 Coal injection: equipment

The basic design of a coal injection installation is indicated in Figure 5.2.

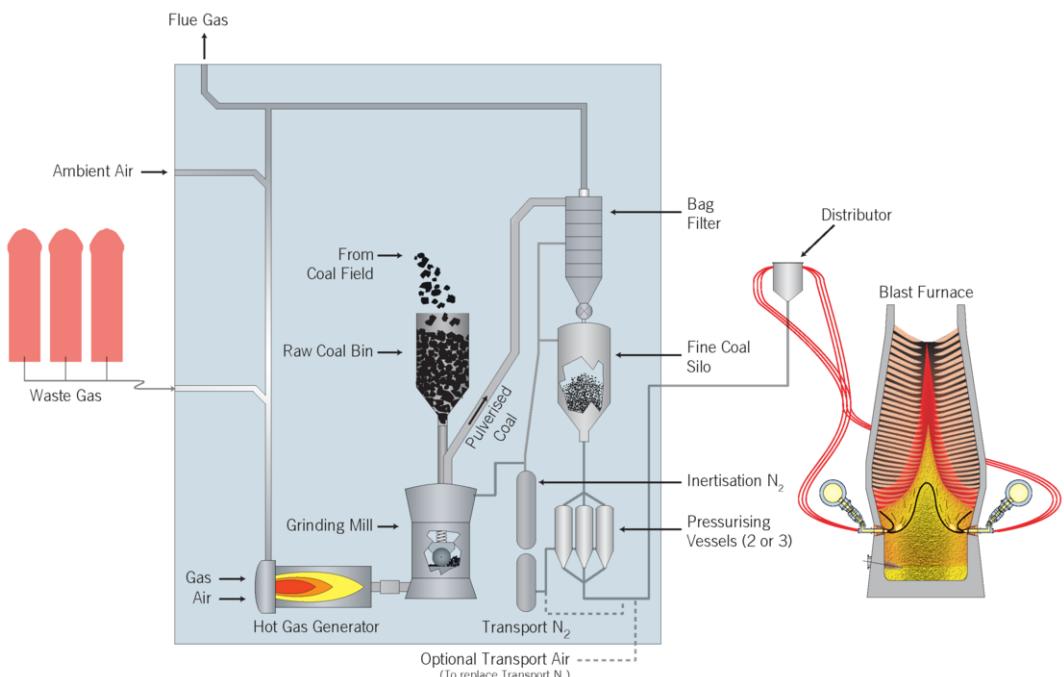


Figure 5.2 Example of PCI installation

These installations require the following functions to be carried out:

- Grinding of the coal. Coal has to be ground to very small sizes. Most commonly used is pulverized coal: around 60 % of the coal is under 75 µm (like powder). Granular coal is somewhat coarser with sizes up to 1 to 2 mm (like beach sand).

- Drying of the coal. Coal contains substantial amounts of moisture, from 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 or contains moisture, the pneumatic transport will be hampered. It may result in formation of minor deposits on the transportation pipe walls and also lead to coal leakage from the transportation pipes.
- Injection of the pulverized coal. Coal has to be injected in equal amounts through all the tuyeres. Particularly at low coke rates and high productivities, the circumferential symmetry of the injection should be maintained.

There are various suppliers available for pulverized coal injection (PCI) systems, 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, since coal injection can be from 20 to 45 % of the total fuel.

5.2.2 Coal specification for PCI

5.2.2.1 Coke replacement

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, between 12 and 30 % is mid volatile and anything over 30 % high volatile. All types of coal have successfully been used. The most important property of the injection coal is the 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. The replacement ratio of coal can be calculated with a mass and heat balance of the furnace.

The chemical composition of the coal (i.e. carbon percentage, hydrogen percentage, oxygen content and ash content), the remaining moisture and the heat required to crack the coal chemical structure (especially the carbon–hydrogen bonds) have to be taken into account. Young coals have relatively high oxygen percentages, which means that the heating value of the coal is lower, since the carbon–oxygen bonds have already been formed in the coal structure. A simplified formula for the replacement ratio (RR, compared with coke with 87.5% carbon) is:

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

This formula shows, that the coke replacement depends on carbon and hydrogen content of the coal. Any remaining moisture in the coal consumes energy introduced with the coal. The positive factor of the ash content comes from a correction for heat balance effects.

5.2.2.2 Coal quality

Besides the carbon and hydrogen content, the most important parameters for coal quality to be injected are the following:

- Composition. High sulphur and high phosphorus 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 a lower replacement ratio in the process.
- Hardness. The hardness of the coal, characterised by the Hardgrove Grindability Index (HGI) must correspond with the specifications of the grinding equipment. The resulting size of the ground coal is also strongly dependent on this parameter and must correspond with 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.
- Chloride content. The chloride content of coal can cause corrosion of the steel in the blast furnace gas cleaning system. Many blast furnaces use coal coat the inside of the gas scrubber with a corrosion resistant paint or epoxy coating.
- Alkali (K_2O and Na_2O) content. The alkali in the coal for injection should be considered in the total alkali loading into the blast furnace, which is known to attack the refractory lining.

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 multiple types of coal to dilute unfavorable properties (section 5.2.2.4).

5.2.2.3 Coal types

There are many different types of coal. Among others, coals are characterized by the ash content and volatile matter content: the longer coals are left at high temperatures in the earth crust, the more volatiles escape. The special aspect in a blast furnace is, that the most valuable heat is generated by the incomplete combustion of coal to carbon monoxide (and hydrogen), while in a power plant the coal is combusted completely to carbon dioxide and water (H_2O). The carbon monoxide and hydrogen are used to reduce the oxygen content of the iron ore and are called reductants.

Since in the lower part of the furnace we need heat (enthalpy) at a high temperature, the most appropriate coals are the coals, that have lower ash content as well as lower oxygen content in the structure. This is because the less oxygen bonds with carbon that are present, the more heat is generated when forming carbon monoxide. From the point of view of heat generation, the better coals are low volatile, low ash coals.

Table 5.5 shows a typical example of the replacement ratios of LV (low volatile) and HV (high volatile) coals. The calculation has been done to show the effect of coal type on oxygen enrichment and productivity. To this end, the effect is shown in a 14 meter furnace, adjusting the production rate to a maximum Δp (1.7 bar) and adjusting the top temperature to a minimum of 110 °C by adjusting oxygen enrichment at a constant coke rate of 300 kg/tHM. The results show, that based on the low volatile and high volatile coals below, the process:

- Requires less coal when using LV coal (-16 %)
- Requires less oxygen (-30 %) with LV coal and
- Can get a slightly higher productivity (+1 %) with HV coal.

		HV coal	LV coal			HV coal	LV coal
production	t/d	11,450	11,360	volatile	%	36.7	16.0
coke rate	kg/tHM	300	300	C	%	79.9	86.7
coal rate	kg/tHM	207	174	H	%	5.3	4.1
O ₂ in blast	%	32.8	28.6	O	%	8	2
O ₂ from ASU	m ³ STP/tHM	117	81	N	%	1.3	1.1
Top temperature	°C	110	110	S	%	0.7	0.7
T _{flame}	°C	2222	2302	Ash	%	4.7	5.4
Δp	bar	1.7	1.7	Q _{decomp}	GJ/t	1.1	-0.3
				Moisture	%	1	1

Table 5.5 Productivity effect of LV and HV coals on process results (at identical Δp and top temperatures)
ASU = Air Separation Unit

It has also been shown, that high volatile coals are easier to gasify in the raceway. For this reason, for many years, high volatile coals were the preferred coals, especially in Europe and the USA. Although the experience in China shows that LV coals can be used very efficiently.

5.2.2.4 Coal blending

Most companies use coal blends for injection. Blending allows for (financial) optimization of coal purchases. E.g. a company with a grinding mill for hard coals can use a considerable percentage of softer coals by blending it into hard coals. In doing so, an optimized value can be obtained. Blending dilutes the disadvantages of some coal types. Every material has disadvantages like high moisture content, sulphur or phosphorus level, a relatively poor replacement ratio and so on. The blending can be done rather crudely. Depositing materials in the raw coal bin by alternating truck loads will be sufficient. Proper control of coal logistics and analysis of coal blend have to be put in place. In some blast furnaces, further cost reductions have been realized by blending lignite (brown coal), charcoal powder, petroleum coke powder or anthracite powder with the normal coal blends.

5.2.3 Coal injection in the tuyeres

Coals are injected via lances into the tuyeres, then gasified and ignited in the raceway. The coal is in the raceway area only for a very short time (5 ms), 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 5.3. First, the coal is heated and the moisture evaporates. Gasification of the volatile components then occurs after further heating. The volatile components are gasified and ignited, which causes an increase in the temperature. All of these steps occur sequentially with some overlap.

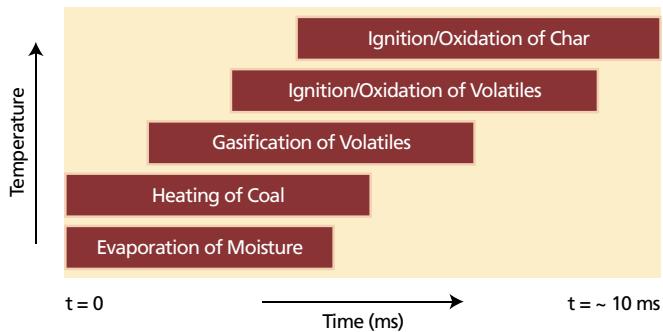


Figure 5.3 Coal gasification after injection

The effects of lance design, extra oxygen and coal type on the coal combustion have been analyzed. Originally, the coal lances were straight stainless steel lances that were positioned at or close to the tuyere/blowpipe interface as indicated in Figure 5.4.

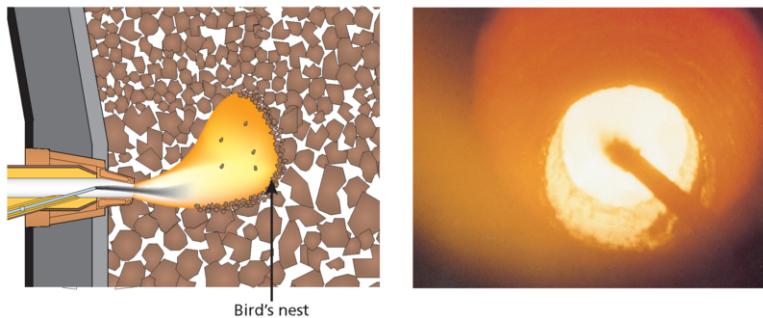


Figure 5.4 Coal injection in the tuyeres and lance positioning

Occasionally, very fine carbon soot formed from gas is detected at the furnace top. To avoid this problem, especially at high injection rates, companies have installed different types of injection systems at the tuyeres, such as:

- Co-axial lances with oxygen flow in an outer co-centric pipe and coal flow in an inner co-centric pipe.
- Specially designed lances with a special tip to cause more turbulence at the tip.

- Use of two lances per tuyere at different entry positions or angles to cause more turbulence.
- Bent lance tips positioned more inwards in the tuyere (closer to the tuyere nose).

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. The rate of gasification increases as:

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

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. High volatile (HV, over 30 % VM) and ultra high volatile (over 40 % VM) coal produce 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 packed coke region has been termed the “bird’s nest” (see Figure 5.4).

5.2.4 Oxygen and PCI

At high levels of pulverized coal injection, 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 accurately as the coke rate is controlled. The feed tanks of the coal injection system are weighed continuously and the flow rate of the coal is controlled. This 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/min), 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 a process 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 still has 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 1500 °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.

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 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 5.5.

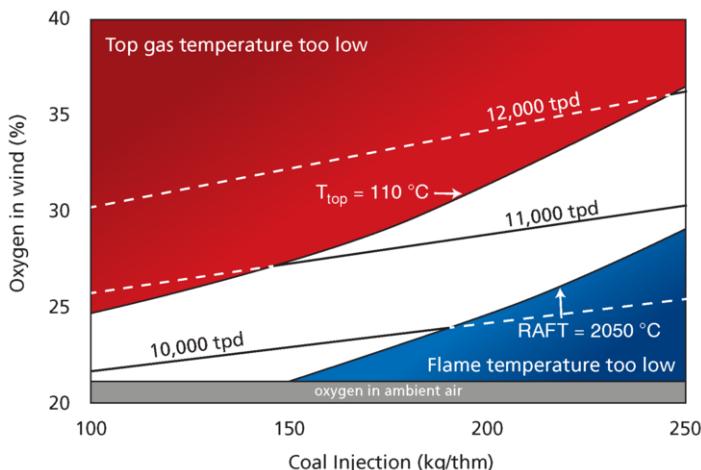


Figure 5.5 Working area showing limiting factors affecting raceway conditions with Pulverized Coal Injection
(RAFT = Raceway Adiabatic Flame Temperature)

For higher injection rates, more oxygen is required. The limitations are given by:

- Minimum 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 decreases. Deposits of wet dust particles in the first stage of gas cleaning (dust catcher or cyclone) can result in plugging of the dump valves.
- Maximum flame temperature. If the flame temperature becomes too high, burden descent can become erratic.
- Minimum flame temperature. Low flame temperatures will hamper coal gasification and melting of the ore burden. In Figure 5.5, 2050 °C is the minimum Flame Temperature.
- Technical limitations to the allowed or available oxygen enrichment in the steel plant oxygen production facility or pipeline size or pressure.

The higher the oxygen injection, the higher the productivity of the furnace can be, as shown in Figure 5.5, which is an example based on the mass and heat balance of an operating furnace. The highest productivity is reached with a maximum oxygen level, while keeping the top gas temperature at a minimum. This minimum is the level, where all water of the coke, burden and process is eliminated from the furnace, i.e. slightly above 100 °C. From a technological perspective, it can be said that the heat balances over the lower part (i.e. from 900 °C to tuyere level) and upper part (i.e. from top to the 900 °C isotherm) of the furnace are in balance (Section 8.5). In operational practice, the availability of oxygen in a plant often is the limiting factor.

5.2.5 Effect of additional PCI

The effect of the use of extra coal injection for recovery of a cooling furnace is two-fold. By putting extra coal on the furnace, the production rate decreases. Simultaneously, the flame temperature drops. 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. This is illustrated in Table 5.6.

Starting Situation	
Operating parameters	
Coke rate	300 kg/tHM
Coal injection rate	200 kg/tHM
Replacement ratio	0.85 kg coal/kg coke
Flame temperature	2200 °C
Coke and coal consumption in normal operation (as kg standard coke/tHM)	
Coke introduced	300
Coal introduced	170
Total coke and coal	470
Consumption to be subtracted to determine burn rates:	
Carbon in hot metal	-50
Direct reduction	-120
Result: total burn rate in front of tuyeres	300
of which coal	170
and thus coke	130
Changed situation if an additional 10 kg/tHM of coal is injected	
Total burn rate remains	300
of which coal	178.5
and thus coke	121.5
Production rate decrease (fully determined by coke burn rate)	6.5 %
Flame temperature drop	32 °C
Gas melting capacity drop (heat > 1,500 °C)	4.6 %

Table 5.6 Effect of additional coal injection

The table shows that additional coal injection slows down the production rate, because the coke burning rate decreases. It is a typical example; the precise effect depends on coke rate and coal type used. A furnace recovers from a cold condition by increasing PCI, because it slows down the production rate. If, however, the flame temperature is relatively low, the effect of the drop in flame temperature can be as large as the effect of the decreased production rate.

5.3 Natural gas injection

Natural gas typically consists of methane with some minor quantities of other hydrocarbons and nitrogen. Most frequently, natural gas injection is applied in the range of 60 to 90 kg/tHM, although higher levels up to 150 kg/tHM (Agarwall et al, 1999) or on other furnaces more than 100 kg/tHM (see e.g. Lingiardi et al, 2001) have been reached. The replacement ratio of natural gas is typically 1–1,05 kg coke/kg natural gas and this replacement ratio is non-linear, so decreases as the natural gas rate increases.

5.3.1 Injection systems for natural gas

Natural gas is mostly injected into the tuyeres with one lance per tuyere, but it can also be injected through a port in the tuyere copper casting sidewall. Natural gas is injected into the tuyeres at ambient temperatures and is gasified in the tuyeres. The residence time of the gas in an oxidizing atmosphere of the raceway is very short: 4–7 ms. Some companies apply pre-mixing of gas and oxygen and injecting with gas–oxygen mix lances aimed at complete combustion in the raceway.

In order to keep the natural gas rate constant on the basis of kg/tHM, the production level of the furnace has to be monitored and calculated constantly, so that the total flow control can be adjusted in real time. The control system of natural gas normally has a flow measurement and control valve for injection in each tuyere.

In contrast to PCI, an injection system for natural gas is relatively simple and requires limited investments.

5.3.2 Process control with natural gas

5.3.2.1 Oxygen enrichment and natural gas

As in Figure 5.5 for coal injection, the working area for natural gas–oxygen enrichment is shown in Figure 5.6. When applying natural gas, the point where the minimum T_{flame} (for example 2050 °C) and minimum oxygen temperature cross each other, will be the point with highest possible productivity and lowest coke rate. The exact position of the crossing point depends on local conditions (like moisture input with burden and coke) and on the minimum possible flame

temperature. Fortunately, because of the hydrogen effect described further on, the minimum T_{flame} with gas injection is lower than with coal injection; this is shown with the line indicating a flame temperature of 2000 °C in Figure 5.6, resulting in a potentially higher gas injection rate of about 20 kg/tHM.

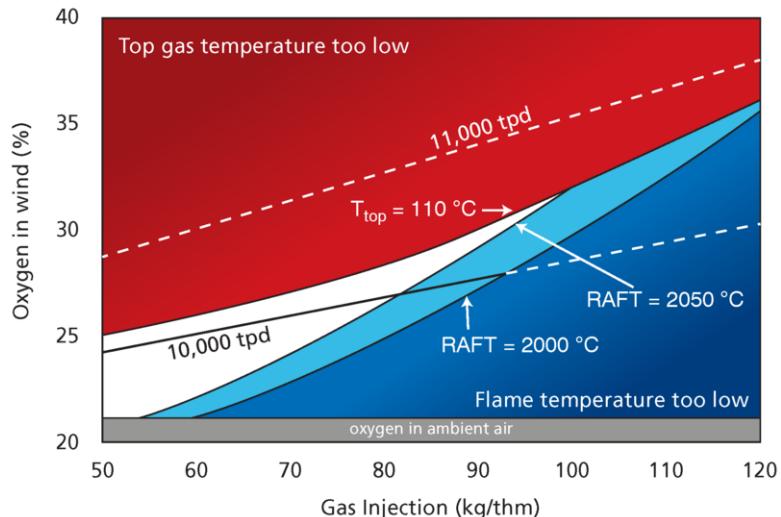


Figure 5.6 Working area for a furnace on natural gas injection.

5.3.2.2 Thermal control

The usability of coal injection for thermal control is based on the fact that as soon as coal injection rate is increased, the production slows down, which compensates the effect of the lower flame temperature. Natural gas works much slower in the furnace: the immediate effect of an increase in natural gas injection is that the flame temperature drops, while the production rate decreases slower than with PCI. Natural gas works more indirectly by changing the reduction reactions. Additional coal injection works immediately and is visible in the casthouse after 2–3 hours. For additional gas injection, the effect of for example 10 kg/tHM is smaller and the reaction is visible after 4–6 hours.

5.3.3 Flame temperature, hydrogen and total energy approach

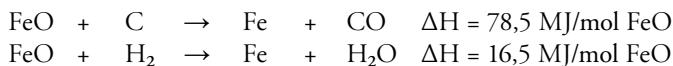
The adiabatic flame temperature reflects the heat that is used to melt and heat up the hot metal and slag, as well as the heat used in the final reduction of oxides. As shown before, the flame temperature effect of natural gas is much larger than that of coal, oil or tar. However, the decrease in direct reduction reaction means that less heat is required for the final melting and heating and furnaces operating with natural gas can operate on lower flame temperatures. Furnaces operating with coal injection are typically operating with a flame temperature between 2050 and 2200 °C, while furnaces with natural gas can operate at flame temperatures well below 1900 °C.

The effect is caused by the relatively large percentage of hydrogen in natural gas. It is discussed here, but equally well applicable for hydrogen coming from other sources (coal, oil and moisture in hot blast).

The hydrogen is generated by the incomplete combustion of natural gas and partly replaces the carbon monoxide. Hydrogen is very efficient for reduction in the lower part of the furnace. The presence of hydrogen in the bosh gas lowers the direct reduction and gas reduction in the shaft increases. So a higher hydrogen in the bosh gas lowers the heat requirement of the lower part of the furnace. This approach has been quantified by Charles River associates (Agarwall et al, 1991). The basis of their approach is that the hydrogen effect is recalculated to a lower heat requirement for the lower furnace.

The total energy is the thermal energy plus the “chemical” energy. The thermal energy is the amount of heat generated in the raceway above a reference temperature of 1482 °C. So it is the gas quantity ($\text{m}^3 \text{ STP/tHM}$) times ($T_{\text{flame}} - 1482$) times the heat capacity of the gas ($\text{kJ/m}^3 \text{ STP}$).

The chemical energy is the equivalent energy of the bosh gas hydrogen and the hydrogen utilization, taking into account that additional gas reduction by hydrogen gives an energy advantage of 62 MJ/mol FeO. This is based on the following reaction equations:



The difference is 62 MJ/mol FeO. The total energy is the sum of thermal and chemical energy.

Analysis of an operating blast furnace at varying gas rates has shown that the total energy required in the lower furnace is constant (Lingiardi et al, 2001). This means that at higher gas injection rates and thus higher hydrogen input per ton, the chemical energy increases allowing the thermal energy (T_{flame}) to decrease.

But hydrogen has additional effects on the process. Firstly, since hydrogen is a very light molecule, the density of the blast furnace gas decreases with increasing hydrogen content and more wind can be blown for the same Δp . This can be quantified as follows. The Ergun equation is $\Delta p/L = K\rho V^2$, where $\Delta p/L$ is the pressure difference per unit height (and is related to burden properties, permeability index K), the gas density (ρ) and the gas speed (V). For 5 % more hydrogen in the bosh gas, the density decreases with 4.3 % and the blast volume can increase with 2.3 % for the same Δp .

Secondly, since part of the reduction process is done with hydrogen, the requirement for carbon monoxide decreases. The amount of carbon – the carbon footprint – required per tonne hot metal decreases. The difference is that

the carbon requirement for a furnace on natural gas is 10–15 kg lower than for a furnace with coal injection, when we neglect any coke effects.

Thirdly, standard deviation of hot metal silicon and sulfur decrease with increasing injection levels. This is ascribed to the observation, that the variability in direct reduction is largely responsible for the variability of the thermal state of the blast furnace.

It should be noted that even though the higher hydrogen produced by high natural gas injection (over 100 kg/tHM) improves the process, as described above, the top gas hydrogen can be between 10 and 14 %, which is highly flammable. Therefore, the furnace top and charging equipment must be maintained to be free from leakage or a top fire could be the result when mixed with oxygen in outside air.

5.4 Coal-gas co-injection

Depending on the local situation, operators choose to co-inject coal and gas simultaneously. Since in the raceway, the gas and coal are (nearly) completely converted to CO and hydrogen, the effects of both injectants are additive. Coal still generates the most heat, while gas generates hydrogen, which is efficient in the lower part of the furnace.

The figure below shows the mass and heat balance results for a furnace, operating with natural gas, coal and coal with an additional 50 kg/tHM natural gas. The presence of some natural gas allows the operator to reach the same low coke rates as with only coal injection.

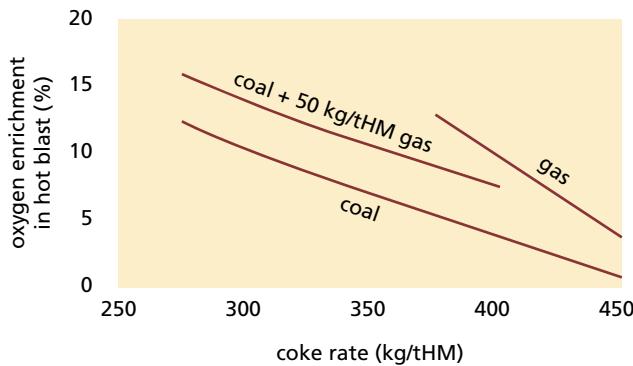


Figure 5.7 Typical example of coal-gas co-injection

But there are more differences. An operator will use the lowest coke rate possible under the given conditions. Applying coal-gas co-injection means that coal and gas are exchanged. If coal injection is partially replaced by gas injection, the effects are as follows (typical example in Table 5.7):

- Sulfur input in the furnace becomes lower.
- Less coal ash, allowing the operator to work with a lower basicity in the cohesive zone, which is favourable for burden descent. It also results in a lower slag rate and the furnace can be operated at a lower basicity for the same hot metal sulfur.
- More gas reduction by hydrogen, which results in a lower carbon footprint and an improved hot metal quality variation.
- Because the hydrogen molecule is less dense, the operator can put more wind on the furnace for the same Δp .
- Higher oxygen requirement per ton hot metal.

gas injection	kg/tHM	+50
coal injection	kg/tHM	-45
oxygen enrichment	% in hot blast	+3.5
oxygen enrichment	m ³ STP/tHM	+34
slag volume	kg/tHM	-6.4*
sulphur input	% of total	-8
CO ₂ footprint	kg/tHM	-27

Table 5.7 Typical effects of 50 kg/tHM gas injection in a blast furnace operated with coal injection

* has additional fuel rate effect of 0.4 kg/tHM coal

Another positive feature of having both natural gas and coal injection available is that there is a back-up injectant in case of failure of one of the injection systems. However, the economic optimum is depending on local conditions:

- prices
- burden quality
- operator skills
- the minimum coke rate given the local coke quality

As stated earlier, furnaces operating with coal injection are typically operating with a flame temperature between 2050 and 2200 °C, while furnaces with natural gas can operate at flame temperatures well below 1900 °C (Lingiardi et al, 2001). The lower flame temperature for natural gas injection is caused by the relatively large percentage of hydrogen in natural gas.

For thermal control in day-to-day operations and for stopping a cooling trend of the furnace, coal injection can still be used when using co-injection. However, at low flame temperatures, the simultaneous decrease of gas injection (with for example 20 kg/tHM) and increase of coal injection with the same amount has to be preferred since it increases the heat available per tonne hot metal.

5.5 Complete combustion of injectants

Gasification of coal and natural gas are a concern for many operators. There are many methods applied to improve the gasification of injectants. For example, the use of pre-mixed oxygen and natural gas as mentioned earlier.

Coal is not completely gasified in the raceway. Some material leaves the raceway as char. The observation of “unburned char” at the top has led to numerous investigations with respect to coal gasification. With respect to the presence of “unburned coal” in the top gas, we have the following observations from operational experience:

- Occasionally, unburned material can be found, but it coincides with poor burden descent. Our explanation is that in these situations, channelling of the gas flow from tuyere to top takes place.
- The presence of unburned char in the top gas coincides with high top gas temperatures. High top gas temperatures always cause high dust emissions from blast furnaces, as we have found from correlation calculations for furnaces measuring their dust load (kg/tHM).
- High coal injection requires a central coke chimney and results in high top gas temperatures in the center of the furnace. We like to suggest that the unburned char found at the top is Boudouard carbon formed when this gas cools down.

More specifically, hot gas coming to the top through the center of the furnace contains high levels of CO at a high temperature. This gas mixes with the gas that went through the iron-bearing materials and is much colder. Mixing of two gases causes rapid cooling of the center-originating gas, leading to the formation of Boudouard carbon and CO_2 . The mechanism is illustrated in Figure 5.8. The formation of the Boudouard carbon can possibly be prevented by spraying water in the hot, central top gas from an above burden probe.

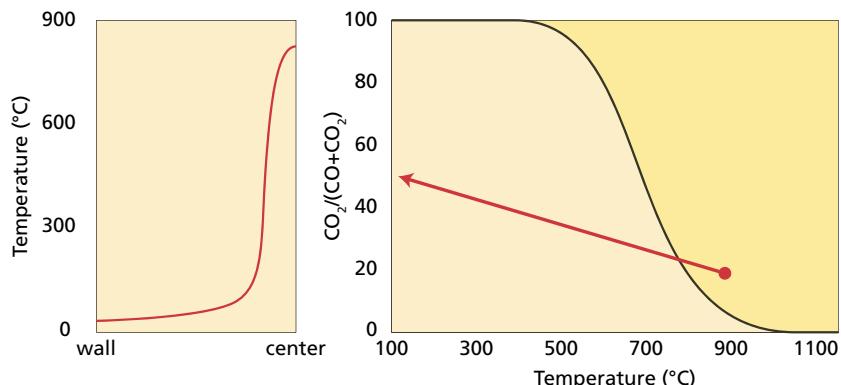


Figure 5.8 Formation of Boudouard carbon from hot (700–900 °C) top gas with high CO content (ETA CO below 20 %). The central top gas cools down to top temperature in approximately 1 second, passing the equilibrium line $2 \text{ CO} \rightarrow \text{CO}_2 + \text{C}$, as indicated with the red arrow.

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VI 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 chemistry has to be designed for optimum properties with respect to liquidus temperature, fluidity, desulfurizing 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 is typically 0.4 to 0.5 %. Low sulfur (under 0.03 %) and defined phosphorus 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 IX.

The burden has to fulfill requirements with respect to the highest allowable input of:

- Phosphorus, since phosphorus leaves the furnace with the iron.
- Alkali input. Especially potassium, which can attack the refractory and affect the process. Typically a limit of 1 to 3 kg/tHM is used.
- Zinc, which can condense in the furnace and, similar to alkali, lead to a zinc cycle.

Typically, limits for zinc input are 100–150 g/tHM. With high central gas temperatures, zinc and alkali are partly removed as vapor with the top gas.

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_2O_5 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 gas scrubber systems. The calculation is presented in Table 6.1.

Chemical analysis (%)											
	Ash	Moisture	Loss	C	Fe	SiO_2	CaO	MgO	Al_2O_3		
Coke		5.0	2	87.5	0.5	5.0			3.0		
Coal		1.0	0	80.0	0.2	3.0			1.5		
Sinter		1.0	1		58.0	4.0	8.3	1.4	0.6		
Pellets		1.0	1		64.8	3.5		1.3	0.8		
Lump		3.0	1		61.0	4.0			1.0		
Burden											
	Weight, dry kg/tHM	Weight, after losses kg/tHM			Input kg/tHM						
					Fe	SiO_2	CaO	MgO	Al_2O_3		
Coke	300	294			1	15	0	0	9		
Coal	200	200			0	6	0	0	3		
Sinter	1000	990			574	40	82	14	6		
Pellets	500	495			321	17	0	6	4		
Lump	80	79			48	3	0	0	1		
Total Burden	1580				945	81	82	20	23		
HM silicon	0.46 % uses					-10					
In 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	B2 – CaO/SiO_2				1.16						
	B3 – $(\text{CaO}+\text{MgO})/\text{SiO}_2$				1.45						
	B4 – $(\text{CaO}+\text{MgO})/(\text{SiO}_2+\text{Al}_2\text{O}_3)$				1.10						

*) $(\text{SiO}_2+\text{CaO}+\text{MgO}+\text{Al}_2\text{O}_3)/0.96$

Table 6.1

Example of Burden Calculation

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 tHM/d furnace is used. The stepwise approach is indicated in Table 6.2.

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 %	

Table 6.2 Stepwise approach for a simplified mass balance

The approach is as follows:

- Step 1 Nitrogen balance: from the nitrogen balance, the total top gas volume is estimated.
- Step 2 Hydrogen balance: the top gas hydrogen can be estimated from the hydrogen input and a hydrogen utilization of 40 %. In practice, hydrogen utilizations 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 per tonne hot metal is known.
- 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 kg of carbon is a number of carbon atoms defined as a kmol.
- Every mole of an element or compound has a certain weight defined in the periodic table of the elements.
- 1 kmol 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 6.3. The present balance is used for educational purposes where figures and compositions are rounded numbers. Effects of moisture in pulverized coal and the argon in the blast are neglected.

	Molecular weights		Molecular weights	
N ₂	28	kg/kmol	CO	28 kg/kmol
O ₂	32	kg/kmol	CO ₂	44 kg/kmol
H ₂	2	kg/kmol	H ₂ O	18 kg/mol
C	12	kg/kmol		
Fe	55.85	kg/kmol		
O	16	kg/kmol		
Chemical composition TAN				
	C %	H %	O %	N %
Coal	80	5.00	8	1.30
Coke	87	0.20	0	1.30
Burden	O/Fe	1/480	Atomic Basis	
	Fe %	C %	Si %	
Hot metal	94.5	4.50	0.46	
1 kmol corresponds with 22.4 m ³ STP				
Hydrogen utilization 40 %				

Table 6.3 Properties of materials used for mass balance calculations

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 6.4 and the top gas volume is calculated in Table 6.5.

Inputs			Measured results (top gas)	
wind	m ³ STP/min	6500	CO content	25 %
oxygen in wind	%	26	CO ₂ content	22.5 %
production	tHM/d	10,000	H ₂ content	5.2 %
production	tHM/min	6.9	N ₂ content	47.3 %
coal rate	kg/tHM	200		
coke rate	kt/tHM	300		

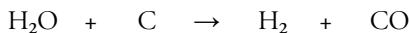
Table 6.4 Mass balance input

	m ³ STP/tHM	kmol/tHM
Blast volume	936	
Nitrogen balance from hot blast	693	30.9
Nitrogen balance from coal/coke	6	0.3
Total nitrogen	698	31.2
N ₂ in top gas	47.3 %	
Top gas volume	1476	

Table 6.5 The nitrogen balance and top gas volume

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 is converted to H₂ in the furnace. In the furnace, the H₂ is removing oxygen from the burden and forms H₂O again; part of the hydrogen is utilized again. Since the top gas volume is known, as well as the hydrogen input, the top gas hydrogen can be calculated if a utilization of 40 % is assumed. There are ways to check the hydrogen utilization, but this is beyond the scope of the present exercise. Table 6.6 shows the input and calculates the top gas hydrogen.

	m ³ STP/tHM	kmol/tHM
Hydrogen input (moisture in wind)	12	0.5
Hydrogen input (coal)	112	5.0
Hydrogen input (coke)	7	0.3
Total Input	130	5.8
Utilization (40 %)	-52	-2.3
In top gas	78	3.5
H ₂ in top gas	5.3 %	
Oxygen eliminated with hydrogen		2,3 kat O/tHM

Table 6.6 The hydrogen balance

6.4.3 The carbon balance

Hot metal contains 945 kg Fe per tonne with the balance consisting of carbon (45 kg), silicon, manganese, sulfur, phosphorus, 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 16.9 kmol Fe (945/55.85).

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 6.7 shows the results. The carbon via the top gas is also given in katom C per tonne hot metal.

	kg/tHM	katom/tHM
Carbon from coke	261	21.8
Carbon from coal	160	13.3
Total input	421	35.1
Carbon in hot metal	-45	-3.8
Carbon in top gas	376	31.3

Table 6.7 The carbon balance

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 1.48, which means that for every atom of Fe there is 1.48 atom O. On a weight basis it means, that for every tonne hot metal, which contains 945 kg Fe atoms there is 401 kg O atoms. All this oxygen leaves the furnace with the top gas. The balance is given in Table 6.8.

	m ³ STP/tHM	kg/tHM	katom/tHM
Oxygen in wind	243.4	347.7	21.7
Oxygen from coal		16.0	1.0
Oxygen from moisture		8.3	0.5
Input at tuyeres		372.0	23.2
From burden		400.7	25.0
Total input		772.7	48.3

Table 6.8 The oxygen balance

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 H_2O .
- Bound to carbon as CO.
- Bound to carbon as CO_2 .

From the combination of the carbon balance and the oxygen balance, we can now derive the top gas utilization, as shown in Table 6.9.

	kat/tHM	calculated	measured
Oxygen via top gas	48.3		
Oxygen eliminated as water	-2.3		
Oxygen as CO and CO ₂	46.0		
Carbon via top gas	31.3		
As CO ₂	14.6	22.2 %	22.5 %
As CO	16.7	25.3 %	25.0 %
H ₂		5.3 %	5.2 %
Gas utilization	46.7 %	46.7 %	47.4 %

Table 6.9 Calculation of top gas utilization

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 (Rist diagram, Annex 8, see also Rist and Meysson, 1966). The models are useful for analysis, especially questions like “are we producing efficiently?” and for predictions like “what if PCI is increased?”, “what if hot blast temperature is increased?” and so on. On the other hand, analysis of practical data allows us to monitor the process in real time, since the gas passes the furnace in seconds. The top gas composition gives an insight into the real condition in the furnace much more quickly, as shown in the section below.

6.5 Estimation of direct reduction

The oxygen, which is not removed by gas in the upper part of the furnace, is removed by direct reduction (details in section 8.3.3). The top gas CO and CO₂ are coming from coke that is gasified. Most of it arises at the raceway by reaction of oxygen (and oxygen containing material like moisture and coal). Some additional CO is generated by reaction inside the furnace. This is of special importance, since the direct reduction reactions are the major “heat sink” in the whole process (these reactions consume substantial amounts of energy). When we are able to follow the direct reduction amount from the gas analysis, we have a powerful tool for timely reaction on a cooling trend in the process (Figure 6.1).

The carbon used for direct reduction is the difference between the carbon that leaves the top of the furnace and the carbon that is gasified in the raceway. In our example, direct reduction can be estimated as follows (Table 6.10 on the next page). In our example, the direct reduction gasifies 97 kg carbon, corresponding with 111 kg of coke.

Oxygen input in the raceway	372	kg O/tHM	Table 6.8
Which gasifies	279	kg C/tHM	
Top gas volume	1476	m ³ STP/tHM	Table 6.4
As CO and CO ₂ are eliminated	701	m ³ STP/tHM	CO+CO ₂ % = 47.5 %
this corresponds with	376	kg C/tHM	
Thus direct reduction gasifies	97	kg C/tHM	
Corresponding with	111	kg coke/tHM	

Table 6.10 Calculation of amount of carbon and coke used for direct reduction

The variability of direct reduction is often the most important factor affecting process stability. The trend in direct reduction can be monitored in real time in the control room. It gives the operator an early warning signal for a cooling trend of the production process, because the direct reduction reaction consumes extra coke. In case of a cooling trend, the operator has the burden descent rate as a check. Increased direct reduction and increased burden descent rate are far quicker indications than hot metal temperature or hot metal silicon measured in the casthouse. Such a cooling trend can be stopped by additional reductant injection.

An example of increased direct reduction and burden descent rate is given in Figure 6.1.

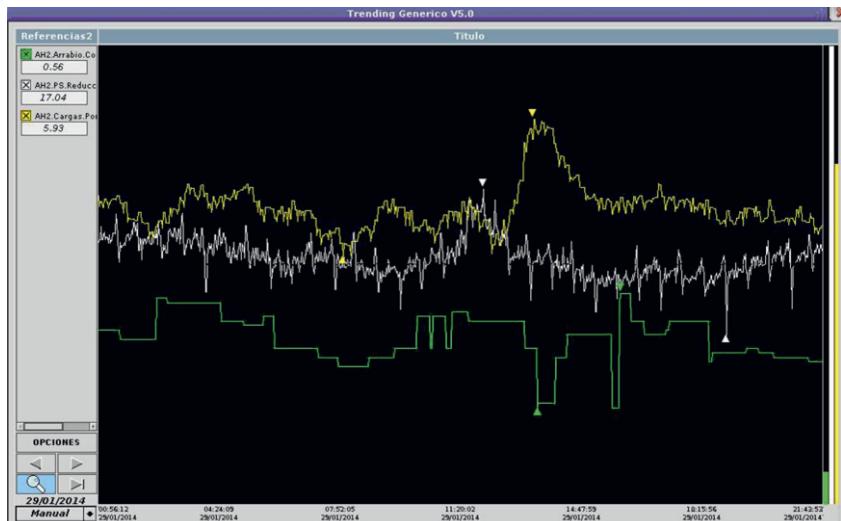


Figure 6.1 Example of increased direct reduction, followed by increased charging rate and decreasing hot metal silicon
(yellow=charges per hour; white=direct reduction; green=hot metal Si)

VII The Process: Burden Descent and Gas Flow Control

7.1 Burden descent: where is voidage created?

The burden descends through the blast furnace from top to bottom. Figure 7.1 shows the burden descent. It is indicated with mechanical stock rods, which are resting on the burden surface and descending with the burden between charging. The burden surface descends at a speed of 8 to 15 cm/min.

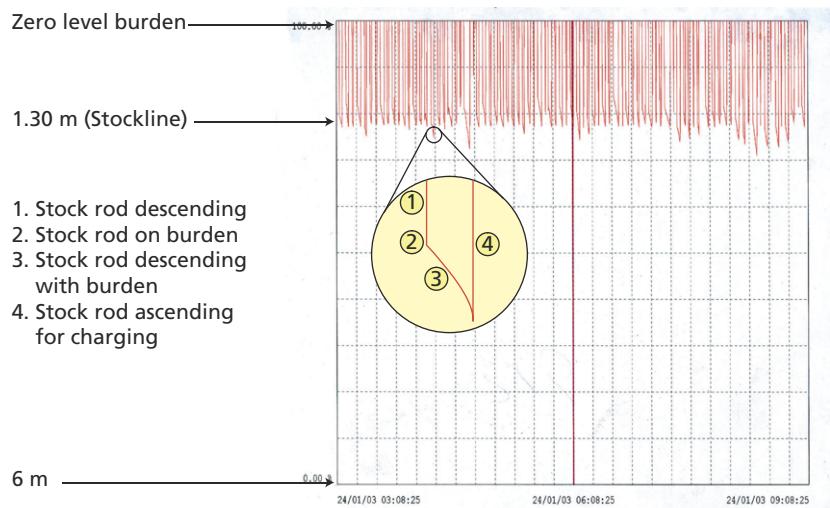


Figure 7.1 Stable burden descent

In order for the burden to descend, voidage has to be created somewhere in the furnace. Where this voidage is created is illustrated in Figure 7.2.

- 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 carburization 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 with a high PCI rate, only about 25 % of the voidage is created at the tuyeres.

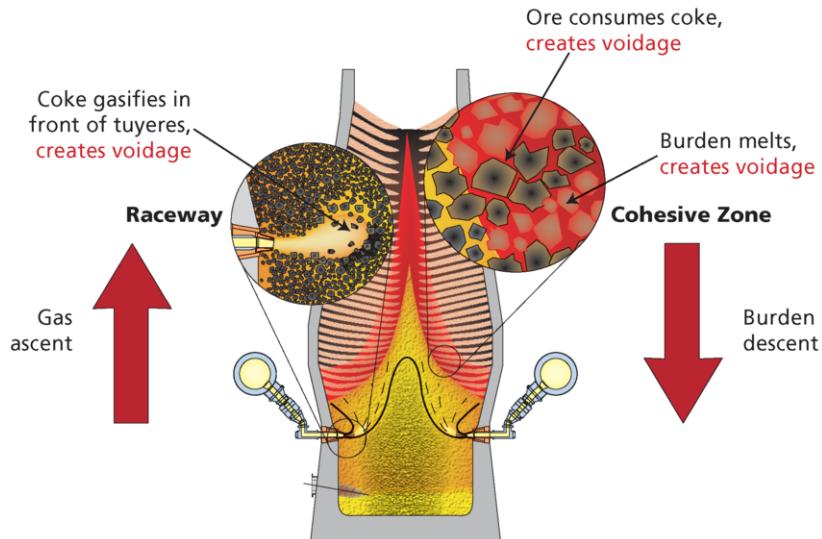


Figure 7.2 Creation of voidage in the blast furnace

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 a high ore concentration at any ring in the circumference, especially the wall area, has to be avoided.

Sometimes, the burden descent of a blast furnace is erratic. What 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 7.3). Also the phenomena 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 must be found in the configuration of the cohesive 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 (Section 11.2.2).

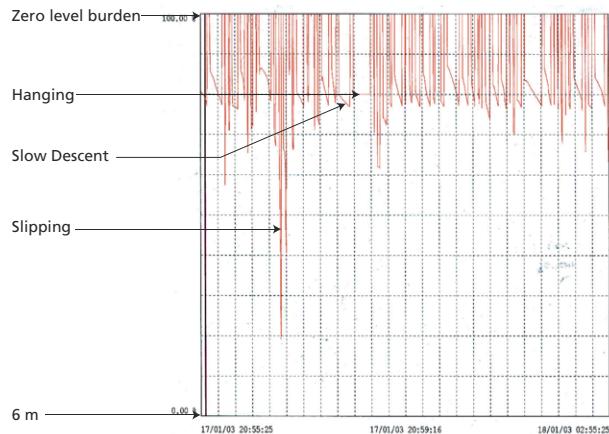


Figure 7.3 Irregular burden descent

7.2 Burden descent: system of vertical forces

The burden descends because the downward forces of the burden exceed counteracting upward forces. This idea has been especially developed by Mr. Cor van der Vliet, co-author of the first and second editions of this book. 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.

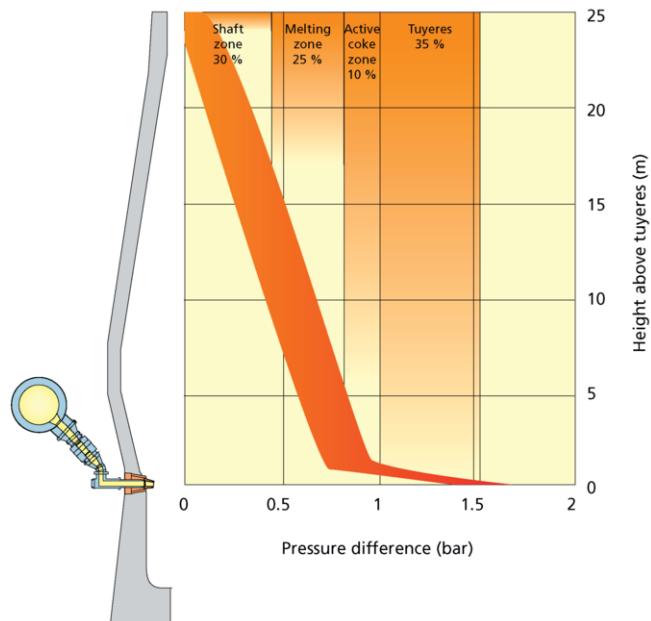


Figure 7.4 Pressure difference over burden

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. 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 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. In case of irregular burden descent, these friction forces can become rather large.

The coke submerged in hot metal also exerts a high upward force on the burden due to buoyancy forces (Figure 7.5) as long as the coke is free to move upwards and does not adhere to the bottom.

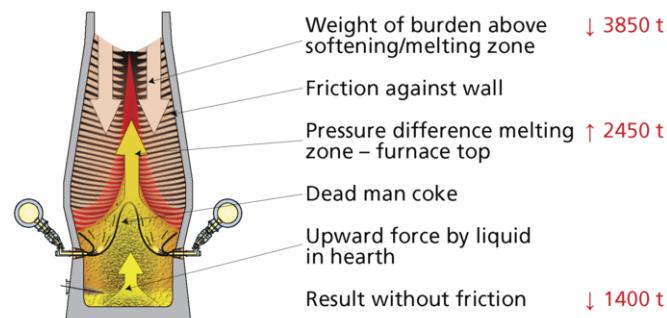


Figure 7.5 System of vertical forces in the blast furnace

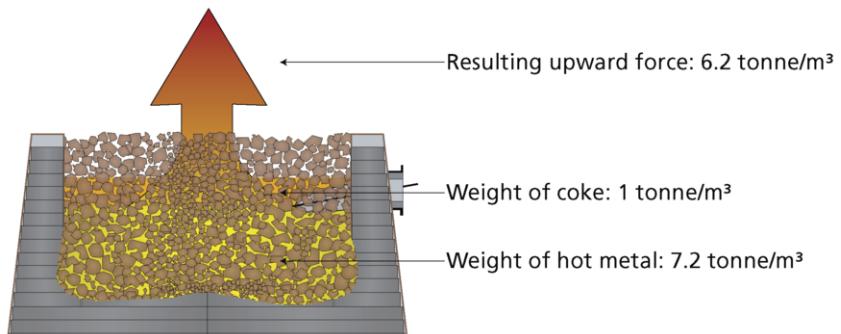


Figure 7.6 Upward force from hearth liquids

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 2

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 dripping distance to the hearth 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. A high liquid level in the hearth creates greater buoyancy forces.

7.3 Gas flow in the blast furnace

The gas generated at the tuyeres and the melting zone has a short residence time of 6 to 12 seconds in the blast furnace (section 2.3). During this time, the gas cools down from the flame temperature to the top gas temperature, from 2000 to 2200 °C down to 100 to 150 °C, while simultaneously removing oxygen from the burden. The vertical distance between tuyeres and stockline is approximately 22 meters. 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 Beaufort 2 to 3. During these 6 to 12 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 layers. It is important to note, as indicated in Figure 7.7, 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.

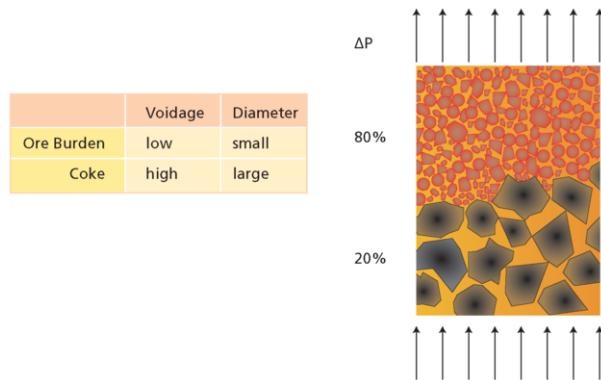


Figure 7.7 Pressure drop through coke and ore

As soon as the ore burden starts to soften and melt, at about 1100 °C, the burden layer collapses and becomes (nearly) impermeable for gas. If this happens in the center of the furnace, the central gas flow is blocked.

7.3.1 Observation of heat fluxes through the wall

Figure 7.8 shows the temperature at the hot face of the furnace wall. It has been observed in many furnaces, that the temperature suddenly rises in minutes and decreases over the next hour(s). This is often attributed to the loss of scabs (deposits) 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 gases flow preferentially. This phenomenon is also known as “gas channeling”. This can be observed from pressure tap measurements (see Figure 7.23). 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.

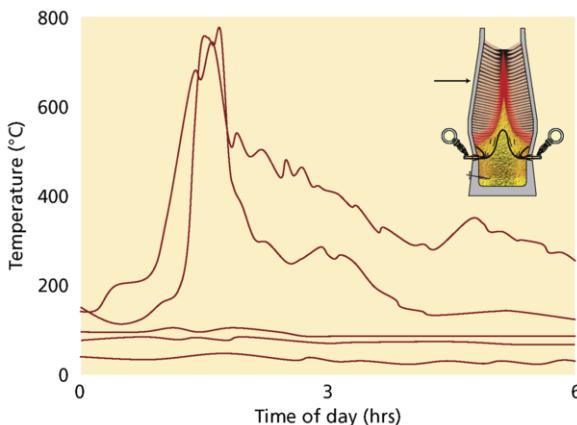


Figure 7.8 Temperatures at hot face

Why does the gas flow along the wall? Gas takes the route with the lowest resistance and, therefore, highest permeability. The resistance for gas flow in a full blast furnace is located in the ore layers, since its initial permeability is 4 to 5 times less than that of coke layers. There are two areas that have the highest permeability: the center 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 center 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 cohesive 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 temperatures are examples of energy losses. The top gas contains CO and H₂, which have a high calorific value. 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 center. In this case, the center of the furnace contains only coke and coarse burden materials and is the most permeable area in the furnace. The cohesive zone takes on an “inverted V shape”. In a “wall-working” furnace, the gas flow through the center 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 cohesive zone takes the form of “W shape”. Figure 7.9 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.

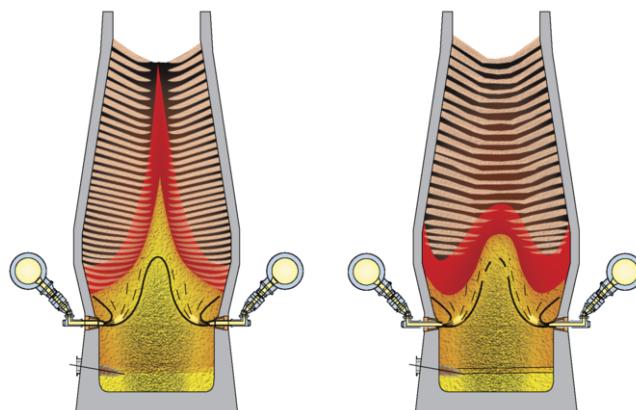


Figure 7.9 Two types of melting zone, central working (left) and wall-working (right)

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 7.10 on the next page, the ore to coke ratio over the radius is shown for a central working furnace. In the figure, the center of the furnace only contains coke. Therefore, in the center of the furnace, no melting zone can be formed and the gas is distributed via the coke slits from the center 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 also as a type of pressure relief valve: it functions to stabilize the blast pressure.

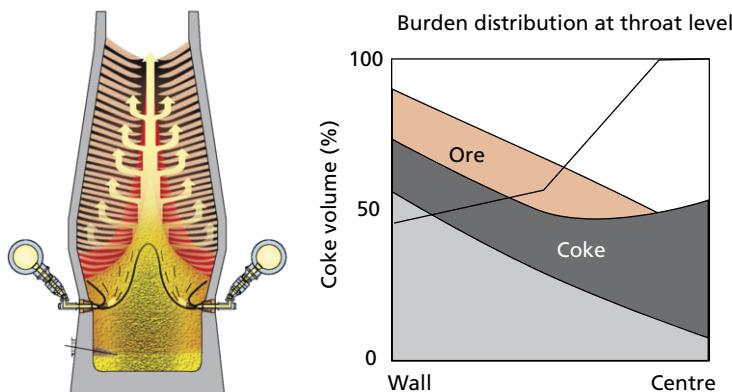


Figure 7.10 Central working furnace

It depends on the type of burden distribution equipment how the coke can be brought to the center. With a bell-less top, the most inward positions or lowest angles of the rotating chute can be used. With a double bell system, the coke has to be brought to the centre by “coke push” (see further on) and by choosing the right ore layer thickness in order to prevent the flooding of the center with ore burden materials. In the central working furnace, there is a relatively small amount of hot gas at the furnace wall and 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 may not be 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 cohesive 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 can be made at the wall (see section 7.6.1).
- 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 center 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 within reasonable limits.

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 lengths for the furnace above the tuyeres. However, the process is very sensitive for deviations in burden materials, especially the size distribution.

7.3.4 Wall-working furnace

In Figure 7.11, the wall-working furnace is presented. Melting ore burden blocks the center of the furnace and the gas flow is directed towards the wall area.

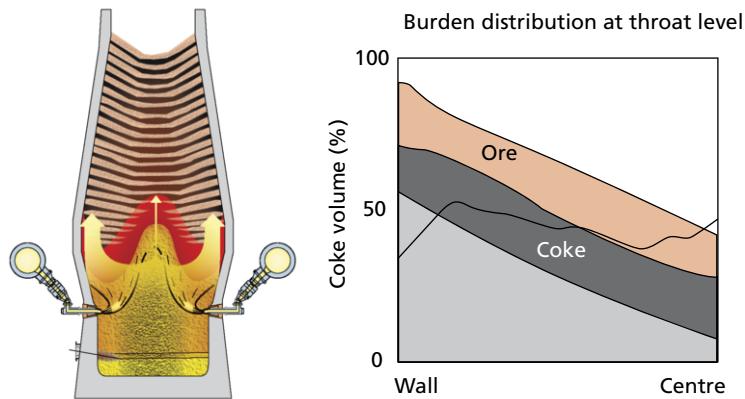


Figure 7.11 Wall-working furnace

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 the lower and middle shaft. The melting zone gets a "W" shape or even the shape of a horizontal 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 rather efficient. However, due to the high heat losses, the wear of the refractory or damage to cooling staves or plates 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. As a consequence, the fuel rate is high. 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 or “coke slits” in the cohesive zone and into the granular coke and ore layers, as shown in Figure 7.12.

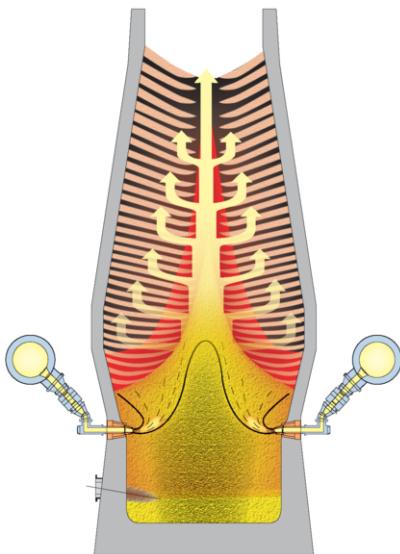


Figure 7.12 Schematic presentation of gas distribution through coke layers

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 mentioned in section 3.3.1, 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 III). In practical operations, the permeability of ore burden material is determined by the amount of fines (percentage under 5 mm). Fines are very unevenly distributed over the radius of the furnace, as is indicated by the typical example shown in Figure 7.13. 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 voids 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 (due to swelling during the chemical reaction 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.

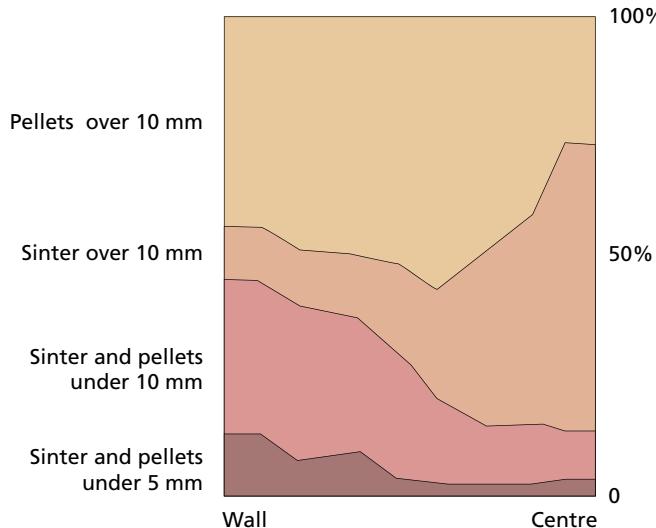


Figure 7.13 *Distribution of fines over the radius, double bell simulation*
(after Geerdes et al, 1991)

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 melt down. Sometimes, unmolten ore burden materials are visible as scabs through the peepsites of the tuyeres. The unmolten material 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 Fluidization and channelling

The average gas speed above the burden is rather low, as shown in Chapter II. However, in a central working furnace the gas speed might locally reach 10 m/s or more especially in the center of the furnace. This is well above theoretical gas velocities, at which fluidization can be observed (Figure 7.14). Coke fluidizes much more easily than ore burden because of its lower density. It is believed, that the ore burden secures the coke particles in the centre. Nevertheless, if local gas speeds become too high, fluidization or lifting of coke may occur. Fluidization 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 fluidizing coke “dive” into the coke layers.

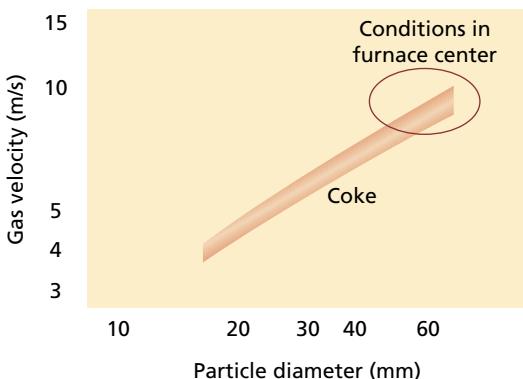


Figure 7.14 Gas velocities for fluidization of ore burden and coke. Shaded areas indicate critical empty tube gas velocities for fluidization at 800 °C and 300 °C and 1 atmosphere pressure (after Biswas, 1981)

If the fluidization 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 gas channels without coke or ore burden. 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 high temperature and low utilization, since the gas was not in good contact with the burden. The limit of channelling is where the furnace slips.

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 7.15 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 roll to the centre.

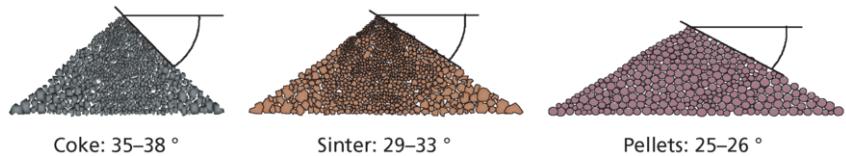


Figure 7.15 Segregation and angles of repose

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 7.15. The result is fine material at the center and coarse material at the foot of the conical pile.

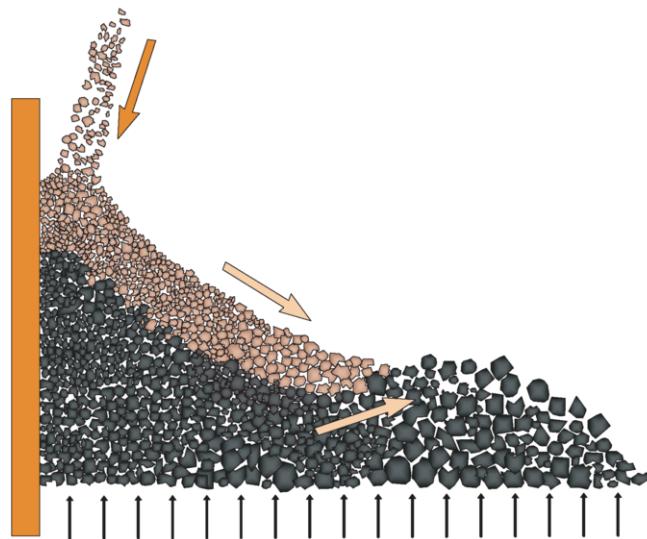


Figure 7.16 Coke push effect with gas flow

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 7.16.

7.5.2 The charging equipment

The type of charging mechanism used has a major impact on the distribution of fines. Figure 7.17 shows the bell-less top and double bell with movable armor systems.

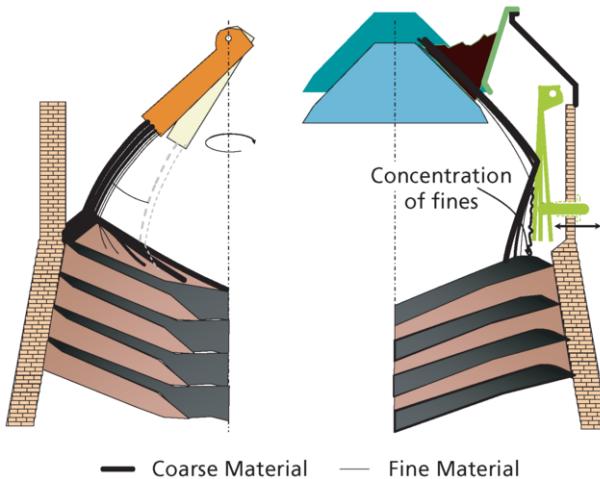


Figure 7.17 Bell-less top charging (left) and double bell charging (right): comparison of the segregation of fines on the stockline

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 center 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 armor, which give certain flexibility with respect to distribution of fines and the ore to coke ratio over the diameter, especially at the wall. 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" (Figure 7.18). 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. The smoother the burden descent, the less mixed layer formation.

7.5.4 Gas flow control

The optimized 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) center leaves the furnace with a low utilisation (high CO and low CO₂). This loss of “unused” gas should be minimized. If the central gas flow is too high, there is a too little 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 though the center 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 low in the furnace. The reduction reactions slow down. In this situation, the central gas flow is small and heat losses to the cooling system 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 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 or empirical 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 (C/O) ratio or ore to coke (O/C) over the radius. An example of a calculated burden distribution is shown in Figure 7.18 further on. Note the ore free center.
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 70 cm of ore in the throat consumes about 25 cm of coke for direct reduction (Figure 7.19). 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 center. The amount of ore reaching the center 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 46 cm. The diameter of the belly is 1.4 to 1.5 times bigger 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 thicknesses 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 armor furnaces than in furnaces equipped with a bell-less top. An example for a burden distribution control scheme is given in Table 7.1. If more central gas flow is required, then Coke 3 replaces schedule Coke 2. Replacing Coke 2 with Coke 1 reduces central gas flow.

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

Table 7.1 Bell-less top charging schedules with varying central gas flow

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 center. 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 7.2. The ore base is kept constant and coke base reduced. Experience has shown that relatively minor changes in burden distribution will be required for optimization of the central gas flow (i.e. coke distribution). The burden distribution adjustments can be applied as a second step if required.

	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

Table 7.2 Coke base change when PCI rate changes

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 7.18.

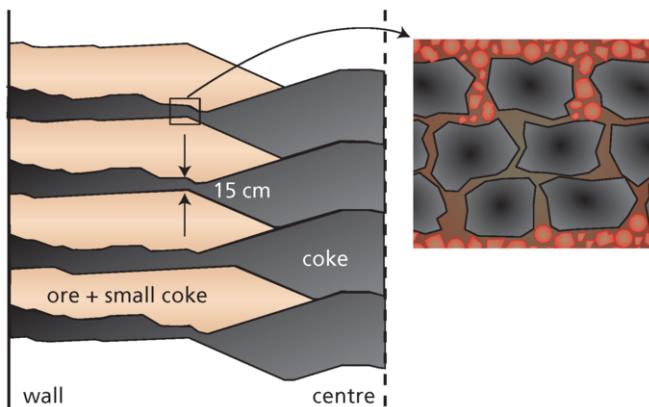


Figure 7.18 Example of burden distribution with an ore-free center and ore burden penetration in coke layer

7.6 Coke layer

7.6.1 Coke percentage at wall

For optimum gas distribution through the coke layers, it is desirable to have an ore-free chimney in the center of the furnace. This then requires a large amount of coke to be present in the center, but still some coke is required at the wall. This section deals with the question as to how much coke is required at the wall area.

A 70 cm thick ore layer at the wall contains about 1.5 tonnes ore burden in one square meter and therefore about 1 tonne hot metal. As shown in Section 8.2.1 dealing with direct reduction, the ore burden consumes coke, at a rate of about 120 kg coke per tonne. This amount of coke corresponds to a layer thickness of

24 cm, so the minimum coke amount at the wall is about 25 % of the volume, (see Figure 7.19), assuming that the coke is used only for direct reduction.

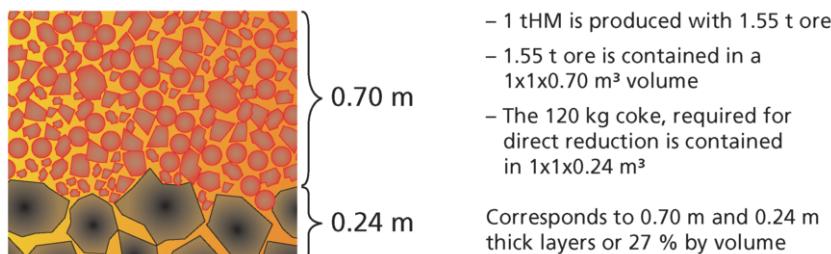


Figure 7.19 Coke required for direct reduction

If the amount of coke at the wall is less than the 25 % of the volume, then the ore layers will make contact between the sequential layers upon melting. This will form a column of unmolten ore that descends down the furnace to the tuyeres. This will lead to disturbed gas flow, but also there is a risk that this unmolten material will rest on the tuyere nose and cause the tuyere to tip. This can be observed through the peepsight where an oval opening of the tuyere is seen rather than a round one, and has been caused by the tuyere being tipped into the furnace by the heavy weight of the ore layer pushing down upon it.

The coke requirement at the wall can also be met using nut coke blended into the ore layer. In this case, the nut coke is preferentially available for direct reduction and will preserve the larger, metallurgical coke in the layer structure. Note also that the direct reduction percentage in the wall area can be higher than estimated above, so that even more coke is required at the wall.

7.6.2 Coke layer thickness

When reaching higher and higher coal injection levels, the question arises as to whether a minimum coke layer thickness exists, and what would it be? The gas ascending the furnace from the tuyeres to the top is distributed through the coke layers, so the coke layers must be present at all elevations of the furnace for this to continue. As the layers are made up of discrete coke particles, the theoretical minimum coke layer thickness translates into a number of coke particles. To produce a path for the gas, it is considered that the minimum number of coke particles that should be present in the height of one layer is three. The minimum thickness is therefore three times the mean size of coke in the belly of the blast furnace. Taking for example an average coke size of 50 mm, it would therefore be reasonable to expect that the minimum coke layer thickness in the belly is 15 cm. As the effective ratio of the surfaces of belly to throat is generally around two, the minimum coke layer thickness at the throat should have a minimum of about 30 cm. In operational practice of furnaces operating at high coal injection levels, the coke layer at the throat has reached values as low as 32 cm and in the belly 14 cm.

The minimum “metallurgical” coke rate (this is the coke rate excluding nut coke) as a monthly average for operating blast furnaces has reached levels of 210–220 kg/tHM in the best months and 230–240 kg/tHM as a longer term average for very well operated blast furnaces.

7.7 Ore layer thickness

What is the effect of ore layer thickness on the process? If thicker ore layers are charged, less ore layers are present in the operating furnace and less coke slits are available to distribute the gas. But, especially in conveyor belt-fed furnaces, the thicker the ore layer, the more charging capacity is available. For reduction and melting, two effects must be considered, those being the reduction in the granular zone of the furnace and the melting of the layers in the cohesive zone.

7.7.1 Reduction in granular zone

The reduction capacity of gas entering thicker ore layers will be depleted faster, and as a consequence, the reduction of ore burden in the granular zone will be poorer.

7.7.2 Softening and melting

As soon as an ore layer starts to soften and melt, it becomes impermeable for gas. This means that ore layers are heated up at the contact surface between the coke and ore layer. The thicker the ore layer, the longer it will take to melt down completely. Moreover, the melting of the ore layer slows down because there is more oxygen in the ore layer, because of lower rate of pre-reduction (see preceding section). So the thicker the ore layer, the more difficult the melting of the layer (Figure 7.20).

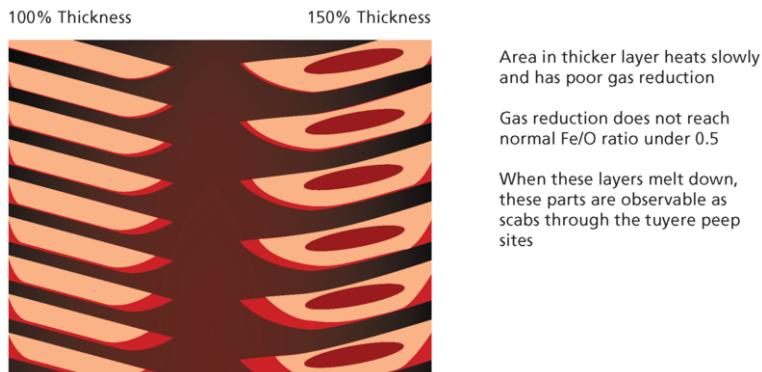


Figure 7.20 Melting of thin and thick ore layers compared

7.7.3 Optimizing ore and coke layer thickness

So, the blast furnace operator wants good permeable coke layers (i.e. thick layers) and good melting ore layers i.e. thin layers. As is often the case in blast furnace operations, the best operational results can only be reached with a compromise between these two factors. Generally speaking, from operational observation, the ore layers should not exceed 70–80 cm in the throat of a blast furnace and coke layers should not be smaller than 32 cm. The operational optimization depends on local situations.

Experience has shown that:

- Permeable ore layers can be maintained even when the layers have become quite thick, provided a permeable ore burden is used. For pellet burdens, this would require screening of the pellets, and for sinter it would have to be sized to a relatively large diameter (more than 5 mm).
- The minimum coke layer thickness experienced was 14 cm metallurgical coke in the belly.

Conveyor belt-fed furnaces tend to work with thicker ore layers. This is caused by the fact that in a conveyor-fed furnace, the charging capacity increases with increasing layer thickness. In skip-fed furnaces, the optimum charging capacity is reached with full skips of coke. In the past, the volume of coke was normally the determining factor, so furnaces tended to work with full skips of coke. At high coal injection rates the skip weight is normally the determining factor and thus furnaces now work with full skips of ore.

Another aspect of the optimization of the coke layer thickness has to do with the gas permeability of the coke layer. The coarser the coke is screened in the blast furnace stockhouse, the more permeable the layer is. There are, however, two drawbacks of the coarse (35 mm or more) screening of coke.

Consequence 1: The coarser the coke is screened, the more nut coke or small coke is produced. The nut coke is added to the ore burden layer, increasing the thickness of the ore burden layer and decreasing the size of the coke layer.

Consequence 2: The coarser the coke is screened at the stockhouse, the thicker the formation of a mixed layer at the coke–burden interface.

Optimization depends on local conditions, but high productivity has been reached with a coke screen size in the stockhouse of 25 mm and a nut coke quantity of 25 kg/tHM.

When using coal injection, the lowest metallurgical coke rates that have been reached by operating furnaces are around 225 kg/tHM metallurgical coke, to which the nut coke has to be added to reach the total coke rate.

7.7.4 “Ideal” burden distribution

The ideal burden distribution for high productivity and high PCI rates is—according to the authors—as follows (Figure 7.21):

- An ore-free center.
- Nearly horizontal layers of coke and ore burden.
- Some nut coke in the ore burden in the wall area.
- Coarse coke in the center.

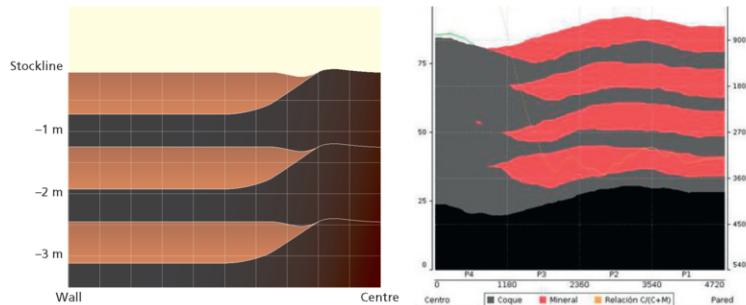


Figure 7.21 “Ideal” burden distribution (left) and operational practice as measured with profile measurements of close to ideal burden distribution (right)

Ore-free center

The ore-free center allows the gas to distribute itself through the coke layers from the center to the wall. We can consider the coke layers as layers with equal pressure. If the total internal pressure difference is 1.2 bar, the pressure difference over each of the 40 ore layers is about 0.03 bar. The ore free center typically has a diameter of 1.5 to 2 meters. The ore-free center can be made in a furnace with a bell-less top by discharging 10–15 % of the coke on a very inward chute position. In furnaces with a double bell top, formation of an ore free center is more difficult.

Nearly horizontal layers

Using nearly horizontal layers of coke and ore minimizes the effect of natural deviations of parameters important for the formation of the layers. E.g. wet pellets have a different angle of repose as compared with dry pellets. This does not affect burden distribution if nearly horizontal layers are used. Care should be taken, that there is no inversion of the profile, i.e. a pile in the centre of the furnace. This can be monitored with e.g. a profilemeter.

Nut coke

The gas in the wall area is cooled by the heat losses to the wall. Moreover, a relatively large percentage of fine ore burden material is located at the wall and reduction-disintegration is highest at the wall (because of slower heating and reduction). For these reactions, reduction and melting of the ore burden in the wall area is most difficult. Nut coke in the wall area helps to lower reduction gas and heat requirements in the wall area. The nut coke has a lower heat capacity than the ore burden. Moreover, when the ore burden in the wall area starts melting, the nut coke is immediately available for direct reduction. In doing so, it prevents the direct reduction attack on the metallurgical coke.

Coarse coke in the center

The coke charged in the center is the least attacked by the solution loss reaction and has the smallest chance to be burnt in front of the tuyeres. Therefore, it is thought that the coke charged in the center finally constitutes the coke in the hearth. Good permeability of the hearth helps to improve casting and prevents preferential flow of iron along the wall, thus increasing hearth campaign life.

7.8 Blast furnace instrumentation

An overview of blast furnace instrumentation as discussed in various parts of the text is given in Figure 7.22.

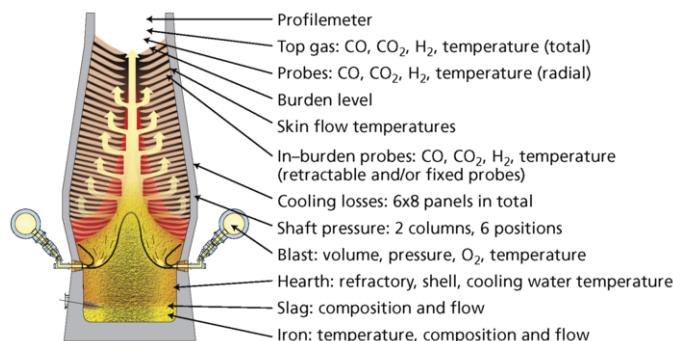


Figure 7.22 Overview of blast furnace instrumentation

7.9 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 sulfur. 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 injected through the tuyeres can achieve the latter.
- Stable process control. Burden descent (as measured by the stock rods, Figure 7.1, or pressure taps, Figure 7.23), 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 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.

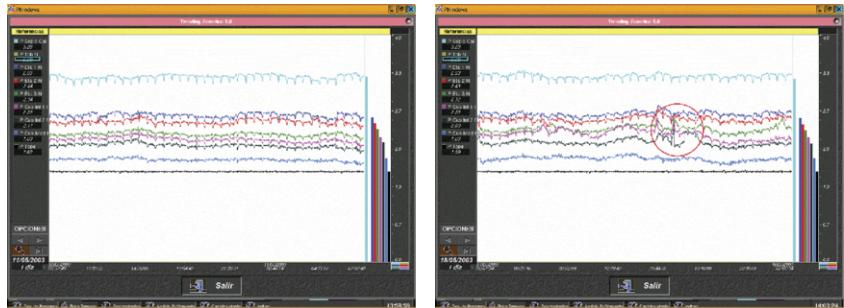


Figure 7.23 Pressure taps indicating the stability of the process, 24 hour graphs. The example shows stable (left) and unstable (right) operation, with short-circuiting of gas flow encircled in red.

– Gas flow control. The subject of gas flow control is discussed in more detail below. Measurements and data required for daily gas flow control are shown in Figure 7.24. 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 meters 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 the 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 utilization, 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 utilization 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 hot metal, 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.

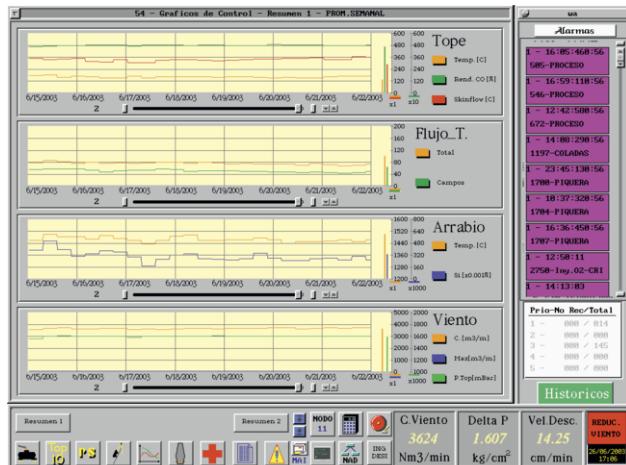


Figure 7.24 Presentation of process data in an operational furnace. The weekly graph gives an overview of the stability and development of the process. From the top downwards: Tope = CO utilization (%) and skin flow temperature (°C) and top temperature (°C); Flujo T = Total heat loss and sum of fields (GJ/hr); Arrabio = Hot metal temperature (°C) and silicon (%); Viento = Blast volume (m^3 STP/min) and top pressure (bar)

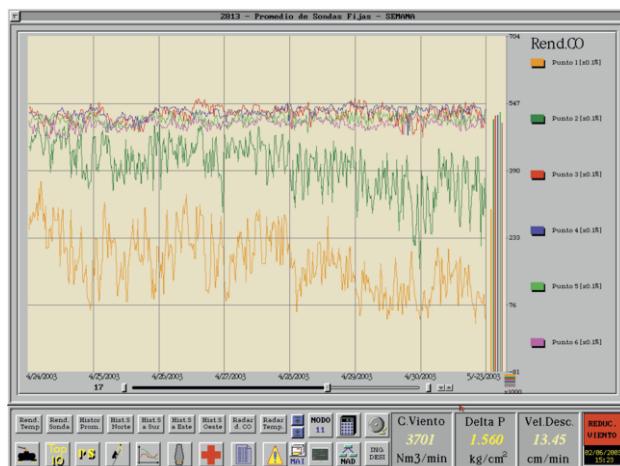


Figure 7.25 Example of gas flow control. The radial gas distribution is measured with above burden probes, expressed as CO utilization (7 day graph). The decreasing gas utilization in the center of the furnace (point 1 and 2, yellow and dark green) shows increased central working.

VIII *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 coke is consumed in a furnace by three mechanisms:

- the coke is gasified by oxygen entering the furnace through the tuyeres, mainly the wind oxygen but some oxygen comes from moisture in the wind and some oxygen is present in coal and natural gas injectants.
- the coke is consumed directly by the oxygen, that reaches the high temperature area of the furnace, especially the oxygen from partially reduced iron as well as the oxygen coming available from some other elements like reduction of SiO_2 (direct reduction).
- the coke is consumed because the carbon dissolves in the hot metal.

The productivity of a blast furnace is the amount of hot metal produced per time unit. It is often expressed as tonne hot metal produced per 24 hrs (without delays) either per m^3 working volume or per m^3 inner volume or per m^2 hearth surface.

The efficiency of a furnace is the amount of reductant (coal, coke, gas) that is used per tonne hot metal. The productivity of a furnace increases as the reductant rate decreases.

The limiting factors for the production rate can be

- the process: how much wind the furnace accepts.
- the casthouse: how much liquids can be drained.
- the equipment: charging capacity, capacity of turbo-blowers, availability of oxygen.

In the present chapter, the basics of the process behind blast furnace productivity, efficiency and the chemical reactions are discussed (see also Hartig et al, 2000).

8.1 Productivity

The productivity of blast furnaces varies enormously and in this section, the question of what productivity a furnace can reach, is addressed, while keeping the same burden materials and reductants.

The first point to consider is that the furnace is a gas reactor, so the more blast is blown into the furnace, the more it produces, since more coke is gasified. The furnace can be driven to a maximum Δp (blast pressure minus top pressure), but when going to a Δp above this maximum, the burden descent will deteriorate, the furnace becomes less efficient and productivity will decrease.

Therefore the first limit is maximum Δp , or, with constant top pressure, maximum blast pressure. The 'allowable' maximum Δp depends on the furnace and the burden and is a specific value for each furnace. Large, 14 meter hearth diameter furnaces can work up to levels of a maximum Δp of 1.95 bar or even higher. Note that this is an instantaneous maximum. Some furnaces have systems which automatically lower the blast volume when reaching the maximum Δp in order to prevent hanging, slips and process upsets. Smaller furnaces have maximum Δp of 1,5–1,8 bar.

Secondly, the more coke that is gasified at the tuyeres, the more iron is produced. So, the more oxygen that is present in the hot blast, the more coke that is gasified and productivity increases. The oxygen percentage in the hot blast can be increased by adding pure oxygen to the cold blast. However, there is a limit to the amount of oxygen enrichment that can be used, since an increase of the oxygen percentage in the hot blast leads to a lower top gas temperature (see section 5.2.4). As soon as the top temperature becomes too low (below 100 °C), the furnace becomes less efficient, since it takes more time to start the reduction reactions. Therefore, the productivity is highest if the top gas temperature is at its minimum. The minimum we recommend is to stay above the dewpoint, so that all moisture is driven off. This translates to a narrow band slightly above 100 °C.

The maximum productivity in a furnace (with a given burden and injectants) is reached when:

- Top pressure is at the maximum value.
- Blast volume is set so that the furnace is operated to the maximum Δp .
- Hot blast temperature is set at a the highest achievable value, since it lowers the fuel rate and thus the gas flow.
- Moisture is at ambient level.
- Fuel injection (coal, gas, oil) is at the maximum the furnace accepts.
- The top gas temperature is controlled to slightly above 100 °C with oxygen injection.
- The furnace operates circumferentially symmetric, meaning that all sections of the furnace representing one tuyere contribute equally to the process (see section 11.4).

The quality of the burden affects the productivity by means of the reductant rate: the lower the reductant rate per tonne hot metal, the more hot metal can be produced with the same amount of blast. On average, for every 3–3.5 kg/tHM decrease in reductant rate (coke, coal gas), productivity increases by 1 %.

8.2 Efficiency

The efficiency of the furnace is the amount of reductant used per tonne hot metal. The total amount of reductant per tonne decreases when the heat input in the furnace is larger (for example higher hot blast temperature) or the heat requirement is smaller. The major parts of the heat requirement are

- the heat used for the direct reduction reactions.
- the heat required for bringing the materials to casting temperature.
- the heat losses through the furnace wall.

These depend on the following.

8.2.1 Burden quality

The better the reducing gas is in contact with the ferrous burden, the better the gas reduction and the lower the direct reduction reactions and the reductant rate. Lump ore is less accessible for gas than sinter or pellets, so lump ore has a penalty, which varies from type to type but is typically an additional 50 kg coke for a tonne hot metal made from lump ore.

Another aspect is the amount of fines, so the amount of materials smaller than 5 mm that is charged or generated in the furnace. This is important because more fines block the gas flow and decrease permeability of an ore layer, especially when the fines are concentrated in certain radial areas, for example at the wall.

8.2.2 Burden composition and metallic iron

Since metallic iron does not have to be reduced, the charge of metallic iron units (scrap, DRI–Direct Reduced Iron, HBI–Hot Briquetted Iron) lowers the reductant rate and increases the furnace efficiency. Details are in section 3.6. The less slag that is made, the more efficient the furnace is, since there is less heat required to obtain the casting temperature.

8.2.3 Burden distribution and heat losses

In section 7.7.4, we have shown “ideal” burden distribution. The burden distribution is optimized when the ore layer between two coke layers is more or less uniform in thickness and has the lower heat requirement along the wall. Nevertheless, there still are two major areas of “efficiency loss”. First, the heat losses through the wall represent a considerable amount of reductant. Typically, heat losses are 150–300 MJ/tHM. 150 MJ/tHM corresponds with approximately 17 kg/tHM coke. The second “efficiency loss” is coming from the central gas flow. The central gas flow leaves the furnace with poor utilization

(below 20%) and high temperature (over 700 °C or even higher). So the central gas flow is poorly used for reduction. The benefits from central gas flow come from the decrease in heat losses, as well as from a better permeability of the coke slits, so that thinner coke layers can be used and thus higher (coal) injection rates.

8.2.4 Monitoring efficiency

The input into the furnace in coke, injectants and with hot blast (hot blast temperature, moisture, oxygen percentage) determines the efficiency of the furnace. However, it is often difficult to grasp the real, actual picture in the control room of a blast furnace, since different parameters are used for different reductants, for example tonne of coal or m³ of natural gas per hour and coke rate per tonne hot metal. Operators have to be aware how to “translate” these parameters to reductant rate (kg/tHM). Rules of thumb are in Annex 4. Online monitoring of the efficiency is possible by close attention to the chemical composition of the top gas. However, this is not easy either, since there are many effects on the gas composition.

The gas composition of the top gas is often indicated as “Top Gas Efficiency”, “Top Gas Utilization” or ETA CO expressed as $\eta_{CO} = CO_2/(CO+CO_2)$, which is typically in the range of 45–50%. Increase of η_{CO} means, that the oxygen from the burden is removed with less gas thus with less coke and the furnace becomes more efficient.

The complications with the interpretation of the gas utilization come from:

- Interference with driving rate of the furnace: the faster the burden descends, the more oxygen is removed from the burden. If the driving rate increases, the top temperature will go down.
- Interference with burden level: if burden level is low due to charging problems, gas utilization will decrease, since the first reduction step from hematite to magnetite does not take place. When filling the furnace, the gas utilization improves above the earlier level.
- Competition between hydrogen and carbon monoxide. Part of the oxygen of the burden is removed with hydrogen. So as soon as an operator changes the setting for moisture, the CO gas utilization changes. The same applies for changes in set points of injectants.

Another aspect that follows from the gas analysis is the amount of coke that is used in direct reduction reactions. This becomes manifest from the percentage of CO plus CO₂ in the top gas. However, this percentage also varies with the moisture and oxygen percentage in the hot blast, so it is recommended to have an online calculation available, which expresses the direct reduction in carbon rate or coke rate per tonne hot metal.

8.3 Following the gas in the furnace

In the following sections, we follow the gas from the moment it is generated at the tuyeres until the moment it reaches the burden level.

8.3.1 The raceway

In an operating blast furnace, the use of the reductants can be as presented in Table 8.1.

	Input (kg/tHM)	Replacement ratio	Input, as standard coke (kg/tHM)
Coke	300	1	300
Coal	200	0.85	170
Total			470
			Use, as standard coke (kg/tHM)
Total input	470		
Carburisation	-50		
Direct Reduction	-120		
Gasified in front of tuyeres	300		
Gasified coal	-170		
Gasified coke	130		

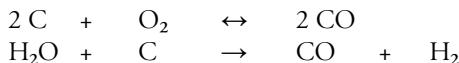
Table 8.1 Reductants in a blast furnace, typical example

The 300 kg/tHM standard coke in Table 8.1, which is gasified in front of the tuyeres, consists of 170 kg/tHM coke equivalent injected as coal and 130 kg/tHM coke (300–170 kg) is gasified at the tuyeres. Note the issue of efficiency: if the same amount of oxygen is blown into the furnace, thus maintaining the 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%! 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.

Bosh gas composition

The heat of the blast and the heat generated by the reactions of coke (and coal or 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 6500 m³ STP/min with 26 % 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: 4810 m³ STP/min ((1–0.26)x6500)
- Oxygen: 1690 m³ STP/min (0.26x6500)

The oxygen generates two molecules of CO for every O₂ molecule, so the gas volume is 8190 m³ STP/min (4810+2x1690). The gas consists of 59 % nitrogen (4836/8190) and 41 % CO (2x1664/8190).

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.

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 (see e.g. Table 8.2).

Flame temperature: temperature in raceway after all oxygen has been converted



The flame temperature is normally in the range of 1900 to 2300 °C and is influenced by the raceway conditions. The flame temperature increases if:

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

Metric Units		
RAFT =	$1489 + 0.82 \times BT - 5.705 \times BM + 52.778 \times (OE) - 18.1 \times Coal/WCx100 - 43.01 \times Oil/WCx100 - 27.9 \times Tar/WCx100 - 50.66 \times NG/WCx100$	
Where	BT	Blast Temperature in °C
	BM	Blast Moisture in g/m³ STP dry blast
	OE	Oxygen enrichment (% O₂ – 21)
	Oil	Dry oil injection rate in kg/tHM
	Tar	Dry tar injection rate in kg/tHM
	Coal	Dry coal injection rate in kg/tHM
	NG	Natural gas injection rate in kg/tHM
	WC	Wind consumption in m³/tHM

Table 8.2 RAFT Calculation (source: AIST)
RAFT = Raceway Adiabatic Flame Temperature

The flame temperature decreases, if:

- Moisture in the blast increases.
- Reductant injection rate increases, since cold reductants are gasified instead of hot coke. The precise effect depends also on auxiliary reductant composition. Table 8.3 gives some basic rules with respect to flame temperature effects.

	Unit	Change	Flame temp. (°C)	Top temp. (°C)
Blast temp.	°C	+ 100	+ 51	- 9
Coal	kg/tHM	+ 10	- 27	+ 9
Gas	kg/tHM	+ 10	- 72	+ 19
Oil	kg/tHM	+ 10	- 40	+ 12
Oxygen	%	+ 1	+ 33	- 14
Moisture	g/m³ STP	+ 10	- 42	+ 8

Table 8.3 Flame temperature effects, rules-of-thumb (calculated with mass and heat balance for our standard operating condition of Chapter II).

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 tonne hot metal results in less gas for heating and drying the burden.

8.3.2 Carbon and iron oxides

Note that in the second step, much more heat is generated than in the first step. Therefore, it is worthwhile to convert CO to CO₂ as much as possible in the process. The ratio CO₂/(CO+CO₂) is called the gas utilization or gas efficiency and is used extensively in blast furnace operation.

In Figure 8.1, the equilibrium $2\text{CO} \leftrightarrow \text{C} + \text{CO}_2$ is presented for various temperatures. The line indicates the equilibrium of the “Boudouard” reactions. At temperatures above 1000 °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. The formation of carbon from CO slows down at lower temperatures and is most pronounced at temperatures of 500–550 °C. (Biswas, p 70). 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.

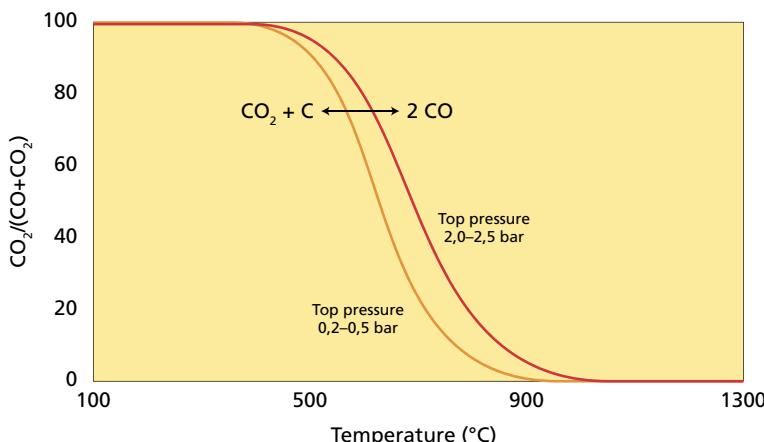


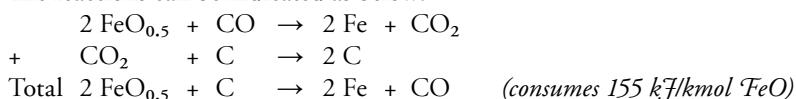
Figure 8.1 Boudouard reaction: the drawn line indicates equilibrium at low top pressure furnaces and high top pressure furnaces (after Biswas, page 71)

Direct reduction of iron oxides

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 atom 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.

The reactions can be indicated as below:



The direct reduction reaction requires an enormous amount of heat, which is provided by the 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 8.2.3.

Note the following 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 online. Experienced operators are well aware that as soon as the direct reduction rate or the solution loss increases, the blast furnace burden 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.

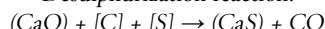
Direct reduction of accompanying elements

In addition to the direct reduction of iron (typically from $\text{FeO}_{0,5}$), some other materials are also directly reduced in the high temperature area of the furnace. The amount of coke used for these direct reduction reactions is indicated in the table below. This can be calculated from the chemical composition and the atomic weights, considering that the amount of oxygen removed reacts with the carbon in the coke. The 123.3 kg coke for direct reduction (Table 8.4) corresponds with an additional 200 m³ STP of CO gas.

Material	Reduced to	Typically in hot metal (%)	Coke used (kg/tHM)
$\text{FeO}_{0,5}$	Fe	94.50	116.7
SiO_2	Si	0.40	3.9
MnO	Mn	0.30	1.0
TiO_2	Ti	0.05	0.3
P_2O_5	P	0.07	0.8
S in hot metal	S in slag ¹	0.90 (in slag!)	0.9
Total coke used for direct reduction			123.3

Table 8.4 Coke consumption for direct reduction, typical example

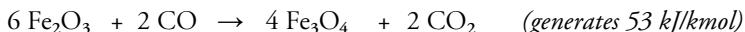
¹ – Desulphurization reaction:



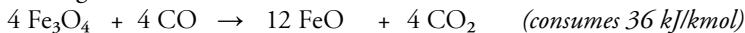
Gas reduction or “indirect” reduction

As the temperatures of the gas decrease, the CO_2 becomes stable and reduction reactions can take place, such as (see Figure 8.2):

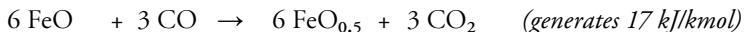
- for hematite:



- for magnetite:



- for wustite:



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 to 1100 °C. At the start of melting (1100 to 1150 °C), $\text{FeO}_{0.5}$ is normally reached. Here FeO is used as a symbol for wustite, however the most stable composition is $\text{Fe}_{0.95}\text{O}$. The reactions are shown in Figure 8.2.

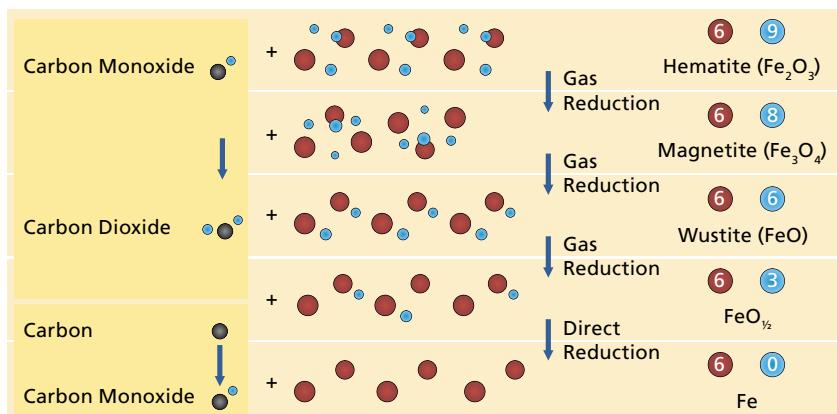


Figure 8.2 Overview of the reduction of iron oxides (black dots are carbon atoms, blue dots are oxygen atoms and red dots are iron atoms)

The equilibrium between the various iron oxides and the gas is shown in Figure 8.3. 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 gas with a relatively high percentage of CO. Gas utilization for reduction of wustite should be below 30 %. If the CO_2 level is higher, wustite is no longer converted to iron.

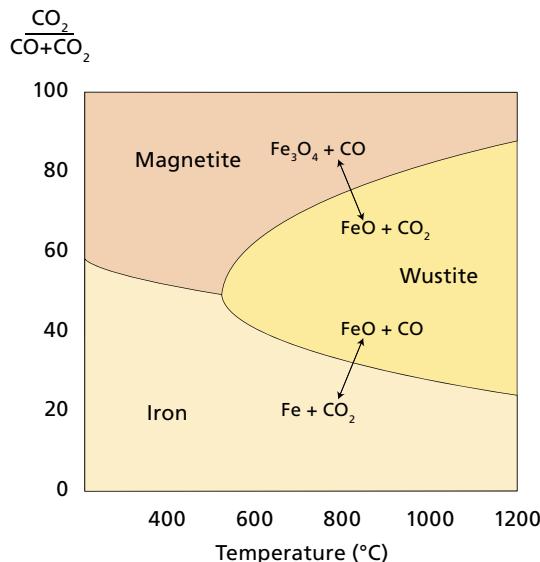


Figure 8.3 Schematic presentation of the relation between temperatures, CO/CO_2 gas composition and iron oxides. The drawn lines indicate equilibrium. Reduction of hematite to magnetite is not shown.

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 8.4 on the next page.
- Gas: by sending gas sampling devices down into the furnace, the progress of temperature/gas composition can be derived. Figure 8.5, also on the 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.

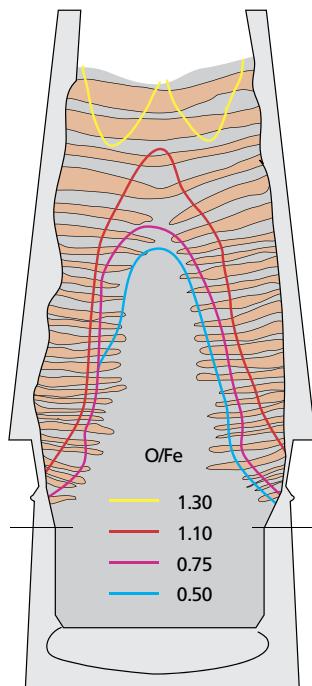


Figure 8.4 Reduction progress in a quenched furnace
(Hirohata, after Omori, 1987, p. 8)

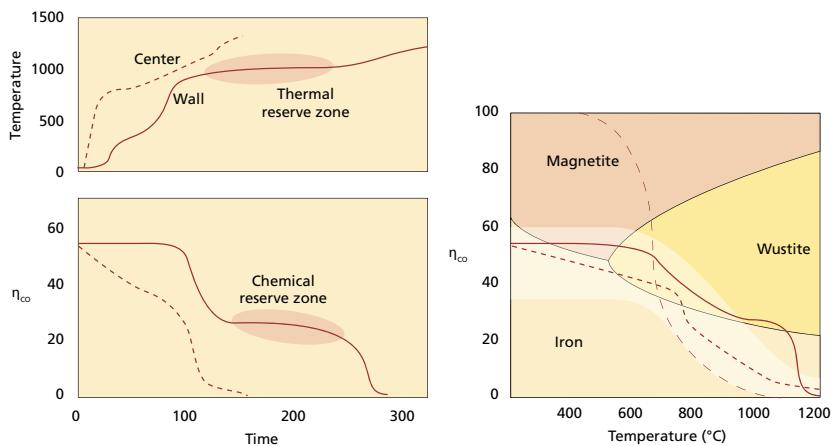
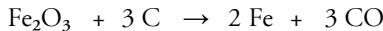


Figure 8.5 Gas composition in an operating furnace. CO, CO₂, H₂ and temperature were measured with descending probes (Chaigneau et al, 2001). Typical measurements from various furnaces are shaded. (After McMaster, 2002.)

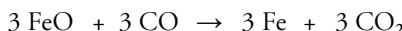
8.3.3 Gas reduction and direct reduction

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

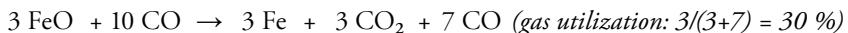


Iron contains about 945 kg/tHM Fe. Coke contains about 87.5 % carbon. Atomic weights of Fe and C are 55.6 and 12. A tonne of iron contains 17 kmol (945/55.6). For every iron atom, we need 1.5 carbon atoms, for a 25.5 kmol carbon requirement (1.5×17), which equals 306 kg (25.5x12). In addition, about 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, since too little heat is generated.

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 8.3) that for gas reduction, the maximum gas utilization is 30%. To get 30% gas utilization, more CO is needed and the reaction becomes:

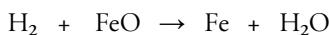


So the coke requirement is calculated as above: every tonne of iron contains 17 kmol. There is a need of 10 atoms carbon per 3 atoms of Fe. So the carbon requirement is 57 kmol ($10/3 \times 17$), which corresponds with 684 kg carbon (57x12). Again, the extra 45 kg carbon in iron has to be added, giving a carbon rate of 729 kg/tHM and a coke rate of 833 kg/tHM ($729/0.875$). 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 reactions. Approximately 60–70 % of the oxygen is removed by gas and the remaining oxygen is removed by direct reduction.

8.3.4 Hydrogen

Hydrogen is formed in the raceway from moisture (H_2O) in the blast and injectants (hydrocarbons). Hydrogen can act as a reducing agent to remove oxygen and form water. The reaction is similar to that for carbon monoxide:



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

- Figure 8.6 shows the equilibrium of the iron oxides and hydrogen. Hydrogen is more effective for the reduction at temperatures above 810 °C. From measurements in the blast furnace it has been derived that hydrogen reactions are already nearly completed at this temperature.
- Hydrogen utilization as measured from the top gas is normally around 40 % while CO utilization is close to 50 %. At the FeO level (900 °C), hydrogen is utilized for 35 %, which means that it is already close to its final utilization of 40 %.
- Hydrogen is less effective a reductant at lower temperatures, because it consumes heat when reducing iron oxides.

At high temperatures, H₂O that is formed in the furnace reacts with coke:



This reaction consumes a lot of heat. At higher temperatures (over 1000 °C), the reaction proceeds rapidly to the right hand side.

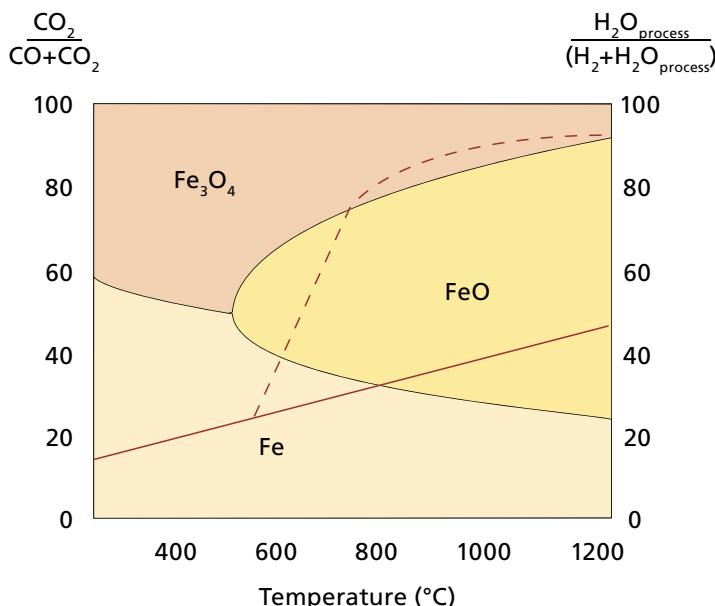
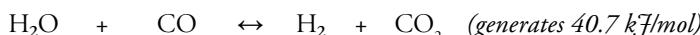


Figure 8.6 Equilibrium iron oxides with hydrogen and carbonmonoxide

At high temperatures (over 1000 °C), water vapor gasifies coke. At lower temperatures (800–1000 °C), the water–gas shift reaction becomes important:



In this temperature range, hydrogen is more effective as a reductant than carbon monoxide. This is shown in Figure 8.7, where the ratio H₂-utilization/CO-utilization is higher than 1.

This water-gas shift reaction shifts to the left when the temperature decreases. The reaction approaches equilibrium rather fast at temperatures over 800 °C, probably catalyzed by metallic iron. But the reaction fails to reach equilibrium at temperatures below 700 °C. In operational practice, the ratio in the top gas is typically 0.8–0.9.

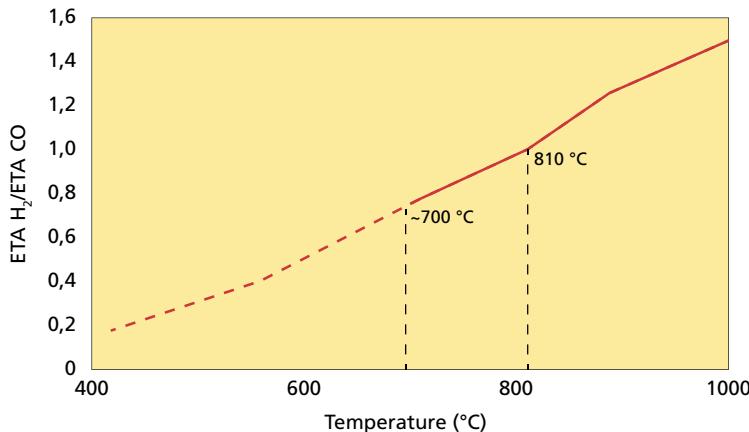


Figure 8.7 Comparison of efficiency of reduction of iron oxides by H₂ and CO gas, shown as the ratio of hydrogen utilization (ETA H₂) and CO utilization (ETA CO). The red line indicates equilibrium, the red dotted line indicates the range where the equilibrium is not reached. Data from Biswas, 1981, p 85.

Note that the hydrogen utilization cannot be measured directly. The H₂O formed in the process cannot be discriminated from the water put into the furnace with coke and burden moisture. The hydrogen utilization of the top gas is defined as $\text{ETA H}_2 = \text{H}_2\text{O}/(\text{H}_2 + \text{H}_2\text{O}_{\text{process}})$. The $\text{H}_2 + \text{H}_2\text{O}_{\text{process}}$ can be derived from the hydrogen input and the hydrogen leaving the furnace can be measured with the gas analysis.

When working at high hydrogen input (via moisture, natural gas, coal), the competition between the reduction reactions will lead to lower top gas CO₂ utilization. The simple reasoning is that H₂ competes with CO. All oxygen taken by H₂ is not taken by CO and thus CO level increases and CO₂ level decreases. 1 % extra H₂ in the top gas will lead to 0.6 % extra H₂O_{process} in the top gas and thus to a 0.6 % lower CO₂ and a 0.6 % higher CO percentage. 1 % extra top gas hydrogen leads to a decrease in top gas CO-utilization 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 FeO level, a 1 % increase in top gas hydrogen leads to a decrease of 0.8 % in top gas CO-utilization.

8.3.5 The temperature profile and chemical reactions

The temperature profile and the chemical reactions in a blast furnace are closely related. It is summarised in Figure 8.8. The reduction of the oxides to wustite takes place at temperatures between 800 and 900 °C. Thereafter, in the temperature range of 900 to 1100 °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.

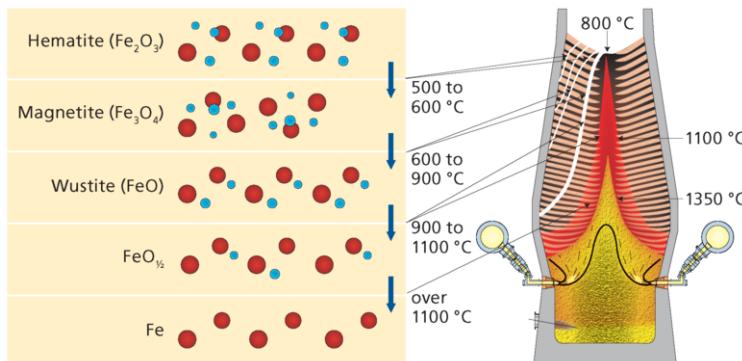


Figure 8.8 Progress of the reduction reactions and temperature of the burden

8.3.6 Overview of what happens with the gas in the burden

In the preceding section, the temperature profile in the blast furnace has been shown. In this section, the gas in the furnace will be dealt with in more detail.

- Step 1 Wind is blown into the tuyeres along with coal or natural gas and moisture. All these components react to form carbon monoxide (CO), hydrogen and nitrogen. So, the conditions at the end of the raceway are a high temperature of 2000 to 2200 °C and CO, H₂ and N₂ in gaseous form.
- Step 2 The gas ascends in the furnace and cools down to 1100 °C. The direct reduction reactions take place generating additional CO gas. When reaching 1100 °C, the gas leaves the cohesive zone and enters into the furnace stack filled with granular materials. At temperatures over 1100 °C, gas reduction is very limited as the CO₂ formed by direct reduction reacts instantaneously with coke to return to CO, a reaction which is thermodynamically equivalent to direct reduction.
- Step 3 The gas ascends further and its temperature decreases from 1100 to 900 °C. In this temperature range, the hydrogen is very effective and about 35 % of the hydrogen picks up oxygen from the ore burden. About 24 % of the carbon monoxide does the same.
- Step 4 The gas ascends further reaching an area of 500 to 600 °C. At this temperature, the ore burden has the composition of magnetite, Fe₃O₄.
- Step 5 The gas cools down further to the temperature at which it will leave the top (110 to 150 °C). In this area, the carbon monoxide is utilized further and removes more oxygen from the ore burden.

Once the temperature of the gas has dropped below 1000 °C, the total gas volume in m³ STP remains the same and only the composition changes, as shown in Figure 8.9. It is clear that the major part of the gas travelling through the furnace is nitrogen, which is chemically inert and only delivers its heat from the hot blast to the burden. During its six to twelve second journey through the furnace, it cools down from the blast temperature to the top gas temperature.

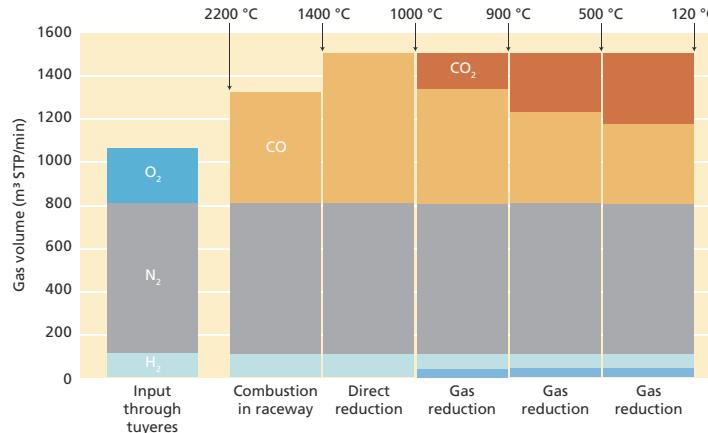


Figure 8.9 How top gas is formed from wind

For comparison, in Figure 8.10 the effect of the gas on the burden weight is shown as well. The graph is now in kg/tHM. The weight at the top of around 1600 kg ferrous burden and 300 kg coke decreases gradually to 1000 kg hot metal and 200 kg slag. The balance is eliminated from the furnace with the top gas. How hot metal and slag are formed is discussed in the next chapter.

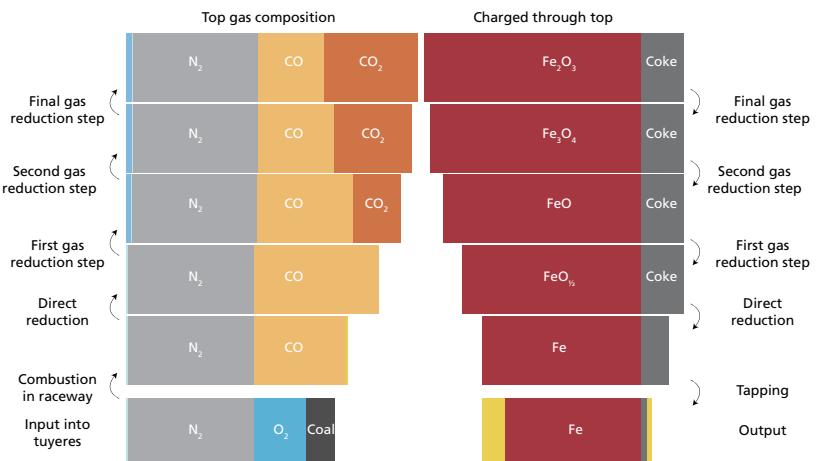


Figure 8.10 Blast furnace process showing counter-current gas and material flow and the weights at varying temperatures. Note that the weight of the hot blast is about equal to the weight of the hot metal and slag tapped.

Another method to describe the counter-current mass and heat exchange in the blast furnace is with a graphical tool called the Rist diagram. This is more advanced and shown in Annex 8.

8.3.7 Burden permeability and K-factor

The maximum pressure difference over the furnace content determines how much hot metal can be produced. The pressure difference depends on the amount of wind blown into the furnace, which generates hot gas, and the permeability of the material column between tuyeres and burden level.

Blast furnace operators have tried to develop a characterization of the total column by monitoring how much pressure is required to drive the bosh gas through the furnace. The permeability of the column is characterized by the K-factor:

$$K = [(BP + 1033)^2 - (TP + 1033)^2] / (BGV)^{1.7}$$

Where BP is the blast pressure in kg/cm², TP is the top pressure in kg/cm² and BGV is the Bosh Gas Flow in m³ STP/min. The factor 1033 has to do with changing pressure from above 1 atmosphere to absolute pressure. Note that the K-factor is called permeability factor, but the higher the K-value the poorer the permeability.

Some companies use K-values over upper and lower part of the furnace. In this situation, the results depend very much on the height where the pressure tap is installed in the furnace. Another method in use is to monitor the simpler $\Delta p/V$, where Δp is the pressure difference between hot blast and top pressure and V is the blast volume.

Interpretation of the K-value is not so straightforward. There are many influences on the blast pressure. One remark: in the blast furnace, the top temperature can vary widely. The higher the top temperature, the more real cubic meters of gas have to be pushed through the furnace. The K-value uses m³ STP of gas and thus neglects this top temperature effect.

IX Hot Metal and Slag

This chapter deals with the subject of hot metal and slag formation in and around the cohesive zone of the blast furnace and how it reaches its final composition and properties. Typical final hot metal and slag compositions are given in Table 9.1. Hot metal leaves the furnace with a typical temperature in the range between 1480 and 1520 °C.

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 %			
Sulfur	S	0.03 %	Sum	96 %	
Phosphorus	P	0.07 %	Sulfur	1 %	

Table 9.1 Typical hot metal and slag composition

9.1 Formation of hot metal and slag

9.1.1 Ferrous burden

Ferrous burden is the term used to describe the iron-containing materials that are charged to the furnace: sinter, pellets and lump ore. The melting properties of these materials depend on the local chemical slag composition. Lump ore has its natural slag composition as it is found in the earth, gangue consists of mainly acid components like SiO₂ and Al₂O₃. Pellets and sinter have an artificial composition with components added to the natural iron ores, such as limestone (CaCO₃), dolomite (MgCO₃, CaCO₃), olivine (2MgO, SiO₂) and others. Sinter has a basicity (CaO/SiO₂) over 1.6, which may even be as high as 2.8 or higher. Pellets have a wide variety of chemical compositions, especially acid pellets (CaO/SiO₂ below 0.2) or fluxed pellets (CaO/SiO₂ over 0.8).

The chemical composition of the materials is not only based on the design of the optimum properties of the final slag with respect to melting temperature, fluidity and desulphurizing properties, but also on the design of the metallurgical properties of the sinter and the pellets. Optimal metallurgical properties mean that the materials should have good reduction-disintegration

properties and melting temperatures as high as possible. The reason for these requirements is defined by the nature of the blast furnace process, being a gas reduction process. If material falls apart into small particles, the gas flow through the ore layer is impeded and the normal reduction process is limited. In addition, materials that start to melt form an impermeable layer and will also affect the reduction progress.

Note that the efficiency of a blast furnace is largely determined by the gas reduction process, and the amount of oxygen bound to the iron, which is removed by gas (CO and H₂).

9.1.2 Reduction from hematite to magnetite and reduction-disintegration

The reduction process starts at temperatures of about 500 °C in the atmosphere of a reducing gas, that is, the blast furnace top gas. The reduction of hematite (Fe₂O₃, O/Fe = 1.5) to magnetite (Fe₃O₄, O/Fe 1.33) takes place rather easily and generates a small amount of heat. In hematite, six atoms of iron are bound to nine atoms of oxygen, which changes to eight atoms of oxygen upon transition to magnetite. The ninth oxygen atom is bound to the CO gas, forming CO₂.

The first step in the reduction process has a profound effect on the properties of the ferrous burden. The crystal structure where six iron atoms and nine oxygen atoms were conjoined is forced to change to six iron atoms on eight oxygen atoms. The crystal structure changes with an associated volume increase of about 10 % and this leads to stress within the particles and the particles can fall apart. This is called reduction-disintegration, and is represented by the Reduction-Disintegration Index (RDI) or, sometimes by Low Temperature Breakdown (LTB) (Chapter III). Pellets are not very prone to reduction-disintegration, as pellets have about 30 % voidage in the structure, which can accommodate expansion.

Some lump ores have a very tight structure and are difficult to reduce, with the reduction starting on the outside of the particle. These lump ores will have reasonable RDI values, however, if a lump ore has a relatively open structure, which is easily permeable for gas, then the RDI will be poor. Lump ores with this characteristic are not suitable for direct use in the blast furnace.

Sinter, on a microscale, has a relatively tight structure with limited possibilities for local expansion. Therefore, sinter has inherently poor RDI unless measures are taken to improve it. The RDI can be improved by impeding the formation of the secondary hematite on the sinter strand. Secondary hematite is the material which is re-oxidized from magnetite back to hematite on the sinter strand. This takes place when sinter is cooled with air. These secondary hematites are very prone to reduction-disintegration in the blast furnace. The reduction-disintegration stops when all hematite is reduced to magnetite.

9.1.3 Gas reduction of magnetite

The magnetite (Fe_3O_4) is further reduced by gas (CO and H_2) to wustite ($\text{FeO}_{1.05}$). At around 900 °C, equilibrium is reached between the reducing power of the gas and the composition of the iron oxides, that is the FeO level of one atom of oxygen per atom of iron. In this area, the temperature is relatively constant (thermal reserve zone), as is the chemical composition of the gas (chemical reserve zone). When blast furnaces are operated at very high productivities, this reserve zone becomes smaller and is ultimately eliminated.

At temperatures around 900 °C, the temperature of the coke is still too low to react with the CO_2 gas. The coke reactivity reaction ($\text{CO}_2 + \text{C} \rightarrow 2 \text{CO}$) starts around 1050 °C. Therefore, all reduction is taking place by means of gas reduction ($\text{Fe}_2\text{O}_3 + \text{CO} \rightarrow 2 \text{FeO} + \text{CO}_2$), and in this temperature range also for a small part by hydrogen reduction ($\text{Fe}_2\text{O}_3 + \text{H}_2 \rightarrow 2 \text{FeO} + \text{H}_2\text{O}$). The gas reduction continues to a gas temperature above 1000 °C and a reduction of iron oxide to a level of $\text{FeO}_{0.5}$. The higher the temperature, the more H_2 contributes to the gas reduction. The gas reduction continues to rise until the temperature has risen to that where the coke reactivity reaction begins. If material starts to soften and melt (around 1100 °C), the direct reduction reaction ($\text{FeO} + \text{C} \rightarrow \text{Fe} + \text{CO}$) will take place. At that moment, the atomic ratio of O/Fe is slightly below 0.5 atom oxygen per atom of iron. In case of natural gas injection it can decrease to around 0.35 atom oxygen per atom iron.

9.1.4 Softening and melting

Softening and melting starts at local chemical compositions with the lowest melting temperatures. This is where there are high local concentrations of SiO_2 and FeO. Internal migration of atoms will cause larger and larger parts of the particles to soften. The first internal ‘melts’ of material will form at approximately 1100 °C and will consist of gangue and to a large extent FeO, because at 1100 °C the O/Fe ratio in the ferrous burden is 0.5 or lower. In the case of fluxed pellets the first melts will form at approximately 1150 °C. If the basicity increases further, the starting temperature of melt formation increases to close to 1200 °C, where even less FeO is required. However, at the basicity of super-fluxed sinter, the formation of melts require high FeO percentages again, up to 50–60 %. This explains why reduction melting tests of super-fluxed sinter generally show a relatively large part of residual material that cannot be melted even at temperatures up to 1530 °C. When gangue starts to melt, it will come into contact with the slag components of other parts of the ore burden and the slag composition will be averaged. This happens at high FeO concentrations.

Let us consider the composition of the “primary melt”. If we have the 220 kg of final slag, the primary melt will contain a huge quantity of FeO. The amount of $\text{FeO}_{0.5}$ is 608 kg. So the FeO content of the “primary melt” is more than 70 %. (When we are using natural gas and reach $\text{FeO}_{0.35}$, the FeO quantity is still more than 400 kg and the FeO content is 65 %.)

The basicity of the remaining materials equals the basicity of the input materials. Figure 9.1 shows a simplification of the real situation in a blast furnace and shows a slag phase diagram with the three main components: CaO, SiO₂ and FeO. It shows, that slag melting starts at low temperatures, but that the basicity of the burden affects melting. As the FeO is reduced out of the primary melt, the slag liquidus temperature increases, but a much higher liquidus is seen for materials with a higher basicity.

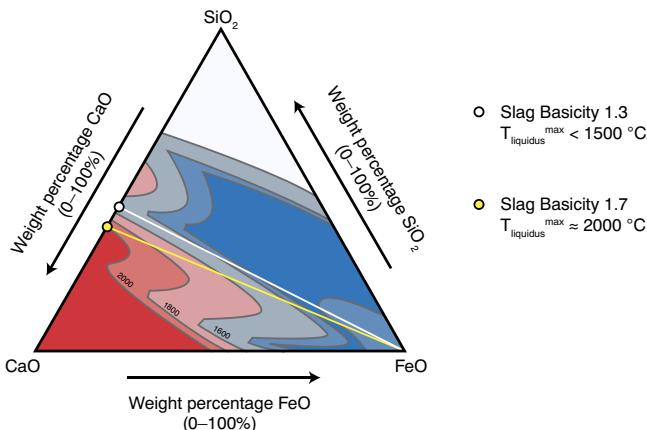


Figure 9.1 Slag liquidus temperatures of CaO–SiO₂–FeO. Slag melting starts at 60–70 % FeO in the structure. The yellow line indicates ferrous burden basicity (CaO/SiO₂) of 1.5, the white line 1.

The reduced iron, for example a sponge iron skull of a pellet, has a much higher melting temperature than hot metal. The sponge iron does not yet contain carbon and its melting temperature comes closer to the 1535 °C of the elemental iron rather than the 1147 °C of iron with 4.2 % carbon content. The reduced iron does not melt, but remains as solid particles in the melting slag–FeO mix. In summary, the first melts that are formed in the blast furnace come from acid slag components mixed with iron oxides, FeO_{0.5} and iron. As soon as melts are formed, the ore bed collapses. The order of events are: first the lump ore structure collapses due to the acidic gangue, followed by the collapse of sinter structure and finally the collapse of the pellet structure. As soon as the layers are collapsed, the permeability for gas decreases. It is estimated that permeability for gas disappears more or less completely between 1200 and 1350 °C. In that situation, the layers of cohesive material are only heated with gas flowing along its surface. Reduction by hydrogen plays a special role in this situation. Since hydrogen can easily diffuse into a more solid structure, the hydrogen reduction continues after CO reduction has stopped.

When the melts are heated further and start to drip, the melt consists of a blend of the gangue, FeO and finely dispersed iron, which has not been separated from the melt. The first process in the “primary” melt is that the gangue loses its FeO. As soon as the FeO is removed and the primary melt flows over coke,

the iron starts to dissolve carbon from the coke, which lowers the melting temperature rapidly. This has the effect of making the iron much more liquid when flowing over coke. The carbon of the coke diffuses into or is taken up by the metallic Fe, allowing the iron droplets to separate from the primary melt. After this process has taken place, the iron starts to increase in silicon content, which comes from the SiO gas that was created in the raceway flame.

As long as the slag contains FeO, the silicon in the hot metal will be oxidized back to SiO₂ and the FeO in the slag reduced to Fe. At the wall of the furnace, the root of the cohesive zone is located a small distance above the tuyeres, while in the center the cohesive zone is located higher in the furnace. As a consequence, the slag formed at the wall will have relatively high FeO and the hot metal formed at the wall will have low silicon, while the hot metal formed and dripping down in the center of the furnace will have high silicon. The final silicon level observed during a cast is a blend of these two “hot” and “cold” components.

The formation of the final composition of hot metal and slag is a stepwise process, which is illustrated in Figure 9.2.

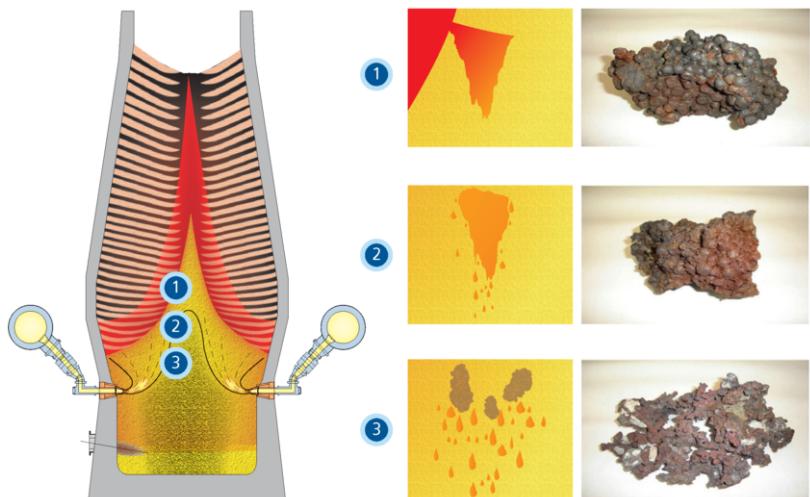


Figure 9.2 Melting of iron ore

9.1.5 Blending of burden components

The process of collapse of the burden layer followed by melting depends on local chemical composition and reduction progress. Therefore, in order to keep the variation within the layers as small as possible, good blending of components is recommended. It is shown elsewhere (section 3.8), that blending improves metallurgical and softening and melting properties. Also the use of nut coke in a burden layer results in better permeability, since nut coke does not soften with the burden and maintains a path for gas through the burden layer.

9.1.6 Slag flooding

The melting of the softened burden to hot metal and slag is a process that involves stages with very viscous liquids. The amount of those liquids that can be drained to and from the hearth may also be a limiting factor. Figure 9.3 shows a comparison between many furnaces regarding the amount of liquids (high FeO slag) that are processed per m^2 hearth surface. It shows one furnace exceeding a liquid production level of about $22 \text{ m}^3 \text{ liquids/m}^2 \text{ HA.24h}$. The limitations are not only influenced by the amount of liquids, but also by the area for the liquids to flow (furnace dimensions as well as coke voidage), the amount of gas that ascends through the dripping liquids and coke voidage and “wetting” properties of the coke for the liquids.

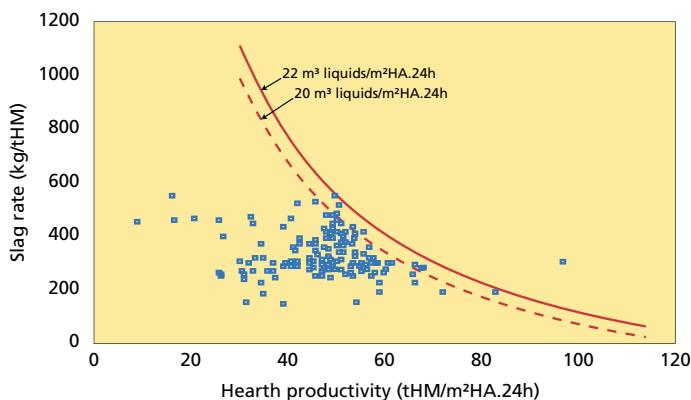


Fig 9.3 Liquid production levels of various furnaces indicative of limiting flooding conditions.

9.2 Hot metal as cast from the furnace

9.2.1 Hot metal composition and temperature

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 9.2.

The following points should be noted:

- Silicon, titanium and sulfur are concentrated in the slag.
- Manganese is concentrated in the hot metal.
- Most alkali (potassium and sodium) are eliminated with the slag, some of the alkali is discharged as vapor with the top gas.
- Nearly all the phosphorus goes to the hot metal.

Element	Input kg/tHM	Output Iron		Output Slag	
		kg/tHM	%	kg/tHM	%
Silicon	46	5	11 %	41	89 %
Manganese	6	3.6	60 %	2.4	40 %
Titanium	3	0.7	23 %	2.3	77 %
Sulfur	3	0.6	20 %	2.4	80 %
Phosphorus	0.5	0.49	98 %	0	0 %
Potassium	0.15	0	0 %	0.11	73 %

Table 9.2 Typical distributions of selected elements over iron and slag

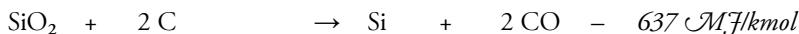
Hot metal temperature and hot metal silicon content are both used as an indicator of the thermal state of the furnace. The hot metal temperature is the result of the heat input in the furnace minus the heat used in the lower part of the furnace. The major heat consumption in this area comes from the (endothermic) chemical reactions taking place below the cohesive zone. These are the direct reduction reactions of iron, silicon, manganese, titanium and phosphorus oxides.

While the hot metal silicon shows the result of the actual chemical balance in the furnace, the hot metal temperature is somewhat delayed compared to the hot metal silicon, since there is a large heat buffer in the hearth.

Silicon, manganese, titanium and phosphorus oxides are reduced via the direct reduction reaction. These reactions consume much heat. Most of the heat is consumed for the SiO_2 reduction and consequently, the hot metal silicon content reflects the thermal state of the furnace. The manganese distribution over hot metal and slag is an even faster indicator of the thermal state because of the smaller MnO content of slag. But for operational purposes, 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.

9.2.2 Hot metal silicon

The silicon reduction reaction in the blast furnace is now discussed in more detail. Reduction of silicon from silica is possible by carbon at a temperature above 1400 °C:



Silicon transition into hot metal is promoted by reaction of iron silicide formation:



Silica reaches the high temperatures required for these reactions when coke and coal are burned and when melting, burden materials and slag are exposed to high temperatures. Coal and coke ash typically contain 45–50 % SiO₂. Silicon reduction in the blast furnace takes place via two stages (Figure 9.4).

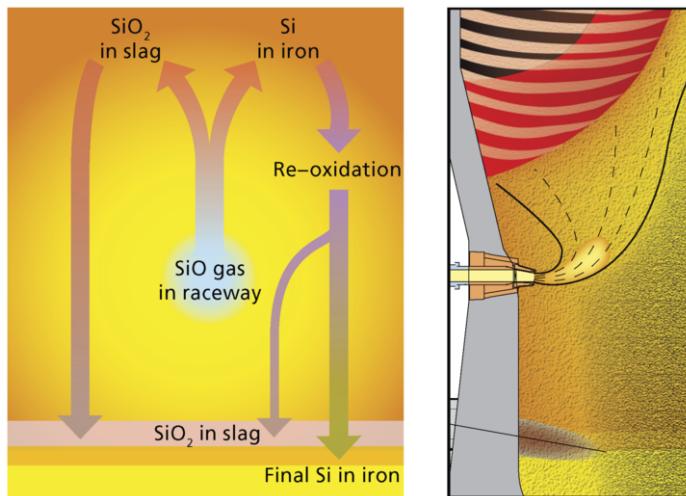


Figure 9.4 Silicon reduction

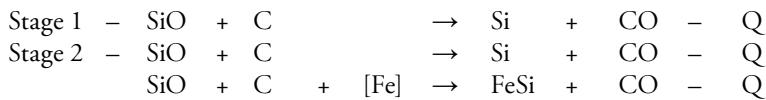
During stage 1 in the combustion zone, volatile silicon monoxide gas is formed from silica of the coke ash and injected pulverized coal ash, (Q indicating heat used for the chemical reaction):



SiO formation in raceway zone increases:

- when ash content in coke and PCI increases, and/or SiO₂ content in the ash increases.
- when the flame temperature (RAFT) increases.
- when the residence time of coke in the high-temperature zone increases, which means that smelting rate and the furnace production output are decreasing.

During stage 2, the silicon is reduced from SiO by coke in the area of the raceway and cohesive zone. It is dissolved in hot metal droplets. The following reactions take place:



Silicon reduction is accompanied by generation of CO gas. Therefore, an increase of the pressure in the furnace impedes silicon reduction and its transition into hot metal.

Research has shown, that reduction of silicon takes place in the high-temperature area above the raceway: when hot metal is sampled through the tuyeres, the hot metal has a higher silicon content than when sampled from the cast. The higher the level of the cohesive zone, the larger the volume of this high-temperature zone and the more time there is for contact of volatile SiO with the carbon of the coke and hot metal. This leads to an increase of silicon reduction and consequently higher hot metal silicon content and a higher hot metal temperature.

The contact time of hot metal with gas and coke in the coke layer between tuyere level and the cohesive zone also depends on the structure of the coke bed and the size of the coke. The more coke fines, the smaller the fraction of middle-size lumps of coke and the better the adhesive behavior for molten slag, the more the porosity of the coke layer decreases. This increases the time for the hot metal and slag to descend through it and thus there is more intensive contact with the (SiO-containing) gas, which results in additional silicon reduction. As discussed above, silicon is partially reduced by coke carbon from the slag. Silicon reduction from slag is impeded by an increase of basicity, which decreases the rate of silicon reduction from calcium silicates.

In summary: the silicon content in the hot metal can be decreased by:

- lowering the cohesive zone inside the furnace, which means running the furnace on a lower thermal level.
- using a lower flame temperature.
- increasing coke CSR and average size of coke lumps charged into furnace.
- reduction of ash content in coke and injection coal.
- increasing top gas pressure.
- increasing the percentages of CaO and MgO in the slag.
- increasing the oxidation potential in the hearth (see below).
- increasing production levels (melting rate) by increasing blast volume.

One thing not yet mentioned is that in the hearth, hot metal comes into contact with slag and remains in equilibrium with it. When hot metal droplets pass through the slag layer in the hearth, silicon may be re-oxidized by ferrous oxide, if it is present in slag:



(Square brackets, i.e. [Si], show that material is dissolved in the hot metal. Round brackets, i.e. (SiO₂), show material dissolved in slag.)

FeO is observed in slag, when the furnace is very cold. Slag is very heavy and granulated slag is black. In these situations, the furnace has a very low silicon level. From a chemical point of view, this is called increase of oxidation potential in the hearth. This mechanism also explains that the hot metal sampled at tuyere level has a higher silicon percentage than hot metal coming from the taphole.

Finally, although we observe a uniform hot metal silicon level coming from a taphole, in reality the hot metal is a blend of hot metal generated in the central part of the furnace with a higher silicon content and hot metal generated in peripheral zone with a lower silicon content.

Variability of hot metal silicon

Hot metal silicon varies from cast to cast and during a cast as well. And basically, the changes in energy input and output become manifest in the hot metal silicon. Very stable hot metal silicon is the result of stable operating conditions, the most important of which being stable burden descent. If the burden descent is not smooth, the gas reduction in the furnace varies, resulting in variation in direct reduction in the high temperature area below the cohesive zone. In this situation, the re-oxidation of silicon will vary more.

For operators it is well known, that the hot metal silicon increases, when the blast volume is decreased. From the previous section it becomes clear that when wind is reduced, the following effects will take place: the lower driving rate of the furnace allows the hot metal silicon more time for contact with (SiO_2 -containing) gas, improved gas reduction by better contact between burden and gas, sometimes improved burden descent and finally a lower top pressure. All these effects result in higher hot metal silicon at lower blast volume.

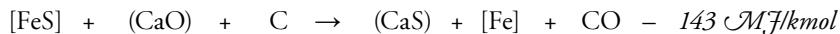
9.2.3 Hot metal sulfur

The major source of sulfur input is fuel, i.e. coke, PCI and fuel oil, bringing 80–90 % of the sulfur to the furnace, while only 10–20 % comes with the burden materials.

Most sulfur of the fuel (60–80 %) is organic, whereas the sulfur of burden materials is present in the form of sulfides and sulfates. Sulfur input at modern blast furnaces is 2.5–3.5 kg/tHM.

As soon as the coke exceeds 1000 °C, part of sulfur volatilizes. During combustion of coke and injected fuel, all sulfur is oxidized in the form of gaseous SO_2 and SO . In the more reducing atmosphere, sulfur oxides react to free sulfur (S , S_2) and its compounds CS , CS_2 , COS , H_2S , HS . Sulfur reacts when ascending through the furnace with burden components, forming calcium, iron and manganese sulfides. These sulfides descend to the hearth again. Calcium, magnesium and manganese sulfides dissolve in slag, while iron sulfide dissolves in hot metal.

More than half of the input sulfur is transferred to slag by the following reaction:



Sulfur distribution between the slag and the hot metal (L_s) is estimated as a relation $L_s = (S)/[S]$ and depends on the slag basicity and on the temperature of

hot metal and slag. The sulfur quantity transiting to the slag also depends on the slag volume per tonne. The sulfur content in gas, solid and liquid phases over the height of the furnace are shown in Figure 9.5. A relatively large part of hot metal desulfurization takes place in the taphole, when iron and slag are cast simultaneously: the reaction above shifts to the right when pressure decreases.

The sulfur brought into the furnace with fuel is first brought to the gas phase before being eliminated from the furnace. As a result, at each moment a significant quantity of sulfur is retained in the blast furnace, estimated as 4–8 tonnes for 3000–5000 m³ furnaces (at coke sulfur 0.6 %).

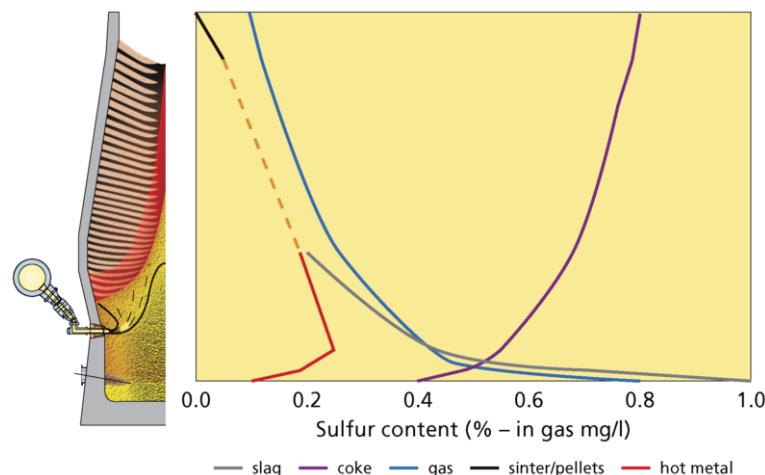
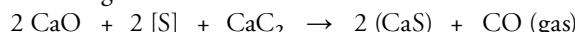


Figure 9.5 Sulfur content of gas, solid and liquid phases over the blast furnace height

9.3 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 sulfur from the hot metal by means of desulfurization. In most cases, the sulfur is removed with carbide and lime(stone) or magnesium, according to:



or



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

In comparison: in a blast furnace, the process of oxygen steelmaking is reversed. With oxygen steelmaking, the elements removed from the hot metal by blowing oxygen are first silicon and manganese, which are oxidized, then carbon is burnt and finally iron starts to be re-oxidized. In the blast furnace, the opposite takes place as is illustrated in Figure 9.6.

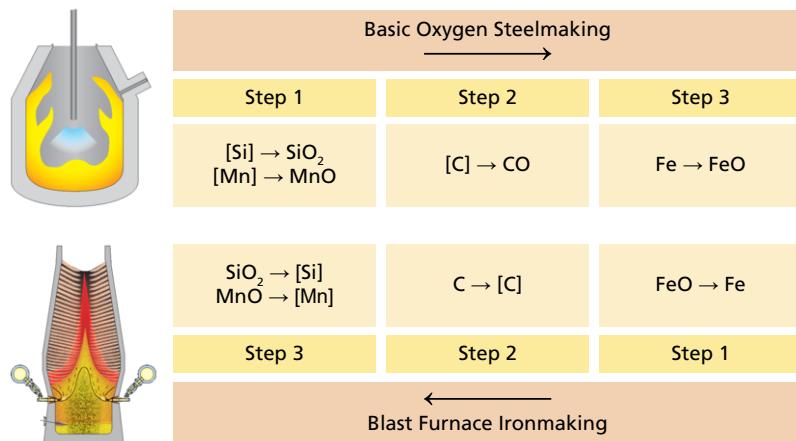


Figure 9.6 The basic oxygen furnace and blast furnace as counterparts
(Rectangular brackets indicate that the element is dissolved in hot metal)

The important considerations for a steel plant are:

- Consistent quality: the control of the converter process incorporates “learning”, which means 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 affect the slag formation.
- Hot metal phosphorus has a major influence on the steel production process. In the blast furnace, 97 to 98 % of the phosphorus leaves the furnace with the hot metal.
- Hot metal sulfur is a problem because sulfur is difficult to remove in the converter process. For high grades of steel, a maximum sulfur level of 0.008 % is required, while the blast furnace produces hot metal with a sulfur content of 0.030 % and higher. Therefore, an external desulfurization step is often required.

9.4 Slag

9.4.1 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 9.4. 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), sulfur (S), titanium (TiO_2), potassium (K_2O), sodium (Na_2O) and phosphorus (P). These components have a tendency to lower the liquidus temperature of the slag. The definitions of basicity are given in Table 9.3.

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 9.3 Definitions of basicity (weight percentage)

	Typical	Range
CaO	40 %	34–42 %
MgO	10 %	6–12 %
SiO_2	36 %	28–38 %
Al_2O_3	10 %	8–20 %
Total	96 %	96 %

Table 9.4 Typical slag compositions

9.4.2 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 behavior of slag can be well understood on the basis of its liquidus temperature. This is due to the fact that the viscosity of the slag is much more dependent on the temperature than on the Al_2O_3 content, basicity or FeO content, as shown in Figure 9.7.

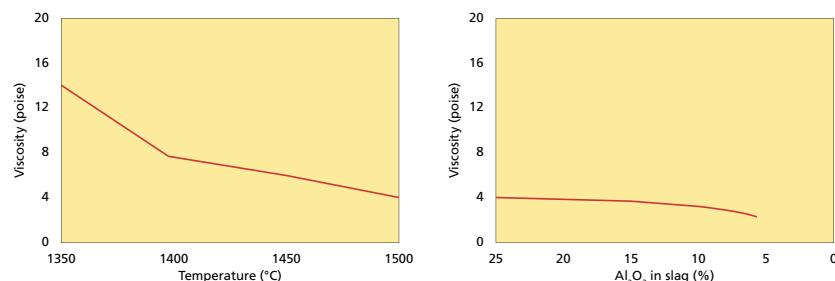


Figure 9.7 Effect of temperature and Al_2O_3 on slag viscosity in poise (10–1 Pa.s).
Left: Basicity 1,07(CaO/SiO_2), Al_2O_3 15%, 5,1% MgO (Slag Atlas p 389)
Right: Basicity 1,0, MgO 10%, measured at 1500 °C (Slag Atlas p 381)

Liquidus temperatures are presented in ternary diagrams as shown in Figure 9.8. These diagrams have been developed for pure components and in practice, the liquidus temperatures are somewhat lower.

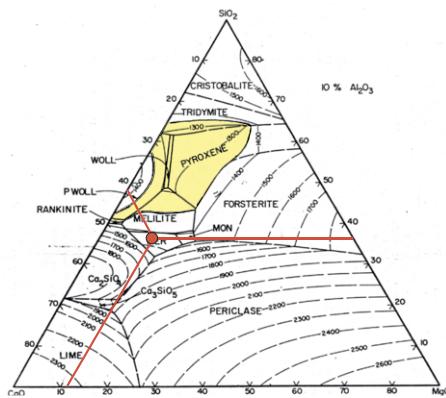


Figure 9.8 Phase diagram of liquidus temperatures of a blast furnace slag system for 10 % Al_2O_3 . The slag composition 40 % CaO , 10 % MgO and 36 % SiO_2 is also indicated. To this end, the components have to be recalculated from 96 to 100 % of the slag. The area where slag liquidus temperatures are lower than 1350 °C is indicated in yellow. (After Slag Atlas, 1981)

Since in the ternary diagrams, only three components can be indicated, one of the major slag components is taken as fixed. i.e. Al_2O_3 content is 10 %. Diagrams at different Al_2O_3 percentages are presented in Figure 9.9. The typical slag composition for a blast furnace slag is also indicated (Table 9.4). Note that the liquidus temperature is about 1400 °C and that the liquidus temperature increases when CaO increases (i.e. when the basicity increases).

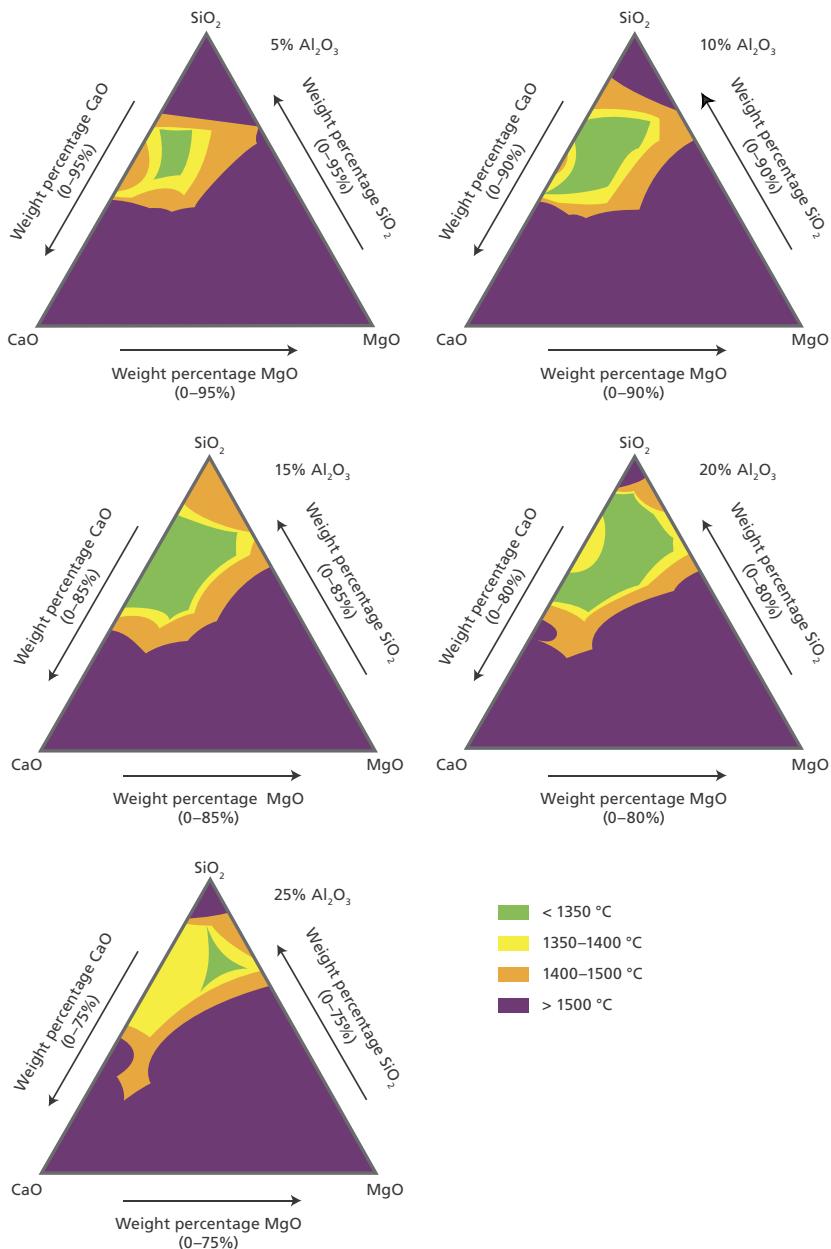


Figure 9.9 Phase diagrams of slag liquidus temperatures at various Al₂O₃ levels.
(After Slag Atlas, 1981)

High Al₂O₃ slags

The Al₂O₃ content of the final slag varies according to the raw materials in use. Especially in India, the raw materials are rich in Al₂O₃. Operational practice shows, that it is possible to run a blast furnace with around 18 % Al₂O₃.

Further increase of the Al_2O_3 content to 20–22 % is done by some companies, but even higher content is (nearly) impossible. This is shown in the small grey area of the diagram at 25% Al_2O_3 . Al_2O_3 also has an impact on the viscosity (see the right diagram in Figure 9.4), but this effect is small as compared to that of the liquidus temperature.

In Figure 9.10, 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 1500 °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 and solidus temperatures. This is indicated in Figures 9.1 and 9.11.

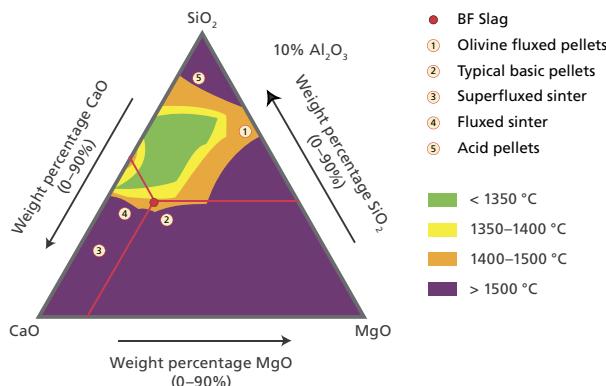


Figure 9.10 The slag composition of typical pellets and sinter qualities.
(After Slag Atlas, 1981)

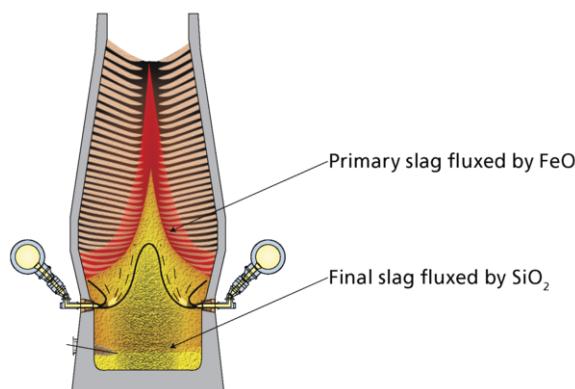


Figure 9.11 Slag formation

9.4.3 Hot metal and slag interactions at starts and stops

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 (Figure 9.12).

Slag Volume	250 kg/t	240 kg/t	230 kg/t
Al_2O_3	25 kg	25 kg	25 kg
MgO	25 kg	25 kg	25 kg
CaO	105 kg	105 kg	105 kg
SiO_2	95 kg	85 kg	75 kg
Hot Metal Silicon	10 kg	20 kg	30 kg
Si for iron	0,46 %	0,92 %	1,38 %
basicity (B3)	1.37	1.53	1.73
liquidus ($^{\circ}\text{C}$)	1420	1460	1528

Figure 9.12 Slag properties if hot metal silicon increases, a typical example.

In a situation with very high basicity, the final slag no longer remains liquid in the furnace and cannot be cast. It will remain in the furnace where it can form a ring of solid 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. A reduced slag basicity by increasing silica content is the typical practice. To this end, extra SiO_2 has to be brought into the furnace and the recommended method is the use of siliceous lump ore.

Some companies use quartzite, which is suitable to correct the basicity in normal operation. However, it is less suitable for chilled situations, since the liquidus temperature of quartzite itself is very high ($1700\ ^{\circ}\text{C}$). The effect of the use of a siliceous ore can also be shown in the ternary diagram in Figure 9.13: by working at a lower basicity, the liquidus temperature decreases along the indicated line.

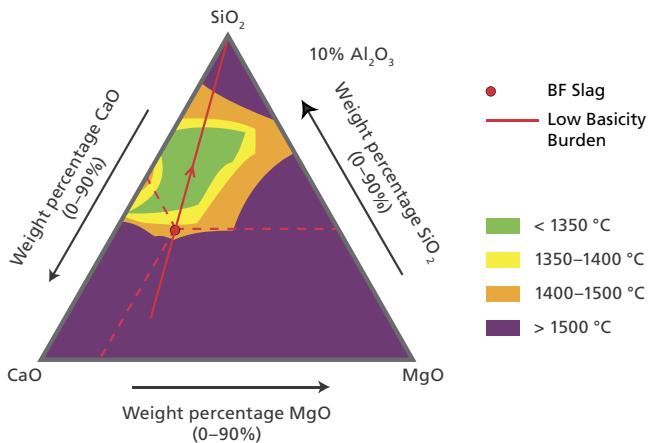


Figure 9.13 Effect of low basicity burden on slag liquidus temperatures

X Casthouse Operation

10.1 Objectives

The casthouse operation is an extremely important area for the blast furnace. The main objectives of good casthouse operation may be summarized as follows:

- To remove liquid iron and slag from the furnace at a rate that does not allow the process to be affected by increasing liquid levels in the hearth (example: slag level at the raceway elevation).
- To separate and sample the iron and slag that is cast from the furnace.
- To direct the iron to the ladle and the slag to the slag pot, pit or granulator.

The extraction of liquids from the hearth is crucial for maintaining stable process parameters, and the damaging effects of not casting the furnace will become apparent very quickly. Therefore, it is required that a cast tracking chart is available in the control room, giving the operator a quick overview of casting in the last shift or day. An example of such a chart is shown in Figure 10.1. From such a chart the slag delays, the behaviour of tapholes, problems with drilling become clear. In this chapter, the link between casting and the blast furnace process will be explained, and the factors that determine the ability to cast the furnace are discussed.

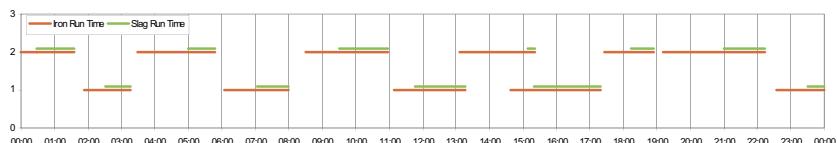


Figure 10.1 Cast tracking chart. 24 hour casthouse overview with alternating casts; red means hot metal flow, green slag flow.

10.2 Liquid iron and slag in the hearth

The blast furnace process results in liquid iron and slag being produced. These two liquids drip down into the coke-filled hearth of the blast furnace, where they wait to be tapped, or cast, from the furnace. The densities of the two liquids are quite different, with that of iron (7.2 t/m^3) being three times that of slag (2.3 t/m^3). This difference leads to very good separation between the iron and the slag once the material is outside the furnace, given the correct trough

dimensions, but also means that separation will occur inside the hearth before the liquids are tapped, see Figure 10.2.

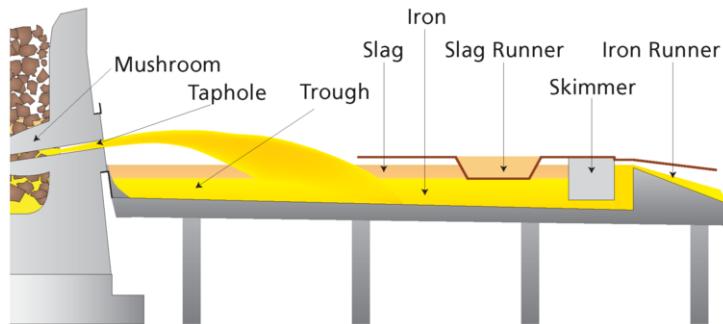


Figure 10.2 Slag and iron separation in the iron runner, or trough

The trough will still hold liquids from the preceding cast, so when the iron from the next cast starts flowing, it will then increase the level in the runner so that the iron already under the skimmer will also increase in height and start flowing again over the iron dam. This iron will then flow to the tilting runner and into a torpedo ladle. Once the ladle is full, the tilting runner will be repositioned into a torpedo ladle which is parked parallel to the full one, so this second ladle can also be filled. The full ladle will be removed and replaced by an empty ladle, so that the cast is not interrupted. This alternating between ladles via the tilting runner is done until the hearth is empty and the cast is complete.

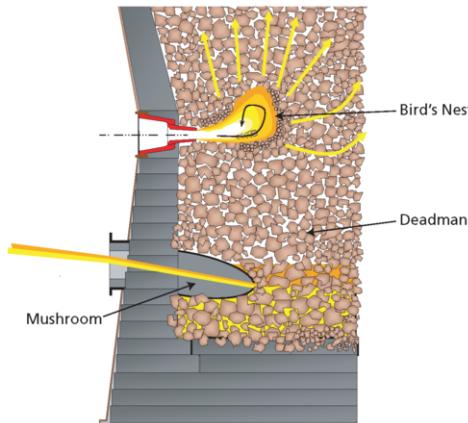


Figure 10.3 The blast furnace hearth

The slag sits on top of the iron, so it does not flow under the skimmer as long as the separation remains good. Once it has reached a certain level in the trough, it will flow over the slag dam and to either a slag granulator or a slag pit or ladle. It is very important that iron is not allowed to go down the slag dam as this can result in explosions in the granulator, or difficulties in emptying the slag pit. For

yield reasons, it is also not desirable to have slag going into the torpedo ladle. Slag in the ladle can also result in a solid slag skull at the mouth of the ladle, which results in difficulty in pouring the hot metal at the steel shop.

The hearth itself is a refractory vessel contained by the steel blast furnace shell, as shown in Figure 10.3. Cooling of the steel shell is essential to avoid overheating of the refractory and shell to the point of failure. The taphole or tapholes are positioned such that a pool, or sump, of liquids remains in the bottom of the hearth to protect the pad, even after casting. The lower part, known as the salamander, is only tapped at the end of a campaign, to allow for access to the pad for demolition and replacement.

10.3 Removal of liquids through the taphole

The regular removal of liquids from the hearth is done through the taphole, or tapholes. The number of tapholes can range from one to five, depending on the size and output of the furnace. The majority of modern high productivity blast furnaces have been between two and four tapholes. In normal operation of a furnace with two or more tapholes, the tapholes will be used alternately, with one cast being on one taphole, and the next cast being on the other. This also applies to furnace with up to five tapholes. The reason for the extra tapholes is to ensure that there are always two tapholes in operation, even through times of casthouse repair or emergency breakdown. This allows the production level to be maintained since the two tapholes can drain the hearth.

The tapholes are openings in the blast furnace shell with special refractory constructions built into the hearth sidewall. The tapholes are opened by either drilling through the refractory or by placing a bar in the refractory that is later removed. The holes are closed by forcing a plug of malleable refractory clay into the hole, which quickly hardens to securely seal the hole. In normal operation, this taphole clay will extend into the hearth, forming a taphole mushroom that will protect the original refractory construction (Figure 10.4).

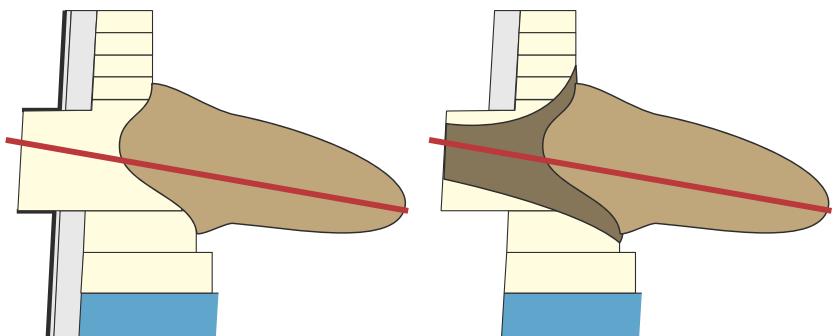


Figure 10.4 Over the taphole campaign, the original lining will gradually be worn away and replaced by taphole clay

The tapholes are perhaps the most vulnerable areas of the blast furnace due to the constant wear and tear and reliance on consumable materials, equipment and manual intervention. If any of these factors are performing less than optimally, then deterioration in the taphole performance is the likely result.

The common taphole degradation causes are listed below;

- Improper (e.g. not central) drill positioning when opening the taphole.
- Manual oxygen lancing to open the taphole.
- Clay leakage out of the taphole on closing the hole.
- Water leakage from inside the furnace.
- Gas leakage through refractory surrounding the taphole itself.
- Slag and iron attack – both chemical and physical.

The liquid iron and slag flow from the taphole are determined partially by the flow to the taphole on the inside of the hearth, but also by the characteristics of the taphole itself, such as:

- The length of the taphole, which is affected by the plugging practice and the clay quality.
- The diameter of the taphole, both the diameter of the drill with which it was opened, but more the wear or erosion of the taphole over the time of the cast.
- The roughness of the surface of the taphole.
- The pressure inside the furnace, consisting of the furnace blast pressure and the liquid hydrostatic pressure.

As the taphole will wear through the cast, especially when slag starts to flow, the rates of iron and slag flow are not constant throughout the cast. Even with good casting regimes there will be some variation in the hearth liquid level, with the desired situation being as little variation as possible. The taphole clay quality determines the resistance to slag attack, and therefore the choice of clay quality is very important. This is often determined by availability of local supply, and so is not discussed in detail here.

The length of the taphole is determined by the amount of clay injected, and so more clay is always injected than is needed to just close the taphole. The excess clay is pushed beyond the end of the taphole and forms a ‘mushroom’ at the opening of the taphole into the hearth itself. This mushroom protects the taphole block itself from wear. The larger the furnace, the bigger the mushroom inside the hearth, so the longer the taphole. An 11 meter furnace can expect to have a taphole length of 2.5 meters minimum, and at 14 meter hearth diameter this increases to 3 meters.

10.4 Typical casting regimes

A blast furnace will be cast between 8 and 14 times per day. These casts may last between 90 and 180 minutes, with the end of the cast indicated by a spraying of the liquids caused by gas from the raceway escaping out of the taphole. In this

time, the furnace processes a considerable part of its working volume. As shown in Chapter II, the residence time of the burden is approximately six hours. Therefore a two hour cast represents a third of the content of the blast furnace being transformed from burden material to molten iron and slag.

Figure 10.5 shows an example of regular tapping sequence using two tapholes. Most two, three and four taphole furnaces will operate in this way, with the extra tapholes being either a spare or out for maintenance.

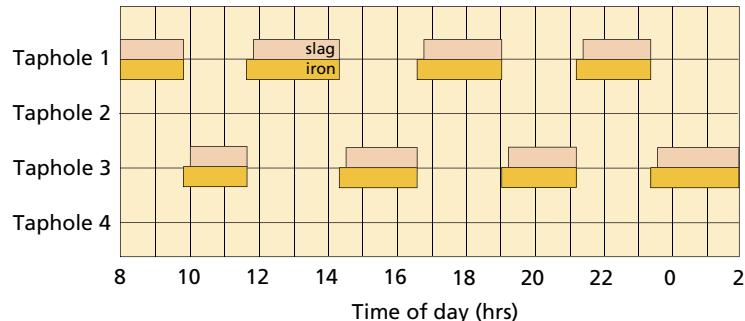


Figure 10.5 Typical casting regimes with a two taphole furnace, showing iron run times with slag above them

When the tapholes are closed, or one is open but the stream of liquid exiting has a low flow rate, then the liquid level in the hearth will increase. That is to say, the production rate is higher than the tapping rate. If this continues for long enough, then the increased liquid level in the hearth can affect the blast furnace process in the following ways:

1. The upward force on the submerged coke deadman is increased by the increased liquid level. This increase in the upward force will slow down the burden descent and increase the blast pressure.
2. If the slag level is so high that it reaches the tuyeres, then the gas flow will be severely affected, with increased gas flow up the wall. This can result in poor reduction of the burden and therefore a chilling furnace.
3. The slag can be blown high up in the active coke zone, impeding normal gas distribution.
4. If the hot metal level is so high that it reached the tuyeres, then it is possible a cut tuyere will be the result, causing water leakage into the furnace. In the worst case scenario, the tuyere will burn severely or a blow-pipe will fail. This will then lead to a blow-out of coke and a very critical emergency stop.

In order to avoid any of these effects, the hearth liquid level should be kept under control and preferably at a low level, as per the example given in Figure 10.6 on the next page. In a modern, high productivity blast furnace, measurement of hot metal and slag quantities should be registered real time, so that the casthouse crew can take timely actions.

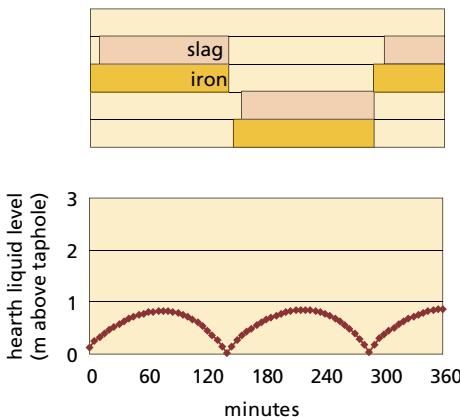


Figure 10.6 Casting and hearth liquid level

10.5 Taphole drill and clay gun

These two pieces of equipment are two of the most critical items on the blast furnace. The maintenance of these items must be of a very high standard as the availability of them on an active taphole can not be any less than 100%. Cleaning of the gun nozzle after every plug is essential for ensuring that the clay can be pushed at the next cast, which in turn will prevent the gun nozzle being burned.

It is important to keep the taphole face clean and to clean down the sides of the trough regularly so that the mud gun can swing into place without obstruction and the nozzle gets a good seal on the taphole face.

The clay quality and method of plugging the taphole with the clay are very important for both the length of the taphole and the flow rates of iron and slag. Plugging has to be done at the same position as the drill has opened the hole to avoid clay leakage.

The speed of the clay gun piston and the pressure used to force the clay into the taphole has a strong influence on the ability of the clay to plug the taphole effectively. If the clay can only partially fill the taphole, then the next time the cast is opened, the drill will have more difficulty in opening the taphole as it is also trying to cut through iron particles. This is one of the reasons why the production rate of the furnace can be limited by the taphole equipment, and so serious consideration should always be given to upgrading the clay gun and taphole drill whenever significantly higher production rates are targeted.

To preserve gas tightness of the taphole, the post-pressing technique can be applied. This technique involves pressurizing the clay with the clay gun after it has filled the hole, to try and close any small cracks or fissures in the taphole.

Ensuring that the taphole drill is in the center of the taphole each and every time is also very important as otherwise the gun will not be able to plug the taphole as well as it should, leading to less clay going in the hole and so a shortening of the taphole and potentially burning the gun. A selection of drill bit diameters can be used, although the aim diameter should be kept relatively constant when aiming for consistent tapping practices. The range of drill diameters is then useful for special situations, when the tapping is irregular, or changes to the production rate requires changes to casthouse practice.

As an alternative to the drill, a soaking bar may be used. This is a bar of solid steel that is hammered through the clay immediately after it has been pushed into the taphole, while it is still soft. The clay is then allowed to harden and the bar is pulled out. This results in a very smooth taphole of equal diameter throughout, although the hammering of the bar in and out of the taphole can increase the stresses on the taphole refractory block construction itself and introduce gas leakages.

10.6 Hearth liquid level

The level of liquids in the hearth should always be kept as low as possible. This means that the hearth should never be used as a “buffer” for the containment of produced liquids. The reason for this is that the liquid level, above a certain level, has a direct impact on the process. As shown earlier in section 7.2, the liquids in the hearth act as an upward force in the blast furnace, along with the blast pressure. Should this force be allowed to increase, it will impact on both the blast pressure and the descending burden. It is shown schematically in Figure 10.7 what happens in the furnace when the liquid level increases too far.

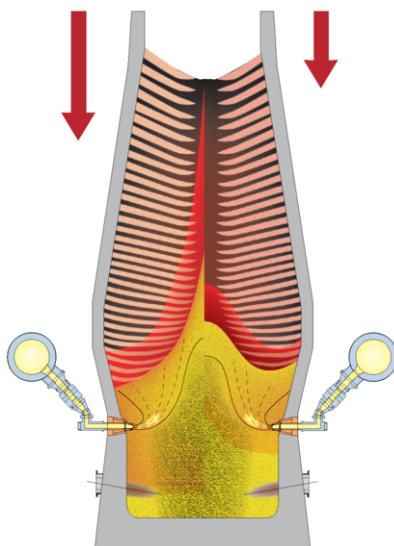


Figure 10.7 Increased liquid level (arrows indicate burden descent rate)

As shown, the high liquid level causes the blast to be deflected more towards the wall, rather than through the center of the furnace. This is because the coke in front of the tuyeres has been infiltrated with slag, and so is much less able to accept the flow of the gases produced at the raceway.

In this instance, the bosh is subject to much higher heat loads than normal, and the root of the cohesive zone will ascend. However, at the same time the center of the furnace the cohesive zone will drop due to the reduction in gas passing through the centre. The increase of gas flow on the walls could also result in the removal of skulls that have formed on the bosh walls and result in cold, solid material falling in front of the raceways interfering with injectants and cold material falling into the hearth, which could cause a hearth chill.

The blast pressure will also be higher as the resistance in front of the tuyeres is higher, and the burden descent will slow considerably. The furnace may even begin to hang, with the danger of slag filling the tuyeres should the furnace then slip, where material will quickly drop into the full bath of liquids.

The wall temperatures all the way up the stack will also increase, as the gas continues to preferentially travel against the furnace wall. This then subjects the cooling elements to a higher heat load than they will usually encounter. This increase in heat losses, coupled with the loss in furnace efficiency can lead to cooling of the furnace. In this scenario, the furnace should be cast without delay, and actions taken to restore the process stability.

Figure 10.8 shows the effect on stockline level in the case where high residual liquid level is affecting the burden descent. The burden descent slows when the taphole is closed, and then speeds up significantly towards the end of cast, to the extent that the charging system is unable to keep up and a lowered stockline is the result.

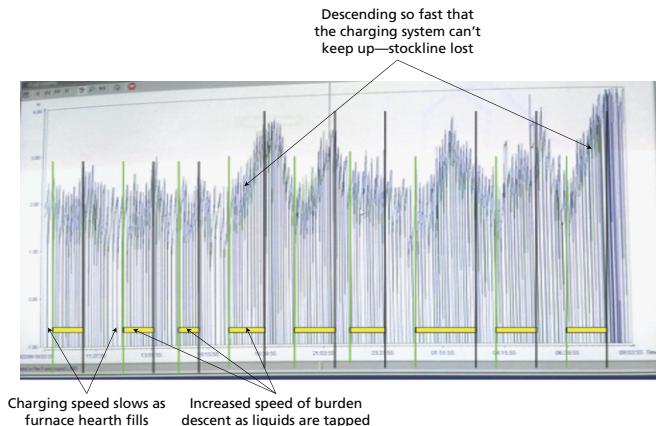


Figure 10.8 High residual liquid levels and burden descent

10.7 Delayed casting

In most plants, the casting regime will have been calculated and observed to arrive at an optimum length of time in between casts. This is referred to as the gap time or intercast time, defined by the time between stopping liquid flow by closing one taphole and starting liquid flow by opening another, or in the case of single taphole furnaces, reopening the same taphole. This will be determined by the production rate, number of tapholes, and casting rate. In the majority of cases, this casting regime will be adhered to, that is to say, the gap time will be met. However, where there are problems in meeting this schedule, remedial actions may be required.

When casting the furnace, it is required to have a good, controlled liquid flow rate from the furnace. Where a taphole is open but is not casting well, the flow should be improved by, for example, re-drilling the hole or re-drilling with a larger drill bit. If the slow flow is allowed to continue, then it is quite possible that the furnace will be producing liquids at a higher rate than they are being cast, which will lead to problems inside the furnace.

Whether the casting is delayed, or indeed the casting speed is slower than the production speed, one of the factors that affects the filling rate of the hearth in terms of height, is that of the coke bed voidage. The coke bed voidage is an unknown value. Studies have shown that it can vary between 20 and 30 %, but as yet there is no direct method of measuring it. It is also quite likely that the voidage of the coke bed will vary between the center and the periphery, and from the bottom to the top, so the assumed overall voidage is not directly applicable to every area in the coke bed. The coke quality will have a strong impact on the voidage, as the breakdown of the coke higher up in the furnace will generate fines, and a wider size distribution of particles will create a more densely packed coke bed.

By way of an illustration of filling speed, take for example an 14 meter hearth diameter blast furnace, with a taphole to tuyere distance of 4 meters producing 10.000 tonnes per day with a slag rate of 220 kg/tHM. By calculating the volume of space between the taphole and tuyeres, assuming a coke bed voidage of 20 %, the length of time until the liquid level is at the tuyere can be calculated. In this case it is 76 minutes. If the coke bed voidage is 25 %, then this increases to 95 minutes. We therefore have the situation whereby in one instance the furnace has 90 minutes of full production before the hearth liquids are at tuyere level, and another instance when it has only 76 minutes.

Once the liquid level is at the tuyeres, it is already expected that problems with blast pressure will have been experienced, so actions may already have been taken to reduce the blast volume. However, if the problems that caused the delayed casting are not resolved when the furnace has already reached this stage, then it will become impossible to take the furnace off blast without slag, and even iron flowing into the blowpipes.

For these reasons, it is considered to be good practice to take remedial actions immediately when it is known that the casting will be delayed, regardless of the reason. Estimates may be given for the completion of work, or the restoration of services, but as far as the blast furnace is considered it will continue to produce iron regardless, and if the original estimates are found to be wrong, it will often be too late to do anything but extreme reactions to try to protect the blast furnace. If the iron, and more importantly the slag, is not removed from the furnace in a timely manner, then the process will suffer very quickly, with the extreme case being a frozen hearth.

In the case where the operator is faced with a casting delay, different actions may be taken depending on the current condition of the blast furnace. If it is still casting the previous cast, and it is safe to continue to do so, then the oxygen and then wind rate may be reduced prior to closing the hole, reducing the production rate and so giving a much longer safe gap time.

In this situation, the action to reduce production rate should be aimed at safe operation continuing. For example, wind rate should be reduced to the minimum at which injection remains on the furnace. Oxygen should be decreased to the minimum, determined by a simple formula, such as for every 30 kg/tHM injection over a limit of 70 kg/tHM, the oxygen enrichment should be increased by 1 %.

Due to the uncertainty in the available voidage for hot metal and slag, it is prudent to make conservative estimates when determining the control actions to be taken.

10.8 No slag casting

As the iron is below the liquid slag, and the taphole elevation will always be at the depth of the iron pool at the start of cast, then iron will be cast before the slag. As the liquid level drops, then a mixture of slag and iron will begin to flow. At the end of the cast, the majority of liquid is slag, with iron flowing at the production rate. Sometimes, however, the furnace will cast iron without casting slag, or at least not as much as should be cast.

Although the iron is the focus of the blast furnace, the iron cannot be made without the slag, and due to the nature of it, the slag proves to be the more difficult liquid to cast. Basic slags have a higher melting temperature than acid slags, but the basic slags are more desirable for the desulfurization properties, so for hot metal quality it is required to use a more basic slag. In time of difficulties, however, one of the first actions to ensure that the furnace will be able to cast well is to reduce the slag basicity, which typically lowers the slag liquidus temperature and viscosity. This will give the operator the best chance of being able to get the slag out of the taphole.

If events in the furnace cause a change to either the temperature or the composition of the slag, then it can become much more viscous than the iron, and drainage through the coke bed becomes increasingly difficult. The iron will flow much more easily, and so it can occur that casting will continue with little or no slag being cast. The slag is still being produced, however, and so it is very important to make sure it comes out of the furnace before it interferes with the process.

The problem may be seen to be developing at an earlier stage by monitoring the following parameters:

- Amount of slag cast, measured by the number of slag pots filled or by indirect methods such as the speed at which the slag granulator drum rotates, or temperature pick-up in the granulator outlet water.
- Percent slag time – this is the number of minutes that slag has been cast divided by the number of minutes in the cast, expressed as a percentage. Ideally this number should be fairly constant and representative of the slag volume that the furnace is producing, however it is only accurate when the flow of slag is constant between casts.
- Slag over time – this is the point in time when the slag first flows over the slag dam. Slag will have started exiting the taphole before this point, but not in large enough quantity to give a good indication.
- Slag gap – this is the number of minutes from when the liquids stopped being cast at the end of the previous cast to the slag over time of the current cast.

When it is clear that the slag is not draining from the furnace as well as it should be, efforts should be made to improve the slag drainage. This may be done by a variety of methods, and it is likely that procedures already exist for it. Using a larger diameter drill bit on the next cast will increase the flow, and may improve the situation. If the taphole is already short, however, and a short cast caused the lack of slag, it may be better to increase the length of the hole so that a longer cast is the result. The problem may only be at one taphole, so changing to the other taphole will already improve the situation inside the furnace.

Opening the second taphole, so two tapholes are open at the same time (overlap casting), should be done after a defined period of no slag casting, as specified in the standard operating procedures for the plant.

If the furnace is on a cooling trend, combined with difficulties tapping slag, increasing the fuel injectant to warm up the fresh iron and slag may temporarily improve the situation, but a coke rate increase will also be required.

Shortening the gap time may also be advisable, especially when it is suspected that liquids remain in the furnace.

10.9 One-side casting

Furnaces with only one taphole are of course optimized for tapping single sided, as are some blast furnaces that follow a routine of having one taphole in operation and one as standby. The majority of two and more taphole furnaces operate on an alternating taphole basis using two tapholes. This will mean tapping through one taphole, closing it, and then either opening the second taphole immediately or waiting the designated gap time.

The single most important effect of single taphole casting compared to alternate casting is that of the gap time. During the gap time, the furnace is still producing liquids but not casting them. Ideally, the gap time is calculated as the optimum to allow enough liquid accumulation in the hearth to allow a smooth cast for the desired period of time, with good iron and slag removal, but without increasing the hearth liquid level enough to affect the blast pressure. However, the gap time can also be affected by external factors such as how long it takes to change torpedo ladles, clay cure time, maintaining and cleaning the runner system, etc. Where this is the case, it is very important to remember that the furnace is still producing liquids at the same rate, unless a change is made to slow down the production, see Figure 10.9.

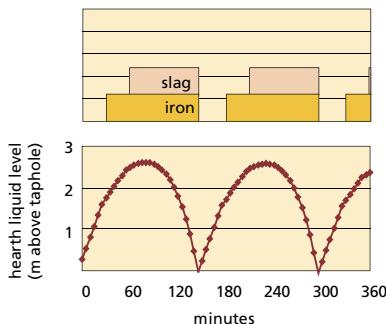


Figure 10.9 Effect of single taphole casting on hearth liquid levels

In single taphole furnaces, the minimum gap time is often dictated by the curing time for the clay. If the taphole is opened before the clay has hardened, much of it will easily wash away, which will quickly erode the taphole mushroom and expose the taphole refractory block itself. With alternating casting this is not a problem as the clay has the time that the other taphole is in use to harden. Therefore, many single taphole furnace use resin-bonded type clay that harden quickly.

The gap time has major impact on hearth liquid level and thus on the process results. In Figure 10.10, the effect of the gap time on hearth liquid level is simulated: it is clear from the figure, that in this calculation the highest hearth liquid level rises from 2.5 meters above taphole to 3.8 meters above taphole when gap time is increased from 30 to 60 minutes.

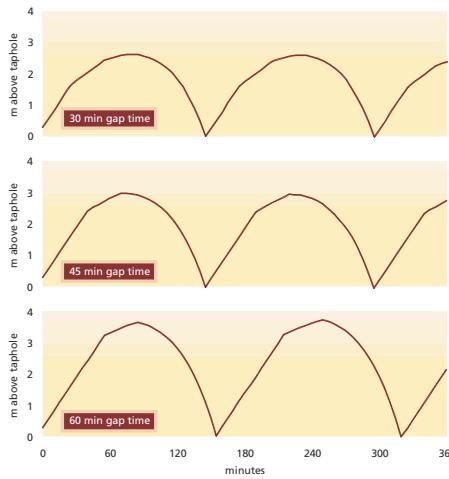


Figure 10.10 Effect of gap time on hearth liquid level, single taphole operation.

If a furnace must switch from alternate to single sided casting, the area to look at firstly is the difference in gap time between the two practices. If alternate casting requires a gap time shorter than the time it takes for the clay to harden, then single sided casting will require a change in practice. If different clay is available, then this may be applied, but caution should be used during the transition as the clay already in the hole may not combine well with the new clay.

If there is a significant difference in the gap time, then it may be advisable to reduce the production rate to minimize the fluctuation in hearth liquid levels. Experience has shown that an 11 meter hearth diameter blast furnace can produce 5500 to 6000 tHM/d with one taphole, and a 14 meter heath diameter furnace can produce around 8000 tHM/d. This is often a significant reduction compared with what the furnace is usually producing.

10.10 Not dry casts

A cast that has ended before all the liquids have been drained from the hearth is described as a not dry cast. This is reported whenever the taphole has to be stopped during a cast, such as when the torpedos are full, or there has been a problem in the casthouse that required the flow of liquids to be stopped. Other causes can be a very short taphole or a crack in the taphole mushroom. It is good practice to record the suspected reason for a not-dry cast so that improvement plans for the worst offenders can be made.

A not dry cast may also be reported when the taphole is showing signs of end of cast, when it can be reasonably suspected that the furnace is not empty. This could be when the slag is not yet over the dam, or it has only been casting for a

very short time, or not enough liquid volume has come out of the furnace. A third example of a not dry cast is more difficult to determine, and can easily be missed as the signs are less obvious and may only be picked up in the control room, rather than on the casthouse floor. In the case of a series of casts where the casting has appeared to be normal, it is still possible that some slag has been retained in the furnace after each cast. This will not be noticed after one or two, or depending on the amount, perhaps even more casts, but after successive casts where a small amount of slag has been retained in the furnace, it will build up to a large amount. At that point, the blast pressure can begin to be affected. Many casthouse now are equipped with a large digital display so the casthouse operators can see the blast pressure and the gap time, so they see the impact of a casting delay or slow running taphole.

This will be more noticeable when the furnace is closed as the blast pressure may increase, and continue to increase until the taphole is opened again. It may not decrease again until the slag begins to tap at a reasonable rate, and so lowering the level in the furnace. As the signs with blast pressure are not always a precise match with the casting times it can sometimes be dismissed as the cause. On these occasions, it is useful to look to the slag time percentage, as well as the slag run durations themselves.

Depending on the cause of the not dry cast, slightly different reactions may be appropriate. Where the not dry cast is known and the taphole is closed for operational reasons, the second taphole should be opened immediately. Where this is not possible, the oxygen and then wind rate should be reduced and the original taphole is re-opened as soon as possible. Where this is not possible, the decision to close the taphole should be delayed as much as possible, with wind rate being reduced as far as liquid levels as seen at the tuyeres will allow. At this point it is a balance between how much damage is being caused outside the furnace due to, for example, molten metal spill, compared to the danger of flooding tuyeres with slag and iron.

In the case where the taphole has shown signs of the hearth being empty, but it is thought that it is not from the casting times and amount of slag cast, then there are a few different actions that may be considered. If there is a second taphole available, then it may be opened prior to the first taphole being closed. Once this is safely open, the first one may then be closed, known as overlap casting. Alternatively, the normal gap time between casts may be reduced to zero, so the second taphole is opened immediately after the first is closed. It is important to ensure that both tapholes do not finish casting at the same time as that will introduce a gap time when there are two clayguns in the taphole waiting for the clay to harden. So once slag appears, one taphole should be closed to allow the other to cast normally. This technique of when to open and when to close a second taphole should be included in the standard operating procedure for casting to ensure that the best sequence, proven in practice, is followed by all operators.

In either case, a larger drill bit may be used to open the original taphole again, when it is due to cast. This may help in removing the liquids from this side, assuming that a short taphole length is not the cause of the problem.

Where only one taphole is available, the taphole may be closed for either a much reduced gap time, for example 10 minutes rather than 30 minutes, with a shorter clay stop. It is also possible to stop the taphole without clay for a minute or so, but it should first be checked whether the clay gun is sufficiently protected to do this. This practice should not be repeated on the same taphole as it will allow the taphole mushroom to erode too quickly, causing further problems.

These same actions may also be taken if the blast pressure is being affected by a possible build-up of slag in the furnace. At the same time, however, other causes of increasing blast pressure should also be investigated.

10.11 Defining a dry hearth

Witnessing a blow at the taphole is often considered to be the definitive criteria for whether the furnace is dry or not dry. Although a good indicator, and should never be taken for granted, a blow at the taphole only indicates that the liquids in the vicinity of the taphole are drained, and does not say anything about liquids in other areas of the hearth. When the drainage to the taphole is poor from areas far from the taphole, then it is possible for liquid levels in the area of the taphole to drop sufficiently low for a blow at the taphole to appear while there are still a lot of liquids left in the furnace. In this scenario, the taphole should still be plugged, but the cast is to be considered to be a not dry cast. Unfortunately, these are not always possible to determine from the casthouse.

The indicators of a dry hearth can be summarized as the following;

1. Casting until a blow at the taphole is witnessed.
2. Enough slag and iron has been removed from the furnace to correspond with the known production rate.
3. The process parameters show no sign of the hearth holding liquids – blast pressure normal, charging rate normal.
4. The furnace can be shut down at any time, without concern that slag or iron will flow into the tuyeres.

It is the last of these criteria that is often the defining one, where the decision to take the furnace off for a short stop is delayed until after the next cast. This in itself indicates that the operator is not confident that the hearth has been drained sufficiently to avoid any residual liquid threatening to enter the tuyeres when the blast pressure is reduced. An operator who can confidently take the furnace off blast at the end of the current cast is one who has confidence that the furnace is draining well during the cast.

10.12 Oxygen lancing

On occasion it is unavoidable to open the taphole using oxygen lancing. This practice should be considered a last resort as it is extremely damaging to the taphole refractory. Where the frequency of oxygen lancing is increasing, the situation should be investigated very closely to identify the root cause.

Where the use of oxygen lances is unavoidable, they should only ever be used by experienced casthouse workers, following the pre-drilled hole to ensure that the lance is burning in a straight line down the center of the taphole. If more than one lance is required, the interval between the two should be as short as possible, with the practice continuing until the taphole is opened. Where this is causing a long delay to the cast, alternative or additional actions such as opening a second taphole or reducing wind rate should be considered at an early stage.

Repeated use of oxygen lances to open the taphole is likely to cause irreparable damage to the taphole area, and may even pre-empt a taphole break-out or necessitate an extensive taphole repair to avoid such a break-out. There is a very large risk associated with using oxygen lances as it is very difficult to ensure that the lance is burning in a straight line. Damage to the taphole block or to taphole staves are the biggest concern.

10.13 Cast data recording

For good analysis of taphole condition and casting performance, it is important to keep very good cast records. Some of the data that should be recorded on a cast basis is as follows:

- Cast number.
- Time start drilling.
- Number of drills or oxygen lances used to open hole.
- Time liquid start flowing.
- Drill diameter used to open hole.
- Taphole length.
- Time slag over.
- Time end cast.
- Amount of clay used to close taphole.
- Clay type used.
- Blow at the taphole.
- Tap-to-tap time.

The cast end times, drill start times, iron run and slag over times can be plotted very easily to allow quick and easy interpretation of the casting. This method is often much more illustrative and quicker to interpret than the lists of times that are often meticulously recorded. Having the times plotted on a block chart that is updated constantly allows problems to be identified very quickly and so solutions applied at an earlier stage than may otherwise have been the case.

XI *Operational Practices and Challenges*

11.1 The burden

11.1.1 Fines in the ore burden

For the blast furnace operator, one of the main concerns about the raw materials is that of the fines loading. This is the proportion of undersize material in the furnace. The direct effect of a high fines loading is that it will affect the permeability of the furnace. The permeability of the ore layer is determined by the amount of fines (below 5 mm) in that layer. Unfortunately, when bulk material is handled, fines are generated. Therefore, coke and ore burden are normally screened before being charged into the furnace. But, for example, when emptying a pellet stockpile or using sinter from the stockyard, high fine loadings cannot be avoided. Especially if the material is wet, proper screening is difficult or even impossible.

Fines tend to segregate. When material is put into the stockpile, the fine material remains on the point of impact and the coarser material rolls outwards, known as size segregation. This effect is known wherever granular material is handled. So, when reclaiming material from stock, it is important to avoid high amounts of fines being reclaimed and sent to the furnace without screening.

Similar segregation can take place while charging the furnace, and can impact the furnace process. Fines in general are undesirable due to the blocking of the spaces between the larger particles, but due to the flow characteristics of fines, they can also deposit preferentially in certain areas. The impact of this is particularly noticeable with bell-charged furnaces, where the fine particles will drop directly down onto the stockline, and the large particles will flow a little more outward and deposit closer to the wall (see Figure 11.1 on the next page). If material hits the wall before it reaches the burden level, the fines will accumulate close to the wall and the coarser material will flow more inwards. This segregation effect also takes place when filling a bunker. Be it in the stockhouse or on the bell-less top, segregation will always take place. When material is required from a bunker, it starts to deliver the material that has been charged in the center: those being the fine materials, while later the coarser materials from the sides of the hopper begin to flow.

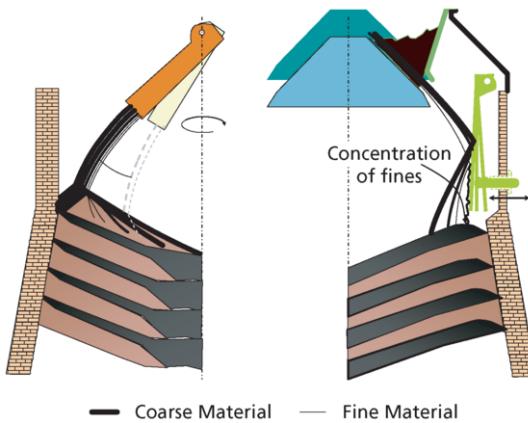


Figure 11.1 Segregation of fines during charging with bell-less and bell-based tops

A concentration of fines close to the wall can have a negative effect on the reduction and melting of the ore as it forms a blockage for the passage of hot reducing gasses. When the fines reach the cohesive zone and are melting, the remaining oxygen is removed by direct reduction. This results in a higher fuel rate or cooling of the furnace. If the material does not melt in due time before the tuyere zone is reached, the “non-molten” material is observable as scabs through the peepsights.

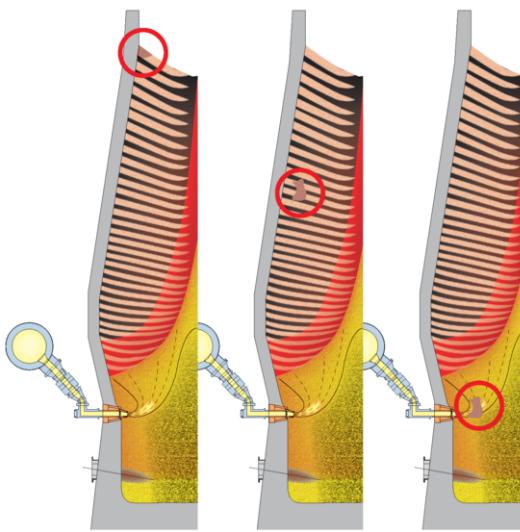


Figure 11.2 Fines charged at wall migrating through the furnace and appearing as “scabs” in front of tuyeres

Note that there is a difference between the paths travelled by the coarse materials and the fines. When the burden descends through the furnace, the fines fill the holes as soon as they are formed, while coarse materials follow

the wall. Fines travel more vertically and faster towards the cohesive zone (see Figure 11.2). With a bell-based top arrangement, it is possible to deflect the fines by using the furnace movable armor as a deflector, and with a bell-less top by charging from the outer position to the inner position.

An additional source of fines that can be avoided through slight modification in stockhouse practices is that of bin management. The bin fill levels may drop if there has been an interruption in the supply of materials. The bin level may have dropped so low as to require a stop of the blast furnace, or supply may have resumed in time for the blast furnace to remain in operation. In either case, the refilling of the bins should be managed to avoid excessive breakdown of the material as it falls a longer distance. It may be tempting to try to increase the bin levels at all of the bins by filling them all up gradually. However, this method will result in more material falling from a greater height as the bins will continue to be emptied as they are being refilled.

It is preferable to work with fewer bins, so that the levels can be increased more quickly, even though they are being emptied at the same time. Once the bin levels are up at 80 to 90 %, the excess feed material can be diverted to a low level bin, which remains out of service. The fuller bins are then topped off when they drop to their usual aim level of 65 to 70 %, giving some headroom for the supply to be diverted to the less full bins (Figure 11.3).

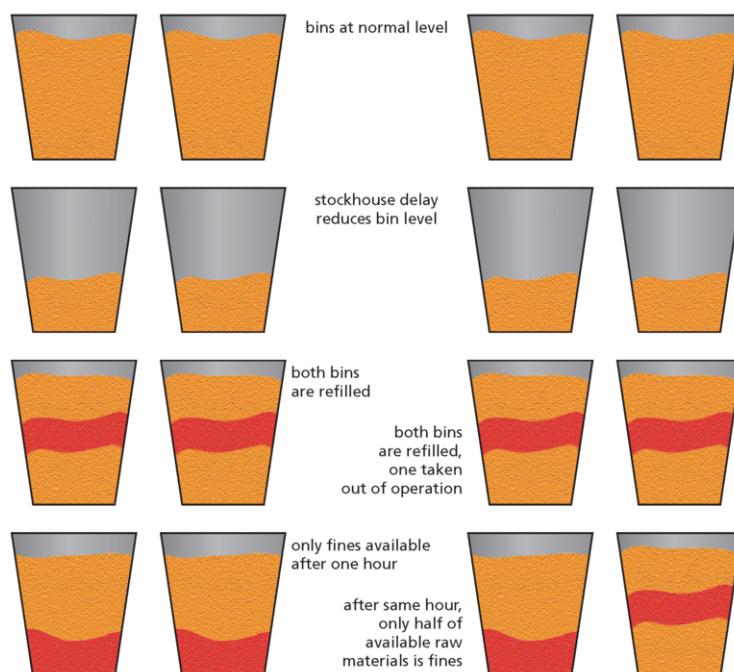


Figure 11.3 Effect of staggering restart on stockhouse bins. Red layers indicate raw materials containing more fines generated by impact.

Refilling the bins by this method will reduce the amount of material that has to drop from a greater height, as there will be less transfer time and more filling time, and the bins that are not being used while they are being filled will fill more quickly. Once all bins are back to normal fill levels, they should be brought into operation. This will result in each bin coming back into operation one at a time. This staggering of the restart of the bins will avoid the material that has been dropped the furthest distance in all bins being charged to the blast furnace at the same time.

11.1.2 Moisture in coke and burden

11.1.2.1 Moisture input

The moisture charged into the furnace with the coke and ore burden must be removed before the process can start. This takes place in the upper part of the furnace. The amount of water with the charge at 1 % burden moisture and 5 % coke moisture is about 33 kg of water. The center dries very quickly, but in the wall area it can take much longer, about 40 minutes (Figure 11.4).

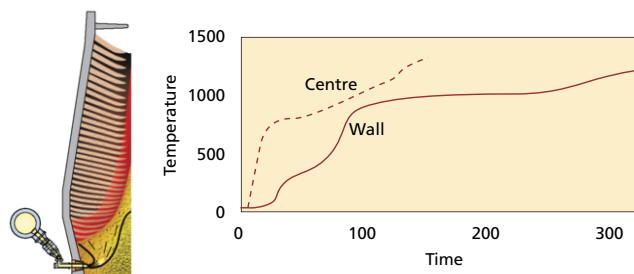


Figure 11.4 Temperature in furnace

If the moisture input increases, then it will take longer for the material to dry and the isotherm where the reduction process will start will descend. If the chemical reserve zone is shortened too much, the reduction process will become less efficient as a consequence. More oxygen will remain bound to iron and this oxygen has to be removed by direct reduction in the lower part of the furnace. This consumes energy and cools the furnace.

Most companies have moisture gauges for coke bins, so that variation of the moisture input in coke is compensated for with an additional weight of coke. Note that this is only a minimum correction to maintain the current thermal state. If the furnace is already operating with a low top temperature (below 100 °C), the compensation with coke moisture gauges will not be sufficient to compensate for the decreased efficiency of the reduction process.

Where moisture is added in place of coke, the furnace cools and the normal thermal control procedures will be activated, usually calling for additional fuel.

If the moisture level then reduces again, the furnace will warm up, triggering another set of actions. If this is allowed to continue, the furnace will enter a thermal cycle that will in turn consume more fuel than required. Extreme coke moisture variation typically occurs when the coke supply is switched from wharf coke to stockpiled coke that has absorbed more moisture due to rain or the storage field that is not well-drained. If coke moisture readings are not correct, it can cause large swings in the thermal state of the furnace.

This effect is just as important with pellet moisture, especially where pellets have been shipped or stored under damp conditions. They can contain up to 6 % water. When a batch of these pellets is charged to the furnace, the top temperature will decrease with the additional moisture, but the furnace will start to warm up since a lower amount of iron is being charged with the higher moisture pellets. Coke rate changes will normally be made to correct for this warm up. However, once this batch of wet pellets has been consumed, it is very important to realize that the furnace will then cool down due to the additional iron that is being charged with low moisture pellets. This can cool down the furnace very quickly, so it is better to anticipate this change by increasing coke rate when it is known that the wet pellets have been consumed and dry pellets are soon to arrive. Some of the larger blast furnaces also have pellet moisture measurement instruments installed on the stockhouse bins.

Ideally, coke and pellet moisture gauges can be installed to monitor and correct for any changes on-line. These moisture gauges take regular readings of the as-charged moisture levels for coke and pellets and will make corrections for the weight, so that the required quantity of the material is charged.

The recommended approach is that the top temperature is not allowed to fall below dew point temperature for a prolonged period (8–16 hours). Some companies are able to run the top gas temperature at low average levels, well below 100 °C. In these situations, it is recommended to monitor the temperatures in the wall area (3–5 m below the burden level) to determine whether or not the burden is dry ‘on time’.

11.1.2.2 Moisture condensation and recirculation

Since burden materials are charged well below the dew point of the moisture in the furnace gas, the moisture in the gas will condense on the colder particles. This does not only apply to the moisture which is charged (30–40 kg/tHM typically), but also to the moisture which is generated in the process. There is more water generated when there is more hydrogen produced from the injectants at the tuyeres. At a coal injection level of 200 kg/tHM and 5 % hydrogen in the coal, about 36 kg of water per tonne hot metal comes from the coal and has to be eliminated with the top gas. The amount of water generated from natural gas is even higher: at an injection level of 80 kg/tHM, about 65 kg of water comes from the natural gas. All the water has to be eliminated with the top gas.

If not all the water is eliminated, a condensation–evaporation cycle will start. Evaporation will take place more towards the center and more down in the furnace, while condensation takes place closer to the wall, as shown in Figure 11.5. In the process of this cycle, the isotherms where the reduction process starts will go lower and lower in the wall area of the furnace, making the efficiency of the process poor and creating a risk of tuyere failure and upsets.

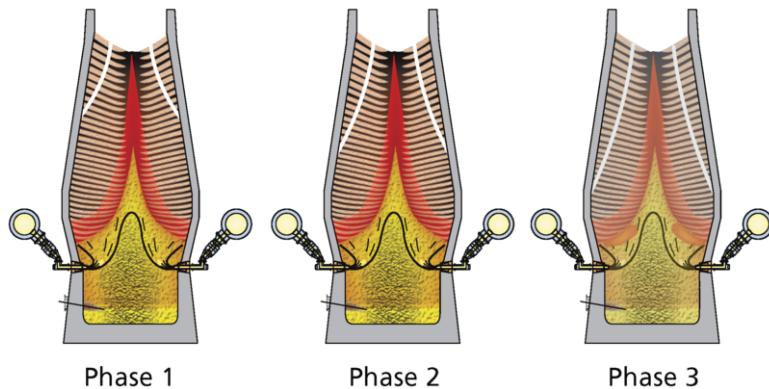


Figure 11.5 Effect of evaporation–condensation cycle of moisture, coming from charged materials and generated in the process. The gas flows from inside out reaching lower temperatures and water will condense. The 100 °C isotherm is indicated in white. The mechanism proceeds as follows:

Phase 1 – Normal operation

Phase 2 – When the water input is high, vapor condenses where the gas temperature is below 80 °C; this occurs in the wall area, especially on the freshly charged raw materials. The 100 °C isotherm is driven downwards.

Phase 3 – As the evaporation–condensation cycle continues, the 100 °C isotherm is driven further and further downwards. The metallurgically active area near the furnace wall becomes smaller and smaller, impeding gas reduction in the periphery. Direct reduction levels increase, the furnace becomes inefficient with low ETACO.

11.2 Burden descent

11.2.1 Variability of charging and production rate

Most operators observe the charging rate in a furnace as defined by the amount of charges put in the furnace per hour. If the charging rate increases, while tuyere conditions are unaltered, the furnace will fall short of heat. Simply put, with the same amount of heat and gas produced at the tuyeres, more hot metal is made, so the furnace will chill. The reasons for this happening can be various.

- fuel shortage as a consequence of increase of direct reduction.
- too low coke input, for instance by incorrect compensation.
- too much input of ferrous material (e.g. when changing from “wet” pellets to dry pellets).
- changed process conditions.

Here we refer to increased direct reduction. In some situations, the gas reduction of the burden does not progress sufficiently. This can have a number of causes.

- too much water input, lowering the isotherms within the furnace and shortening the process height of the furnace, especially at the wall.
- large amount of fines charged into the furnace resulting in poorer pre-reduction and increased direct reduction.
- irregular burden descent, causing mixed layers.
- high liquid residual level affecting the normal gas flow through the burden.
- charging delays causing the newly charged material to see shorter process height and altering burden distribution.

The resultant material with insufficient pre-reduction will in any case continue to descend to the high temperature region above the tuyeres. When this material starts melting, all oxygen will participate in direct reduction. This consumes coke and since coke consumption drives the production rate, the production rate will increase further. This is a self-propagating effect, and will chill the furnace within hours.

Experienced operators equipped with the right tools can observe the increased direct reduction long before the casthouse gives warning of low hot metal temperature. The method to correct the incident is by slowing down the production rate, with extra fuel injection and/or a lower blast volume, and by maximizing heat input into the furnace (maximum hot blast temperature and no blast moisture).

An example of the use of direct reduction as an early warning signal for a cooling trend in the furnace is shown in Figure 6.1. The earliest signal of a cooling trend comes from the increase of direct reduction followed some time later by an increase in charging rate. The hot metal silicon reacts a few hours later, because the colder material has to percolate through the hearth.

11.2.2 Hanging and slipping

The burden descent sometimes becomes erratic (see Figure 7.3). What happens in the furnace if it hangs and slips? The mechanism of hanging and slipping is illustrated in Figures 11.6 through 11.8 on the following pages.

First, the furnace may hang when near the cohesive zone, bridges of melting ore burden are formed. “Bridge formation” is the phenomenon where solid materials can be piled upon each other and will not collapse into a hole, see Figure 11.6 for a bridge formed from marbles.

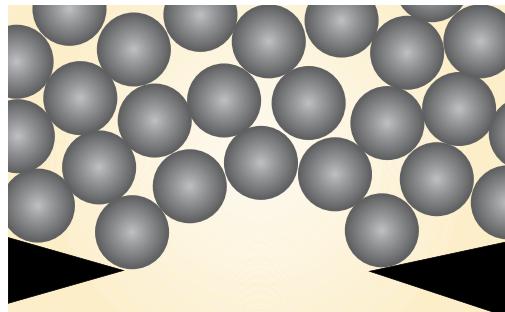


Figure 11.6 Bridge formation illustrated by a theoretical experiment with marbles

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

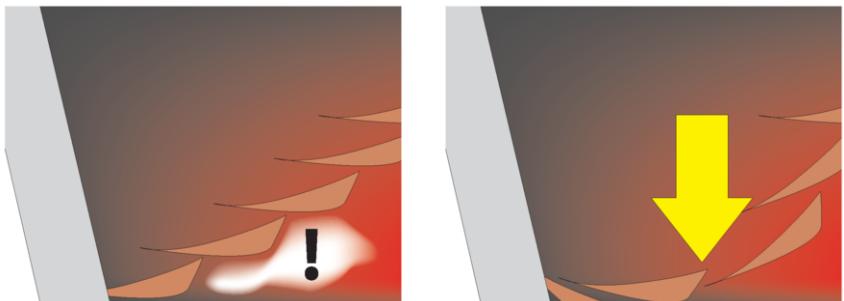


Figure 11.7 Creation of voidage below bridges and consequential collapse

Third, when this voidage becomes too big, it collapses: the furnace burden slips (Figure 11.7). The layer structure is completely disrupted and the gas flow through these layers is impeded. This in turn leads 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 a next hanging incident. 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 to prevent a continuous cycle of hanging and slipping.

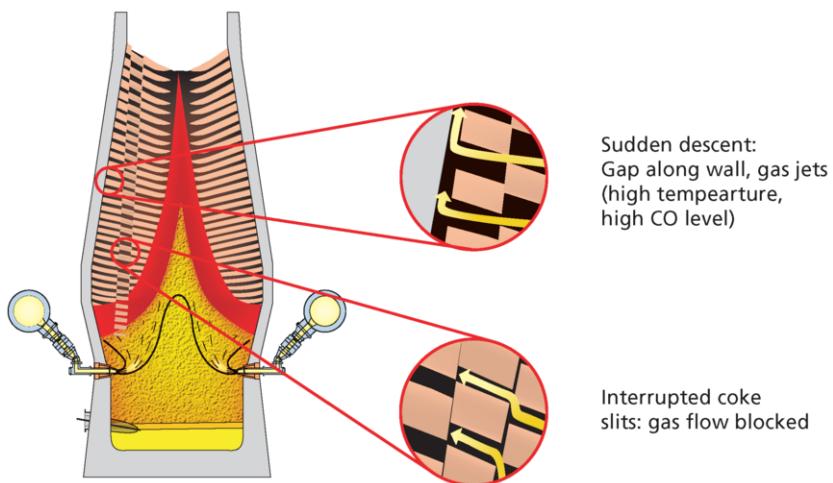


Figure 11.8 Disrupted layer structure and impeded gas flow

11.3 Recirculation of alkali and zinc

Potassium, sodium and zinc tend to re-circulate within the blast furnace. They form gaseous compounds, which condense on colder parts of the burden. These elements can have a negative impact on the refractory condition. Alkalies will affect the coke reactivity (Chapter III) and in doing so will increase direct reduction reactions.

In furnaces operated with a central gas flow, the top gas temperatures in the center increase to such a level that part of the alkali and all the zinc leaves the furnace as a vapor with the top gas. If top gas temperatures are low, the alkalies and zinc may accumulate in the furnace. The zinc normally condenses on the refractory.

Alkali build-up is manifest by observing the potassium content in the slag, especially when the slag is acid and/or the furnace is cold. Alkali leaves the furnace easier with a low basicity (B2 below 0.9) slag and at low hot metal temperatures. One rule of thumb is that as long as the K₂O content in a lean or cold cast is below 1 %, there is no significant accumulation. It is also observed how fast the potassium in the slag returns to a normal level, when slag is lean, such as when preparing for a stop.

11.3.1 Alkali in the blast furnace

Alkali, which are sodium and potassium, enter the blast furnace with the ferrous burden and coke and coal. The alkali takes part in a cycle of reduction-oxidation as well as vaporization-condensation. The alkali input in a blast furnace is typically below 1,5–5 kg per tonne hot metal. In local situations, where companies have access to local ores with relatively high alkali, the input

can reach higher levels. In this situation, high productivity is normally no longer possible, but the competitive edge is gained from the cost advantages of the use of cheap, local ore. From measurements of chilled blast furnaces it has been observed, that the amount of recirculation is about 3–10 times the input.

The alkalis in the blast furnace have various effects on operation. From a chemical point of view, alkali acts as a catalyst for the solution loss reaction ($C + CO_2 \rightarrow 2 CO$). It means that at high alkali input, the fuel rate is slightly higher. This effect is generally rather small, reported values are 6–11 kg coke per kg alkali in the burden. The additional fuel can be added by increased PCI and leads to more top gas energy from the furnace.

A second effect is, that degradation of coke and ferrous materials is promoted, which leads to a poorer permeability and can affect productivity at high production rates.

Moreover, alkali can cause the formation of scaffolds in the furnace: solid material adhered to the wall in the stack of the furnace. The effect of the formation of scaffolds is that the burden descent deteriorates, in extreme cases leading to hanging and slipping. Also in scabs, formed from cohesive material in the bosh/belly area, high levels of alkali have been observed.

Finally, the refractory materials, especially carbon-based refractories, can be attacked by alkali, affecting campaign length. Therefore, alkali control in the furnace and prevention of scaffold formation require the attention of the blast furnace operator. The blast furnaces operators should monitor alkali and zinc loading and removal to have an idea of recirculation of these undesirable elements inside the furnace or calculate removal percentages.

Sodium and potassium

The two types of alkali, sodium and potassium, behave differently. Sodium is more easily removed from the furnace with slag as well as via the top gas. Moreover, in most furnaces the potassium input is higher than the sodium input. Therefore, for our purposes it is sufficient to observe the potassium balance. Only in situations where the sodium input is higher than the potassium input, both have to be observed. The alkali mainly leave the furnace via the slag. Some of the alkali leaves the furnace via the top gas and top gas dust.

The recirculation of alkali in the blast furnace is shown in Figure 11.9. Alkali enters the furnace as silicates, are first dissolved in the primary “melt” formed in the cohesive zone. This melt has a very high level of FeO. As soon as the iron oxides have been reduced out of the primary slag, the final slag is being formed. As indicated below, the alkali can be reduced to elemental sodium and potassium, which do not dissolve in the hot metal nor in the slag. Sodium and potassium are gaseous and ascend with the gas in the furnace. In the high temperature area (around 1300 °C), cyanides are formed, which bind with the

alkali and ascend with the gas. As soon as the oxygen potential becomes higher, the cyanides are no longer stable and disintegrate to alkali oxide and alkali carbonate (around 1100 °C). The material is absorbed into the solid material (coke and ore burden) and starts descending again with the melting ore burden and carbonates will dissociate in CO₂ and alkali oxide again. The alkalis recirculate in the area from where the ore starts melting (around 1100 °C) to where the slag is free of FeO (around 1400–1450 °C).

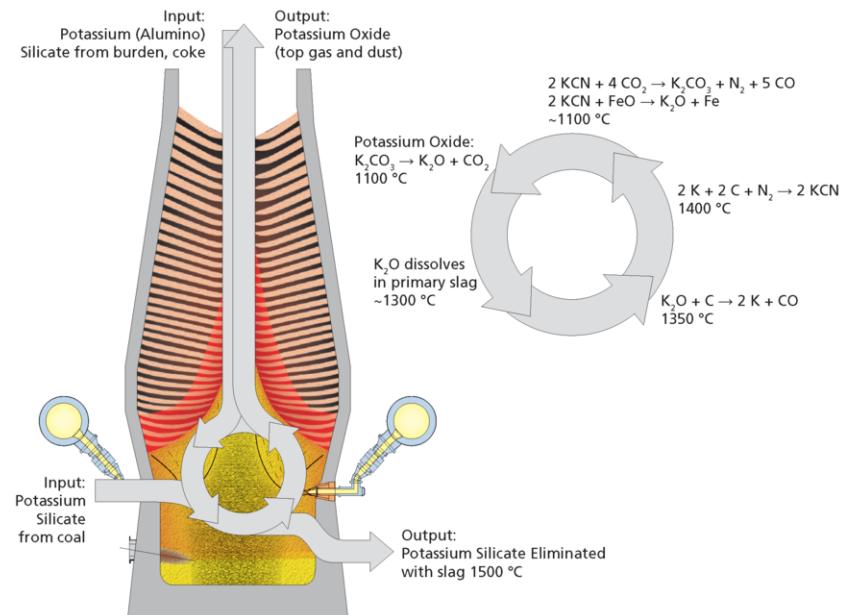


Figure 11.9 Recirculation of potassium inside the furnace. A similar cycle applies to sodium. Temperatures are author's estimates.

The retention of alkali by slag is influenced by the slag basicity, slag composition and the temperature of hot metal and slag. The lower the basicity, the more easily the basic K₂O and Na₂O are removed. The lower the temperature, the less the FeO is removed from the slag phase and the less alkali are reduced. In operational practice, this has interesting consequences.

If a blast furnace is stopped, the slag basicity is normally decreased with 10 to 20 % in order to restart the furnace with a slag with lower liquidus temperatures. After the restart of a furnace, the slag basicity is often low and the temperature is low. So, through analyzing the K₂O in slag before and after a stop, the operator can get an impression whether or not a high amount of alkalis is circulating in the furnace. The slag composition, especially the MgO content and MnO content, also affects alkali removal capacity.

Some alkali is removed with top gas. The top gas temperature varies over the diameter of the furnace throat. Furnaces operating with a central gas flow have high temperatures in the centre (average over 600 °C). The alkali is removed with the central gas flow. It has been generally accepted that operation at high productivity and low coke rate (high PCI rate) requires a central gas flow. This facilitates elimination of alkali.

11.3.2 Zinc

Zinc comes to the blast furnace via the sinter, mainly through recycling of blast furnace top dust, blast furnace sludge and dust from the secondary dedusting. Interestingly, a relatively large part of the final zinc input comes indirectly from the type of scrap the steel plant is using. Especially, scrap from galvanized steel. Zinc enters the furnace as an oxide, silicate or ferrite. It is reduced in the lower part of the furnace to elemental zinc and since it has low melting (419 °C) and boiling points (907 °C), it ascends through the furnace as a vapor and is then reoxidized, condenses to burden and coke and participates in a similar recirculation pattern as potassium. Zinc is eliminated from the furnace with the dust from the top gas, especially from the gas at the furnace centre. Then zinc adheres to the surface of particles. As a consequence, the finer the particles, the higher the zinc percentage. Therefore, zinc can be recaptured by segregating the fine fractions in the wet sludge. The fines can be upgraded to higher zinc concentrations in a specialized blast furnace.

Zinc–oxide can condense on the furnace walls and on/in the coke lumps. Zinc can form scaffolds in the furnace, thus leading to deteriorating operational results. Occasionally, the scaffolds are melting when a tuyere is changed and liquid metal (zinc) drops from above. When zinc is deposited in the pores of the ferrous materials, it can impede the gas reduction, leading to higher reductant rates. Zinc also deposits in carbon–based refractories, causing them to become brittle and crack, which reduces lining life. Zinc control is mostly done by controlling input levels. Typically 150–400 g/tHM is used, but higher values for specialized operations are also known.

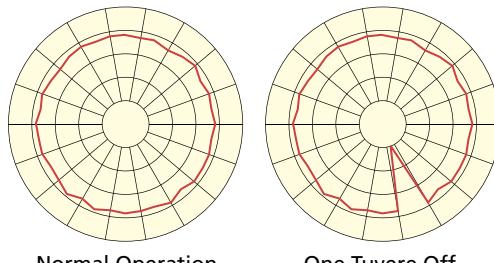
11.4 Circumferential symmetry

For high performance of the furnace, all tuyeres have to contribute equally to the process. We can consider a tuyere as representing a section of the furnace as shown in Fig 11.10. If one part of the furnace does not produce the same amount as other parts, the total will suffer. Operators can observe asymmetry in the process from various parameters. For instance from:

- the burden level – one stock rod at the top gives a different burden level from another.
- differences in gas temperatures in the uptakes.
- asymmetry in the heat losses.
- differences in hot metal temperature between two tapholes.

11.4.1 Circumferential symmetry of injection and wind distribution

If every tuyere in a blast furnace is considered as part of the blast furnace “pie” and is responsible for the process to the stockline, it is self-evident that the circumferential symmetry of the process has to be assured to reach a good, high performance. This means: symmetric wind distribution and symmetric injection of coal and gas. The various systems in use for PCI have different methods to ensure a good distribution.



PCI at all tuyeres	PCI at one tuyere off
• Coke Rate 300 kg/t	• Burns at tuyeres 3000 kg/hr
• PCI 200 (RR = 0,85) 170 kg/t	• All coke
• Total 470 kg/t	Production increase at this tuyere without
• Production 10 t/hr	PCI of 3000/1300 = 230 %!
• Carbon balance:	
• Coke 3000 kg/hr	
• Coal (in SRE) 1700 kg/hr	
• Total 4700 kg/hr	
• Iron carbonization -500 kg/hr	
• To direct reduction -1200 kg/hr	
• Burns at tuyeres 3000 kg/hr	
• Of which Coal 1700 kg/hr	
• and Coke 1300 kg/hr	

Figure 11.10 Schematic presentation of the effect of no PCI on one tuyere.

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 increases substantially (Figure 11.10). Consequently, 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 equalizing 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.

The wind distribution is equally important. Operators tend to work with smaller tuyeres above the tapholes, since a lower production rate above the tapholes stabilizes the taphole mushroom and taphole length. When applying smaller tuyeres, the injectants should be maintained on the tuyeres, otherwise

the production increases, even with smaller tuyeres. When replacing a 145 mm tuyere with a 125 mm tuyere, the wind volume decreases to 74 %. A similar way of reasoning applies when a tuyere is partially blocked, by slag or ash deposits.

11.4.2 Circumferential symmetry and direct reduction.

A furnace can be divided into sectors in which every tuyere forms one sector. If all sectors do not contribute equally to the process, asymmetry in the cohesive zone will arise, as shown in Figure 11.12. 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.

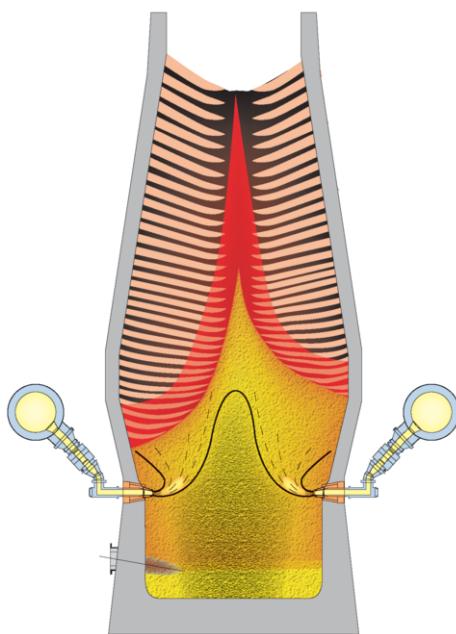


Figure 11.11 Asymmetric cohesive zone

Asymmetry in the process can arise from various sources:

- 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 that of the blast furnace process i.e. more frequent than every six hours.
- Blast distribution. Blast distribution is effected by plugged tuyeres (above a taphole or refractory hot spots) and slag deposits in the tuyere. Asymmetry is also the result of too low blast speeds (below 100 m/s): tuyeres will not function as blast distributors efficiently. This can be observed especially at the tuyeres opposite the inlet between hot blast main and bustle main.

- Worn refractory or throat armor plates at the top of the furnace.
- Uneven coal injection. Especially tuyeres without PCI.
- Deviation of furnace center 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.

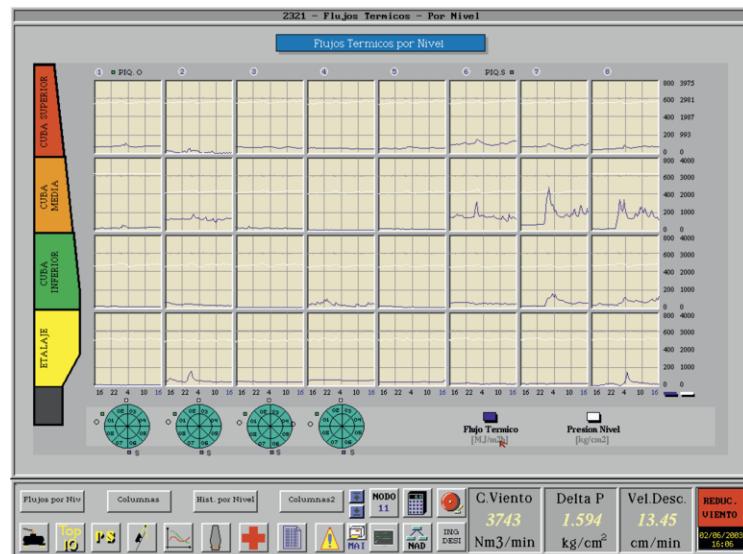


Figure 11.12 24 hours heat loss distribution (blue). Note a slight process asymmetry. One day graph of eight sections, four levels.

With coal injection, it is very important that the tuyeres are clear and open, allowing the coal plume to flow into the raceway to optimize combustion. If the tuyere should become blocked, or a blockage in front of the tuyere appears, then the coal must be removed immediately. If it is not, then the coal will be forced backwards into the tuyere stock and can ignite further up in the connection with the bustle pipe (see Figure 11.13 on the next page). This can cause serious damage or even explosions. It happens especially within an hour after restart of a furnace, since unmolten material blocks the tuyere gas flow. The phenomenon has also been observed with natural gas injection. In many cases, this has been a cause of a chilled hearth since the furnace is shut down in an unprepared state for extended periods to replace tuyere stock or repair the bustle main.

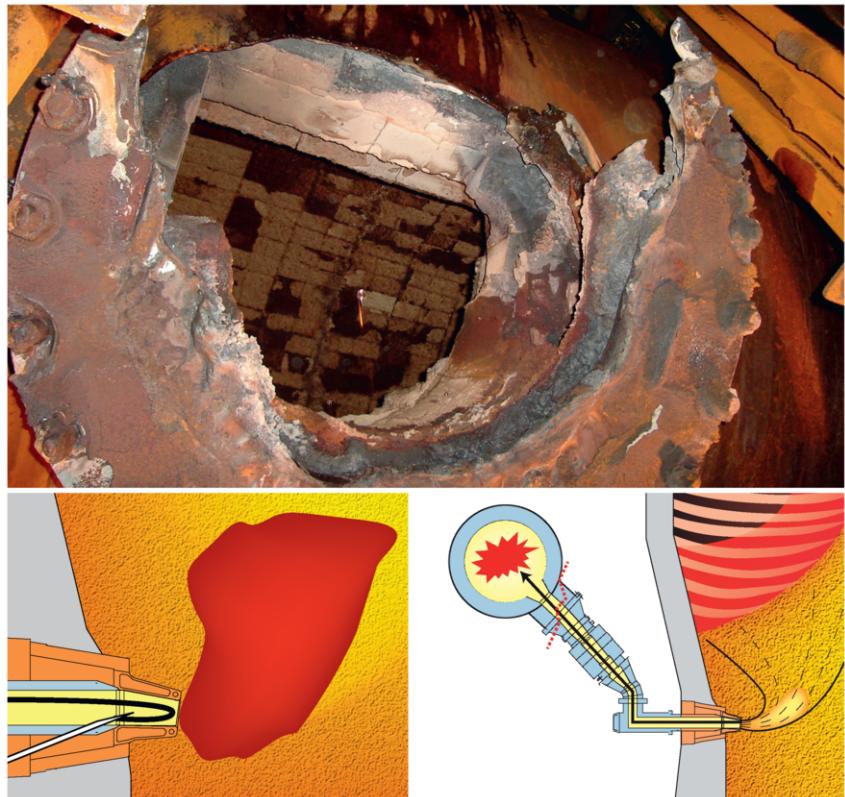


Figure 11.13 Coal backing up into the bustle pipe, caused by scab in front of tuyere, leading to possibility for explosion.

To prevent this, a light sensor may be fitted in front of the peep sight to detect a blockage at the end of the tuyere, or the pressure drop over the tuyere can be measured to detect when flow has stopped, indicating that a blockage is present. The coal to that tuyere is automatically switched off and restarted only once an operator has checked to see if the tuyere can accept coal. Many furnaces using natural gas injection rates over 90 kg/tHM are also installing the Δp blocked tuyere detection to shut off the natural gas to prevent back-ups.

11.5 Tuyeres

The tuyeres are a critical part of furnace equipment, but what can be observed when walking around the tuyere platform? What can happen to the tuyeres? There are two types of tuyeres – single and double chamber. The single chamber tuyeres have one cooling channel, whereas the double chamber tuyeres have a separate nose cooling pipe and a body cooling pipe. The advantage of the double chamber tuyeres is that the body can continue to be cooled after the nose has been burnt, and changing the tuyere can wait until the next planned furnace

stop. If a body cooling circuit in a two chamber tuyere or the water circuit in a single chamber tuyere is burned, then a furnace stop is required immediately to change the tuyere to prevent large quantities of water from entering the furnace hearth, resulting in furnace cooling. In order to learn about the condition of the furnace by looking through the peepsights, the guidance of an experienced furnace operator is needed.

11.5.1 Water discipline

A duller light in front of the tuyere can be an indication that water is present at or around the tuyere. This water could be coming from the tuyere itself, the tuyere next to it, or a cooler in the region above the tuyere.

Other signs to look out for when water is suspected are:

- Water visible under tuyeres, at grouting nipples or at thermocouple sockets.
- Top gas hydrogen increase. However, when hydrogen in the top gas indicates water leakage, there is a very large water flow going into the furnace.
- Smell of ammonia.
- Unusual thermocouple activity.
- Increase of make-up water frequency in closed cooling systems.
- CO gas detected at outlet of open water system.
- Cooling furnace trend, less reaction to additional fuel than expected.
- Water flame at taphole.
- Short or spitting tapholes.

When water is suspected, it is very important to locate and isolate the source as quickly as possible to limit the quantity of water that is entering the furnace. Depending on the location of the leak, a furnace stop may be required to find and isolate the leak.

Water inside the furnace is extremely damaging to both the process and the equipment and the consequences of allowing leakage to continue are extremely serious for both. The major consequences are:

- Attack of refractories in the hearth and/or the taphole.
- Local heat shortage in the process leading to inefficient process, scabs, more burnt tuyeres and an increased requirement for reductant/fuel.

11.5.2 Burnt tuyeres

The ideal scenario is to change tuyeres because the working lifetime is reached, or will be passed before the next furnace stop. This preventative changing of tuyeres is usually done with an aim lifetime of two years for double chambered tuyeres. However, many companies are required to change their tuyeres much more frequently than this, with a tuyere lifetime of less than six months considered to be poor. Other than preventative changing, the most frequent reason for changing tuyeres is due to burning (Figure 11.14). As the tuyere coolers are made of copper, which has a melting temperature of 1083 °C, the

water cooling is absolutely essential in protecting the tuyeres, and even then they may not be protected against a direct 'strike' with hot iron. It is therefore generally accepted that the tuyeres must also be protected by a protective coating of slag on the upper surface during operation.

Burnt tuyeres may occur due to a number of different mechanisms, these being:

- Hot metal attack by a low position of the root of the cohesive zone. In order to prevent this, sometimes a protective layer is made on the upper part of the tuyere.
- High hearth liquid levels.
- Tuyere condition: inadequate cooling or casting defects during production of the tuyeres.

Sometimes, tuyeres are damaged and start leaking if the coal lance is not maintained in the center of the blast flow and coal is allowed to cause erosion of the copper all the way through to the cooling channel.



Figure 11.14 Tuyere failure examples

11.5.3 Tipped tuyeres

Tipped or dipped tuyeres are identifiable by looking in the peepsight to see an oval shape rather than a round orifice. There may also be gas blowing from the tuyere to cooler seat. In the more extreme cases, the outlet of the tuyere will be completely obscured. This is thought to occur when impact from material above has impacted on the tuyere, forcing it downwards (Figure 11.15). It is also an indication of little support being in place below the tuyere to prevent the downward movement. This can either be due to the design or wear in the tuyere band.

The major cause of tipping is that there is a big scab of solid burden material resting on the tuyere, which is normally caused by a too low coke percentage at the wall in the throat.



Figure 11.15 Tuyere tipping

11.5.4 Scabs in front of tuyeres

It is sometimes reported by the blast furnace operator that scabs are seen dropping in front of tuyeres. This has long been accounted for as scabs peeling off from further up the furnace, which may well have been true in the days of high alkali input, but this explanation is less acceptable now that burden quality has improved. A more likely explanation is that it is non-molten burden, which was charged a few hours earlier. There can be several reasons why the burden did not melt in time: late warming up (caused by low burden level at the time of charging, or by poor gas permeability caused by high amounts of fines or by high moisture content), a very cold furnace (low root of cohesive zone, leading to a position of the cohesive zone almost on top of the tuyeres), too fast coke usage (when a tuyere has no injectant).

11.5.5 Blowpipe failures

With coal injection, it is very important that the tuyeres are clear and open, allowing the coal plume to flow into the raceway. If the tuyere should become blocked, or a blockage in front of the tuyere appears, then the coal must be removed immediately. If it is not, then the coal will be forced backwards into the tuyere stock and can ignite further up in the connection with the bustle pipe (see Figure 11.12). This can cause serious damage (Figure 11.16 on the next page).

The blowpipe can also fail as soon as liquid iron enters it or when the coal lance breaks. The damage is caused by the hot gas and coke escaping through a hole in the blowpipe mantle. These types of situations may cause extreme collateral damage outside the furnace on the tuyere platform such as melting wiring, piping and instrumentation.



Figure 11.16 Failed blowpipe

11.6 Stops and starts

11.6.1 Internal state of furnace

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 1000 °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 once again molten (Figure 11.17).

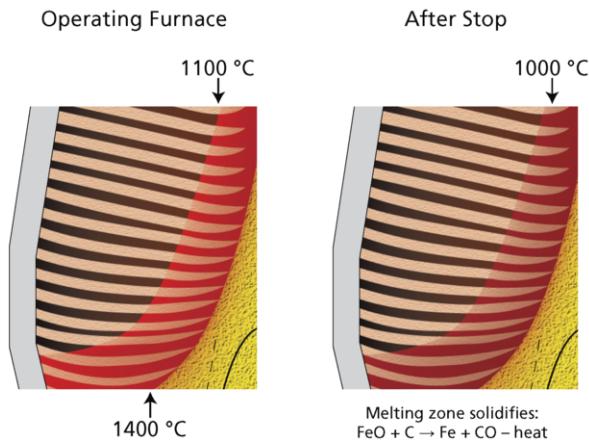


Figure 11.17 Solidified cohesive zone as consequence of a stop

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/fuel 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 9.12, 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 extra coke into the furnace 6 to 8 hours prior to the stop.

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 to compensate for these heat losses. 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 to reduce the slag basicity.

Even if a stop is unplanned, taking these measures after the stop is worthwhile, since the cohesive zone will be lifted as the extra coke is being processed.

For a blow-in or start-up 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.

11.6.2 Stopping a blast furnace

Stopping a blast furnace means that the hot blast is removed from the furnace. Since the furnace is connected in a gas flow from stoves via the furnace to the gas cleaning system, a stepwise approach is required. The objective of the stepwise approach is that there is no chance that an explosive gas-air mixture is formed nor that slag enters into the tuyeres. To this end, the blast furnace has to be kept on slight overpressure during the stopping procedure.

- Step 1: Furnace stopping procedure starts when the furnace hearth is empty. Then blast volume is reduced, a check may be carried out and the blast pressure is reduced to a low level of 30 kPa.

- Step 2: Top gas bleeders are opened and the furnace is disconnected from the gas cleaning system via a goggle valve or water seal. In this situation, all blast furnace gas goes to the atmosphere. The furnace still has at least one taphole open.
- Step 3: Blast volume is reduced to zero, and the backdraft stack is opened in order to burn gas from the furnace. The tuyeres may be clayed in order to prevent air ingress and consequential coke burning during the stop. Some backdoors or peepsites are open to allow for some air ingress into the bustle main.

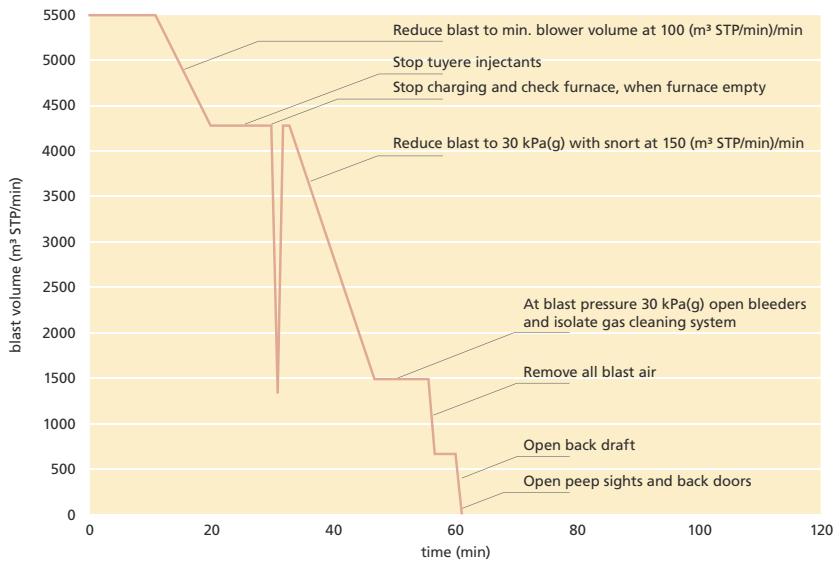


Figure 11.18 Guideline shutdown procedure

11.6.3 Start with all tuyeres

After a stop of generally shorter than 5 to 7 days, the blast furnace can start with all tuyeres. During a blow-in the reverse process takes place.

- Step 1: Furnace is put on low blast pressure.
- Step 2: Furnace is connected to gas grid and bleeders are closed.
- Step 3: Hot blast volume and pressure go through stepwise increase.

Note that it takes time for the burden to start descending after a stop and that coal injection should only be restarted after descent has begun.

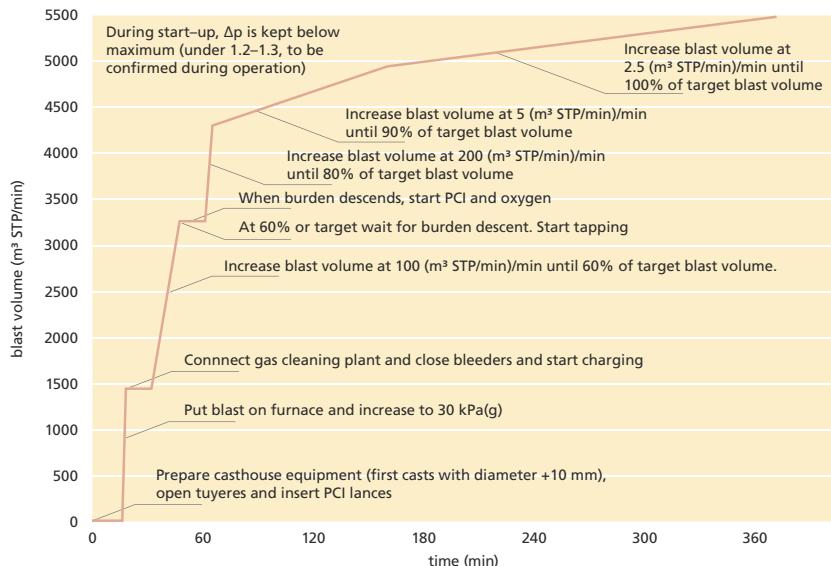


Figure 11.19 Guideline for blow-in with all tuyeres

11.6.4 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 heat of the shaft gas is not transferred to the cold charge, the off-gas temperature increases and the gas composition changes. Since the equipment has not been designed to withstand the high top gas temperatures, these 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 water atomizing 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 CO_2 percentage decreases and CO percentage increases (Figure 11.20, next page).

Moreover, H_2 content generally 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 CO_2 formed at the tuyeres has insufficient opportunity to be transformed to CO and the CO_2 percentage in the top gas increases. As soon as half of the oxygen is in CO_2 (i.e. when the CO_2 percentage equals half the CO percentage), the furnace should be isolated from the gas system.

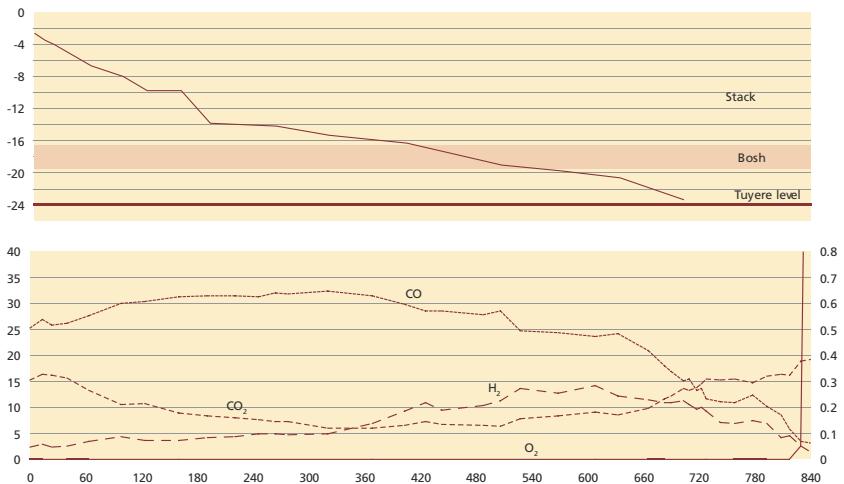


Figure 11.20 Typical progress of a blow-down

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 deadman, and alternating layers of coke and ore in the melting zone and stack zone. Since during the blow-down, the coke of the active coke zone and deadman 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 CO_2 percentage starts to decrease to below 10 %, then there is little iron oxide left to reduce.

The burden level in the furnace is difficult to measure with standard stock rods. Mechanical stock rods have to be equipped with chain or cable 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:

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

Another indication that the blow-down is complete is the collapse of the raceway (no more coke rotating) and detection of oxygen in the top gas, which is caused by some of the blast going directly to the furnace top since there is no longer a full quantity of coke 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, increasing hot metal manganese, increase of central coke charging and a burden without titanium addition. The full effect of these measures is uncertain, but hearth bottom thermocouples can indicate that there is more activity and less of a skull in the hearth.

11.6.5 Blow-in from new or after reline

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.

Heating up the hearth

Heat is generated by the reaction of coke carbon to CO. Coke generates 111 kJ per gmol of carbon, when reacting to CO, which corresponds to 8.1 GJ/t (Peacey & Davenport, p. 32).

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

- Heat coke in the hearth, deadman 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 furnace wall.

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

- Moisture in the coke can be neglected.
- The heat required for filling the hearth, deadman 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/deadman/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, deadman and active coke zone with coke, a coke blank is required with a total weight of 600 tonne: 300 tonne to fill hearth, deadman and active coke zone with coke and 300 tonne for the generation of heat to bring the coke to 1500 °C. Many operators have learned from past experience that a coke blank ranging from 50 to 65 % of the working volume or the coke blank elevation of 11 meters above the tuyere fulfils these heat requirements.
- In the early stages of a blow-in, the blast temperature should be maximized 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.

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 utilization.
- The direct reduction, as manifest from CO and CO₂ percentages exceeding “normal” values.

The gas utilization 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.

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 35 tonne of a high Al₂O₃ slag during the first 8 hours. 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.

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 the hot metal temperature reaches 1400 °C and the 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 530 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 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 a rapid blow-in of a furnace is presented in Figure 11.21. At the blow-in, the furnace was started up with four tuyeres (of 25). After opening all tuyeres, a “heavy” burden (coke rate 440 kg/tHM) was put in the furnace 30

hours after the blow-in and coal injection was put on the furnace 40 hours after the blow-in. Hot metal silicon reached the 1.1 % mark after 44 hours.

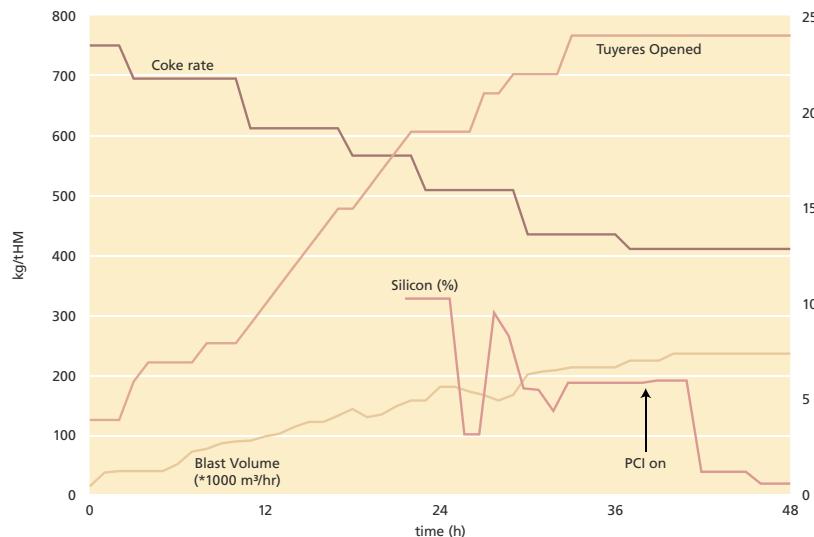


Figure 11.21 Blow-in of a blast furnace in November 2013.

11.7 Casthouse challenges

11.7.1 Chills, loss of connection

A chilled hearth is a situation where the connection between tuyere and taphole is lost. Liquids generated above the tuyeres can no longer drain to the hearth and in most cases, all tuyeres are filled with slag/melting materials. Sometimes, a catastrophic failure of a blowpipe or tuyere stock was the reason to finally stop the furnace. The furnace is filled with (semi-)solid materials well above tuyere level (Figure 11.22).

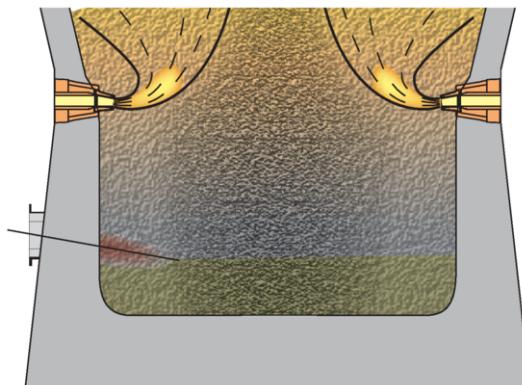


Figure 11.22 Loss of connection of liquid flow to taphole

What are possible causes?

- Poor burden descent/slips, hanging
 - Gunnite or shotcrete falling off the walls.
 - Scabs falling off the walls.
- Gas reduction of burden deteriorated.
 - As a consequence of disturbed gas flow.
 - As a consequence of casting delays, especially if the connection is lost after long stops.
- High water input in lower part of furnace.
 - Burned tuyeres.
 - Leaking tuyeres, staves, cooling plates, hot blast valves.
 - Water input with burden materials.
 - Top sprays leaking through the shut-off valves.
- Increased heat loss.
 - Burden distribution
 - Loss of refractory protection of staves.
- Casting problems.
- Burden quality.
- Improper charging.
- Burden materials or burden distribution.
- Loss of fuel injection.

How to recover?

Since the liquids in the furnace have been frozen, the following line of action has to take place:

1. heat has to be brought into the furnace.
2. the material in the furnace has to melt as easily as possible.
3. the molten material has to be taken from the furnace.

So the recovery principle is:

- bring maximum heat into the furnace: note that the tuyeres are the motor of the melting process.
- make slag as liquid as possible: very lean, typical basicity (B3) of 0,8–0,9.
- cast and clean to drain liquids from the furnace.

- Step 1: is re-establishing the connection between tuyere and taphole. This can be done with or without oxy-fuel lances. The connection is re-established by starting at a few tuyeres above one taphole. We recommend at least three tuyeres, since the tuyeres on the outside lose a lot of heat to laterally to the adjacent tuyere areas that are not working,
- Step 2: is to bring sufficient fuel to the tuyeres, so that the liquids draining to the hearth are hot. This can be done by charging coke blanks. Note that a fuel rate that is too high only leads to burning coke but no hot metal reaching the hearth.
- Step 3: is normalizing the process by placing more and more tuyeres in operation and normalizing fuel rate and process settings.

11.7.2 Coke mess

Coke mess is a situation, where a lot of coke is coming out of the taphole. This can happen when the taphole allows the coke through: it is normally rather short and wide, which can be caused by the fact that a taphole was out of operation for a long time, by poor quality of the clay or by water leakage. Also, the coke quality plays a role: the poorer the coke quality, the easier coke mess can result. The operational reaction is to lower the wind volume, plug the taphole as soon as possible, and start carefully anew. If water leakage plays a role, the problem has to be addressed first before the next cast.

11.8 Greenhouse gas emissions

The amount of carbon we bring into the furnace for normal operation is the carbon present in 300 kg/tHM coke and 200 kg/tHM coal (Table 11.1). If we assume that this carbon is emitted as CO₂, then the emission of CO₂ is 1544 kg/tHM. This is a simplification of reality, since carbon is used for sintering, cokemaking, blast generation and the electric power used for the blast furnace operation nor credits for top gas are taken into account. The European Commission has made a study of CO₂ emissions in the steel industry (Pardo et al, 2012). This study comes to a total CO₂ emission of 1279 t CO₂/tHM.

Since a blast furnace is operated close to the thermodynamic limitations, greenhouse gas emission reductions coming from increased efficiency will be relatively small. Table 11.1 gives the example of replacing 54 kg coal injection with 50 kg gas injection, which reduces the CO₂ emission with about 2%. However, the “best” way to reduce greenhouse gas emission from the blast furnace–BOF route is to use high coal injection rates and to make use of top gas efficiently.

		Scenario		
		Carbon content (%)	Coal injection (kg/tHM)	Co-injection (kg/tHM)
coke		87.0	300	300
coal		80.0	200	146
natural gas		72.0	0	50
Unit				
carbon input	kg C/tHM		421	414
CO ₂ emission	kg CO ₂ /tHM		1,544	1,517

Table 11.1 CO₂ footprint assuming that all carbon input is converted to CO₂

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Annex I *Glossary*

Angle of repose

The natural angle that is formed when material is discharged onto a pile.

Apatite

A group of phosphate minerals $\text{Ca}_5(\text{PO}_4)_3(\text{OH}, \text{F}, \text{Cl})$.

Banded Iron Formation (BIF)

A sedimentary mineral deposit consisting of alternate silica-rich (chert or quartz) and iron rich layers formed 2.5–3.5 billion years ago; the major source of iron ore.

Bentonite

An absorbent aluminum silicate clay formed from volcanic ash and used in various adhesives, cements, and ceramic fillers.

Calcium ferrite

Crystal of CaO and Fe_2O_3 .

Chert

A hard, dense sedimentary rock composed of fine-grained silica (SiO_2).

CO₂ Foot Print

The total amount of CO₂ emitted per ton of product over the whole route and taking all energy requirements into account.

Decrepitation

Breaking up of mineral substances when exposed to heat.

Dolomite

Material consisting of lime and magnesium carbonates; extensively used for adjusting the slag composition directly into the blast furnace or via sinter.

Fayalite

Compound of iron silicate: $2\text{FeO} \cdot \text{SiO}_2$.

Harmonic Mean Size (HMS)

The harmonic mean is the number of values divided by the sum of the reciprocals of the values. This gives a truer average value where ranges of values are used as it tends to mitigate the effect of large outliers from the total data set.

Hematite

Iron oxide in the form of Fe_2O_3 .

Magnetite

Iron oxide in the form of Fe_3O_4 .

Mill scale

The scale removed in a hot strip mill from the steel slab, mainly iron oxide.

Olivine

A mineral silicate of iron and magnesium, principally $2\text{MgO} \cdot \text{SiO}_2$, found in igneous and metamorphic rocks and used as a structural material in refractories and in cements.

Serpentine

Any of a group of greenish, brownish, or spotted minerals, $\text{Mg}_3\text{Si}_2\text{O}_5(\text{OH})_4$, used as a source of magnesium and asbestos. Generally a blend of olivine and fayalite with various impurities.

Spinel

Mineral composed of magnesium aluminate.

Wustite

Iron oxide in the form of FeO , does not occur in nature; produced during reduction process.

Annex II *Further reading*

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Annex III *Starting point for calculation examples*

The calculations and examples in the book are intended for illustration of the points made. Although all numbers and values are reasonable estimates, it is not necessarily the most accurate number available. Moreover, local conditions can be different. For example, the ore weight required to produce 1 tonne hot metal varies with slag volume per tonne.

Blast Furnace	Unit		Chemical parameter	Unit	
Hearth diameter	m	14	C	g/mol	12
Throat diameter	m	10	H	g/mol	1
Working volume	m ³	3,800	O	g/mol	16
Inner volume	m ³	4,400	N	g/mol	14
Tuyeres		36	Fe	g/mol	55.85
Production	tHM/d	10,000	1 kmol gas	m ³ STP	22.4
Voidage in shaft	%	30			
Voidage in hearth	%	15			

	Input kg/tHM	Spec. wt. kg/m ³	Chemical Composition (wt%)						
			ash	C	H	O	N	S	
Coke	300	470	10.0	87.0	0.2		1.3	0.9	
Coal	200	450	5.0	80.0	5.0	8.0	1.3	0.7	
Coal (co-injected)	150			80.0	5.0	8.0	1.3	0.7	
Gas (co-injected)	50			72.0	23.0	2.0	3.0	0.0	
Burden	1600	1.800							

		Spec. wt. kg/m ³	Chemical Composition (wt%)						
			C	CaO	MgO	SiO ₂	Al ₂ O ₃	B2	B3
Slag	220 kg/tHM	2300		51	10	45	10	1.13	1.36
Hot metal	945 kg Fe/tHM	7200	4.5						
1 charge	96 t burden								
1 charge	60 tHM								
wind	6,500 m ³ STP/ min								
oxygen	26 % in hot blast			Composition (vol%)					
hot blast temp.	1250 °C			CO	CO ₂	H ₂	N ₂		
top gas (dry)		1.35		24.3	23.1	5.5	47.1		

Annex IV *Rules of thumb*

Rules of thumb for daily operation of the blast furnace process (constant blast volume).

	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
Natural gas	kg/t	+ 10	- 10.4
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

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

	Unit	Change	Flame temp. (°C)	Top temp. (°C)
Blast temperature	°C	+ 100	+ 65	- 15
Coal	kg/t	+ 10	- 30	+ 9
Natural gas	kg/t	+ 10	- 87	+ 19
Oil	kg/t	+ 10	- 53	+ 4
Oxygen	%	+ 1	+ 45	- 15
Moisture	g/m ³ STP	+ 10	- 50	+ 9

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

Annex V *Coal types used for coke making*

The grade or rank of coals depends on the formation history. Coals come from plants that were growing millions of years ago. The plant remains were converted to coal under the influence of temperature and pressure without air access. The more intensively the plant remains are processed, the more volatiles escape from the coal and the higher its carbon content. The lowest rank of coal is lignite or brown coal. Increasing rank is in the sequence sub-bituminous coal, bituminous coal and anthracite.

Two major properties determine how much of a coal type can be used for coke production: coking behavior and caking capacity. Coals having good coking behavior can give high-strength coke. The caking capacity of coals is the capacity to form soft semi-liquid mass during their heating. The resolidification of this mass results in particles sticking together. Presence of coals with high caking capacity in the coal charge allow for the use of coals with poor caking capacity. Good coking coals have good caking capacity but the reverse is not necessarily true.

The caking capacity of good coking coals is indicated with the Roga index, which characterizes the thickness of the plastic layer formed during coal heating. Caking capacity of poor coking coals is evaluated using the free swelling index, which is a standardized test to measure the volume increase of a coal sample during its heating. Indicators of petrographic composition of coals and indicators characterizing coal metamorphism, in particular, level, vitrinite reflectance index (R_o) are also used.

Coals used for coke production are subdivided into four groups by their coking behavior.

- hard coking coals form strong hard coke such as coals A, B, C in Table A5.1.
- medium-hard coking coals as D, E, F, G.
- soft coking coals are gas coal J as well as H, and I.
- non-coking coals like K and L.

Coals with very low volatile matter, like anthracite, or very high volatile matter are generally considered unsuitable for coke making.

Average characteristics of the various coal types from different (Russian) coal basins are given in Table A5.1. Coal types differ in volatile matter content, thickness of plastic layer (U) or reflectance index R_o .

	Hard coking coal			Medium hard coking coal				Soft coking coal			Non-coking coal	
	A	B	C	D	E	F	G	H	I	J	K	L
Volatile matter, %	24–25	25–30	28–37		36–38	27–37	18–27			33–37	25–37	8–16
U, mm	13–17	18	14–26	10–12	10–13	16–25	6–13		6–9	10–16	–	–
Ro *, %	1–1,69	0,9–1,29	0,8–1,19	0,8–0,99	0,8–0,99	0,8–0,99	1,3–1,8	0,8–1,09	1,1–1,69	0,5–0,89	0,7–1,79	1,3–2,6

Table A5.1 Average characteristics of the various coal types from different (Russian) coal basins.

Coal blend for coke making

Coal blends for coking normally consist of 4–8 coal types from the first three groups. Coals from the fourth group are used less extensively and in small quantities. In order to ensure a consistent quality of coke, the share of the of the various coals in the blend should also be sufficiently consistent and be kept within ±2 % (abs). This will lead to acceptable standard deviations in moisture content (±1 %), ash content (±0,5 %), volatile content (±0,7 %), sulphur content (±0,05 %) and fines (< 0,5 mm, ±2 %).

When the coal blend is heated, pyrolysis takes place in several stages: release of volatile matter, softening, melting, hardening, partial graphitization of carbon. Release of volatile matter starts at 300 °C and continues with varying rate in the 300–1150 °C temperature range.

Different coals have different temperature intervals for the various stages of pyrolysis. When composing a coal blend, the temperature intervals of the various stages must at least partially coincide for all the coals. In the coking process, new conditions for physical and thermo-chemical processes are created, which do not necessarily coincide with conditions for individual coal types. As a result, the coking properties (coking behavior, caking capacity) can be better or worse than expected on the basis of averaging the properties of the individual coals.

The non-linearity of coal properties have to be known to predict the metallurgical coke properties. These factors can determined from multiple trials and/or pilot coking of individual coals and their blends.

For good quality coke, the coal blend should contain at least 40 % of hard coking coals with high caking capacity. The coal blend should have the following characteristics:

- vitrinite reflectance indexes R0 above 1.12
- plastic layer thickness U around 15 mm
- volatile content between 25 and 27 %
- sulphur content below 0,7 %
- ash content between 7,8 and 10 %

Annex VI *Coke quality tests*

Since drum tests are only cold simulations of the load on the coke during its descent through the blast furnace, there are different ideas as to the best way to generate comparative quality values using the drum test. Some of the differences between the various tests are; how the sample is taken as input for the test; the number of rotations; the size of the screens using to determine the size of the resulting coke; and the dimensions of the drum. In Table A6.1 the differences of the most common used drum tests are presented.

	Test						Strength Indices	
	Coke		Drum		Test		Breakage	Abrasion
	Weight kg	Size mm	Length m	Diam. m	rpm	Total rev.		
Micum	50	> 60	1	1	25	100	M_{40} % > 40 mm	M_{10} % < 10 mm
ISO	50	> 20	1	1	25	100	M_{40} % > 40 mm	M_{10} % < 10 mm
Extended Micum	50	> 60	1	1	25	100, 200, 300, 500, 800	Fissure free size Stabilisa- tion index	Micum Slope
IRSID	50	> 20	1	1	25	500	I_{40} % > 40 mm	I_{10} % < 10 mm
ASTM	10	2–3"	0.46	0.91	24	1,400	% > 1" (25 mm)	% > 1/4" (6 mm)
Japanese Drum	10	> 50	1.5	1.5	15	30 or 150		% > 15 mm

Table A6.1 Differences between the most commonly applied drum tests.

To have a better understanding of coke degradation mechanism under mechanical stress we look at Figure A6.1. Here the percentage of the coke larger than 40 mm and smaller than 10 mm of the sample are presented as a function of the number of rotations of the drum.

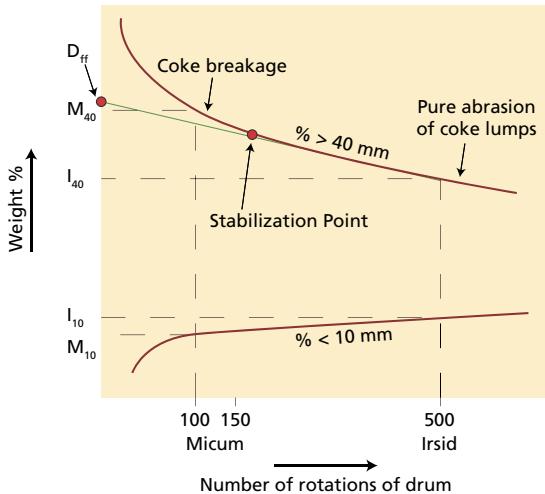


Figure A6.1 Comparison of different mechanical tumble tests and results.

From this figure we see that the lumps > 40 mm starts to degrade only by breakage until the point of stabilization is reached, when no further breakage occur. From this point on only abrasion takes place to further degrade the coke. In general the coke is stabilized after about 150 rotations of the Micum drum or an equivalent mechanical load. From this figure we see the great difference in number of rotations of the drum between the Micum test and the Irsid test. An advantage of the Irsid test is that the coke is always completely stabilized which makes the result less sensitive for the point of sampling. It further shows that it is in principle not correct to compare test results between different production sites unless the exact the degree of stabilization at the sampling points is known. The weight percentage of coke larger than 40 mm after 100 rotations is called M₄₀ and the percentage after 500 rotations is called the I₄₀. The weight percentage of coke smaller than 10 mm is called M₁₀ and I₁₀ respectively.

Besides these values, the Fissure Free Size, the Stabilization Index and the Micum slope have been introduced as coke quality parameters. Although in this test the parameter used is not the percentage larger than 40 mm of the coke but the average mean size (AMS) as a function of rotations. We will explain these concepts with Figure A6.1 as well. First we fit a line (shown in green) to the curve of abrasion only. Then we extrapolate the green line of abrasion—only to the y-intercept (zero rotations) and calculate the AMS of the coke at this point, which gives the Fissure Free Size (FFS), also known as D_{ff}. This then represents the size at which there would be no degradation due to breakage, but only abrasion. The slope of the green line of abrasion—only is called the Micum Slope. Some mills consider this to be a better way to evaluate abradability than traditional M₁₀ or I₁₀. The FFS was developed to simulate a maximum obtainable (theoretical) size for stabilized coke. Some believe the FFS approximately represents the size of stabilized industrial coke at the

blast furnace stock line, which is then considered a more suitable controlling parameter. Also a stabilization index can be defined as FFS/AMS, which will be at its maximum of 1 for fully stabilized coke.

Chemical reactivity

Besides a high mechanical strength, coke should have a high resistance against chemical attack. There are two measurements for the reaction with CO₂ most commonly used, the CRI and the CSR (Coke Reactivity Index and Coke Strength after reaction).

Coke Reactivity Index

Reactivity of coke can be tested in numerous ways, but by far the most common way to determine the coke reactivity is the Nippon Steel Chemical Reactivity Index (CRI). With this test, coke of a certain size is put under a 100% CO₂ atmosphere at 1100 °C. The percentage of coke that is gasified after 120 minutes gives the CRI value. The more reactive the coke, the higher the mass loss will be. Reactivity of the coke is mainly determined by the chemical composition of the parent coal blend, because ash components act as catalysts for the reaction of C with CO₂.

Coke Strength after Reaction

Due to the loss of mass whilst under attack by CO₂, the surface layer of the coke particles get very porous and the mechanical strength against abrasion drops rapidly. To measure this effect the reactivity test is normally followed by a tumbler test to determine the residual coke strength. The percentage of particles that remain larger than 10 mm after 600 rotations is called the 'coke strength after reaction' or CSR index. For most coke produced there exists a strong correlation between CRI and CSR.

Before CRI and CSR were developed, a series of relatively expensive tests were carried out under various research projects that involved partially gasifying the coke in its original particle size under realistic blast furnace conditions before subjecting it to the standard drum test. While the results of this costly research work showed exactly how the coke in the blast furnace was subjected to chemical attack, it provided no better information on coke quality than the more-simple method of determining CRI and CSR. These two parameters are now generally adopted by the coke-making industry as the most important parameters for determining coke quality.

Carburization of Hot Metal

There is no standard test for the dissolution of carbon in hot metal, the carburization. Experiments were conducted on this item by the Institute of Ferrous Metallurgy in Germany to compare different cokes of different coal blends and coke making technologies. The experiments showed a very similar

behaviour between most cokes. The only exception was the traditionally produced beehive coke. Although it had a very good CSR and CRI it was the only coke examined that cannot be used alone in a blast furnace because of its poor carburization characteristics. Production trials prove that this type of coke can only be used in a mixture with other more reactive coke.

Annex VII *Expert systems and models*

An expert system helps the operator to continuously analyse all process data and to suggest or execute corrective actions in real time. The corrective actions are based on the deviation of major process parameters from their target values and predefined rules of how to correct for the deviations. Any company with a book of “Standard Operating Procedures” can consider an expert system by automating these operating procedures in the process computer. The major advantages of expert systems are:

- Continuous monitoring of all relevant process parameters.
- Possibility to develop the system to improve the decision making rules.

However, an expert system has its limitations for some parts of the process and is not suitable for controlling special situations (blow-in, sudden chill). An expert system is no replacement of experienced blast furnace operations experts.

A7.1 Visualization of process conditions in real time

The process control system shows screens where the operator can monitor all variables of the process for a certain aspect of the process. An example is shown below. There are three important areas:

- Diagnostics: process parameters are scaled between -1 and +1, or between 0 and 1 and “red” and “green” areas are presented, red showing parameters outside the normal control area.
- Control actions or suggested control actions. The suggested actions serve to establish a dialogue within the shift, which allows the operator to decide whether or not the suggested action is implemented. The comment of the operator on the suggested actions even when not implemented, allows the optimisation of the process rules within the system.
- “Open” or “closed” decisions on suggested actions: in the open situation, the expert system presents a suggested action, but does not react if no decision is made. Closed decisions are executed automatically after 20 minutes if no rejection has been received by the system.

A7.2 What process parameters and diagnosis can be made?

The expert system follows a huge number of measurements in the furnace and makes a diagnosis whether or not the parameter is in a desired range. Among others, the following diagnosis are made:

- Burden descent: compares the burden descent of the last 24 hours with the last hour and analyses whether or not the burden descent is slow, normal or accelerated. (See section on direct reduction)

- Slag and hot metal quantity produced and drained from the furnace. The production balance is made based on progress of the last 25 casts as a reference and subsequently analyses the last cast and last three casts to monitor high, normal or low slag drainage. The same is done for hot metal.
- Thermal state of the furnace: based on temperature, hot metal carbon, silicon, sulphur and titanium as well as slag FeO level. The thermal state is considered low, normal or high.
- Stave temperatures and stave heat fluxes: continuous analysis of stave temperatures and heat fluxes on the various levels, defining 2 types of conditions: the current level and current tendency. The diagnosis gives an alarm when 25% of the temperatures are below lower level or above upper level limits with the corresponding decreasing or increasing tendency.
- Skull/scaffold formation: follows the temperature and heat fluxes through the staves and defines scaffold formation as decreasing tendencies of both parameters for the last 24 hours. To this end, the individual staves are taken in “areas” representing several staves. Occasionally it is found that the loss of thermal activity in the staves is also manifest in neighbouring areas. The opposite of the skull/scaffold formation is called “peeling”, which is the loss of the skull. This is analysed based with the same data.
- Charged burden: compares setpoint weights of materials charged with actual weights and monitors the deviation from setpoint.
- Hot blast stoves: analysis of the moments that stoves are changed for every stove, allowing the operator to operate the stoves within ± 10 °C of the target value.

A7.3 Process control actions

Process control in the blast furnace has various areas and targets.

- Thermal control: the liquid products have to be drained from the hearth at the desired temperatures.
- Chemical control: the hot metal and slag need to have the desired chemical composition.
- Gas flow control: the gas flow in the furnace can be monitored and has to be optimized for efficiency of contact between gas and burden.
- Casthouse control: the hot metal and slag have to be removed from the furnace hearth at the desired rates and intervals of time.

Control of equipment like tuyeres and coolers.

Incident control: how to manage unwanted and unexpected process conditions for which no cause is known: what to do if a furnace is out-of-control: slips, hangs, sudden chills, etcetera.

In the table below the various subjects for process control are summarised:

- the indicators which are used and
- the corrective actions.

Table A7.1 on the next page shows a typical example of what can be done. Every company develops its own expert system, based on its own experiences and operating practices.

Field	indicator/diagnostics	corrective action
Thermal Control	Hot Metal Silicon	Step 1: adjust injection rate 2–5 kg/tHM
	Hot Metal Temperature	Step 2: adjust coke rate
	Hot Metal Carbon (Ti, Mn)	in severe chills:
	Slag FeO	Extra coke
Chemical Control	New chemical analysis burden materials	Change input according to guidelines
	Slag basicity	Change input
	Burden input	Adjust input
	Hot metal Manganese	Adjust input
Gas Flow Control	Coke % in center	Adjust central flow by increase/decrease coke% in center
	Coke % at wall	Correct burden distribution
	Drying capacity/water elimination	Increase gas volume/tHM (lower oxygen enrichment, higher fuel rate)
	Too high Delta~P, total column	Lower blast volume
	Too high Delta-P, upper or lower part	Check fines coming into furnace, check hearth drainage
Casthouse Control	Wall gas flow as manifest from spikes in stave temperatures (peeling), high local heat loss or decreasing peaks	Adjust blast volume (longer term: burden distribution)
	Slag and hot metal drainage	
Incidents	Burden level too low	Caused by charging system: extra coke, pull wind
		Caused by process: decrease wind volume, extra coke. control Delta-P
	Sudden chills (temperature and silicon drop)	Extra coke, lower wind volume
	Water leakage/top gas hydrogen	From cooling system, burnt tuyeres: repair, a stop may be required

Table A7.1 Indicators and corrective actions in an expert system

An expert system is able to check in real time a huge number of measurements and is able to suggest process control actions to the operators for the various area. In the present section a few examples are indicated how an expert system works.

A7.4 Examples of process control with an expert system

Thermal control

Thermal control can be based on the hot metal temperature and hot metal silicon. Both factors have their own weight. An example is, that the thermal stage of the furnace is judged from the average hot metal temperature and

silicon of the last three casts. Since three casts represents a production, which is larger than the production coming from the complete working volume of the furnace, the hot metal temperature and silicon are always delayed with a number of hours. Therefore, the system checks whether or not there are indications of increased production rate, for example from the charging rate. If so, it is taken into account. The correction can be executed by changing fuel injection or by adjusting the coke rate. This type of choices are build in the system according to local experience and preferences.

Control of injection (coal, gas)

The operator targets a constant injection rate per ton hot metal produced. Since the actual flow rate of the injectant is measured in kg/hr (coal) or m³ STP/hr (natural gas) the set point for the injection has to be corrected for the actual production. The main factor determining the actual production is the amount of oxygen per hour blown into the blast furnace. So the injectant control should take into account hot blast volume changes, as well as, changes in oxygen enrichment and steam injection. An additional correction can be made for deviations in the direct reduction rate: if direct reduction increases (for example as a consequence of input of more fines), then the production rate will increase, even if all set points at tuyeres are maintained. This can be observed from the charging rate. A faster response can be generated from continuous mass balance of the gas, as shown in section 6.5.

Tracking the charge

When the burden is changed, the revised burden descends to the tuyeres in 5–7 hours. The tracking system visualizes where in the furnace the change is present in real time. This is important for extra coke and/or basicity changes when preparing for a furnace stop or shutdown. An example is shown in Figure A7.1.

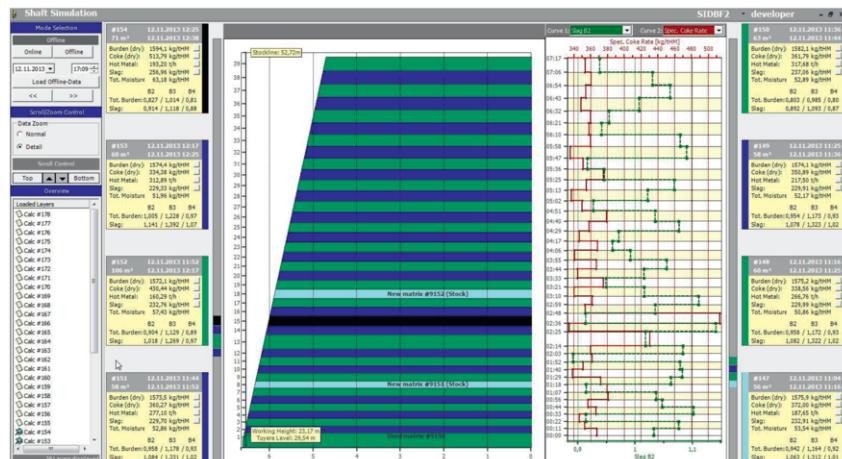


Figure A7.1 Tracking the charge: to monitor when changes are being processed with an extra coke descending in the charge.

A7.5 Examples of models

Mass and heat balance and minimum fuel rate model

The mass and heat balance model checks the input and output of the furnace on a daily basis. It does so by closing the material balance as well as the enthalpy balance. Most companies use a two-stage mass and heat balance, where the upper furnace and lower furnace are separated by the plane where it is considered gas temperature = 900 °C, O/Fe ratio in burden is 1.

The mass and heat balance when used continuously can serve as an early warning signal for sudden chills, because increase of direct reduction can be calculated in real time from the top gas composition.

Hearth wear model

The hearth of a blast furnace is equipped with numerous thermocouples. The temperature readings combined with the thermal properties of the refractories can be used to calculate the isotherms in the furnace hearth. The isotherm of 1150 °C represents the solid–liquid interface, so is indicative for the hearth refractory wear. An example is shown in Figure A7.2

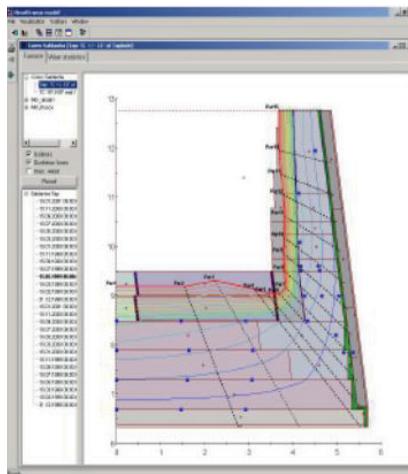


Figure A7.2 Hearth isotherms

The actual condition of the hearth is shown by the actual thermocouples. But to estimate the refractory condition of the hearth, the campaign highest readings have to be analysed: the highest value in the campaign indicates the border of good hearth refractories, since refractories when gone are not restored, but scabs are formed in the hearth. Some blast furnace hearth models even calculate the thickness of the skull on the hearth sidewall and bottom.

Use of the model helps to maintain the integrity of hearth wear monitoring system. Many thermocouples fail during a campaign, so a program for repair and/or elimination from the model has to be in place in order to secure accurate results.

While the model is very useful for repair/reline planning, daily operations are more reactive on high temperature readings in a certain area. If temperatures at hearth wall become too high, there is the risk of a break-out. Most operators know the weak points of their hearth refractory and alarm temperatures are put into the hearth monitoring system. When temperatures become too high, there are corrective actions possible like plugging tuyeres above the affected area.

Annex VIII *Rist diagram*

The Rist diagram is a graphical tool for analysis of the mass and heat exchange in the blast furnace in steady-state condition, i.e. with constant material flows, composition and temperatures of the blast furnace input (charge, fuel, hot blast) and output (hot metal, slag, top gas) as well as heat losses.

The Rist diagram is a graphical presentation of a mathematical balance model of the blast furnace process and it can be used to analyze operational efficiency. In the present annex charge materials without limestone, dolomite or carbonate ore are used.

In this situation the heat consumption of the process is determined by the processes in the lower part of the furnace (i.e. below the thermal reserve zone) since gas reaching the upper part of the furnace contains sufficient energy to heat the materials, drive chemical reactions and for heat losses. The upper part of the furnace is called the dry zone of BF shaft.

The upper part of the furnace is a counter current mass exchanger, where oxygen is transferred from iron to gas (C and H₂). The iron flow (n_{Fe} , atom Fe/min) reducing gas ($n_{(C+H_2)}$, (atom C + mol H₂)/min) have initial oxidation (Y_O , atom O/atom Fe), X_O , (at. O + mol H₂)/(atom C + mol H₂) at the reactor entry and final oxidation (Y_f , X_f) – at its exit. This is shown in Figure A8.1 and in table A8.1.

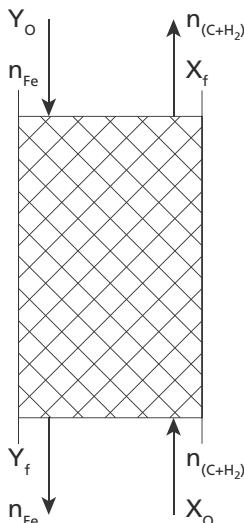


Figure A8.1 The reduction of iron oxides in the upper part of a blast furnace reactor: the amount of iron processed is shown as n_{Fe} , the oxidation degree at input of the reactor Y_O and at exit as Y_f . The amount of gas as $n_{(C+H_2)}$ and its oxidation degree as X_O (at inlet, lower part) and X_f at exit.

The mass exchange equation for such a reactor shall be as follows:

$n(C+H_2)(X_f - X_O) = nFe(Y_O - Y_f)$ or for any other point through the height of the reactor:

$$Y_f = Y_O - n(C+H_2) / nFe(X_f - X_O) \text{ or } Y = Y_O - \mu(X - X_O) \quad (1)$$

where $\mu = n(C+H_2) / nFe$ – is reducing gas consumption, (atom C+mol H₂)/atom Fe

	symbol	what	unit	top	bottom (900 °C)	comment
ferrous burden	n_{Fe}	production	atom Fe/min	124,452	124,452	as example 10,000 tHM/d
oxidation burden	Y_O Y_f	oxidation degree	atom O per atom Fe	1.47 (= Y_O)	0.5 (= Y_f)	Hematite, Fe_2O_3 has $Y_O = 1.5$
gas flow (CO, CO ₂ , H ₂)	$n(C+H_2)$	gas flow	(atom C + mol H ₂)/min	252,232	252,232	based on 250 m ³ STP/t O ₂ , 120 kg C direct reduction, 200 kg PCI with 4% hydrogen
oxygen per atom (C+H ₂)	X_O X_f	oxygen bound to C and H ₂	(at. O + mol H ₂)/(at. C + mol H ₂)	1.38	1.14	bottom ETACO = 24%, EТАH2=35%, top ETACO = 50%, EТАH2=40%

Table A8.1 Definition of the parameters used in the Rist diagram.

The equation (1) is an “operational straight line” and describes the oxygen transfer from burden to gas in the blast furnace shaft.

In a simple case of a furnace operating with dry blast and without hydrocarbon injection, i.e. with no hydrogen content in the blast furnace gas on the axes of the Rist diagram are shown:

Y-axis = atom O/atom Fe

X-axis = atom O/atom C.

When hydrogen is present in blast furnace gas the coordinates of the above axes change to:

Y-axis = (atom O + mol H₂)/atom Fe

X-axis = (atom O+ mol H₂)/(atom C+ mol H₂).

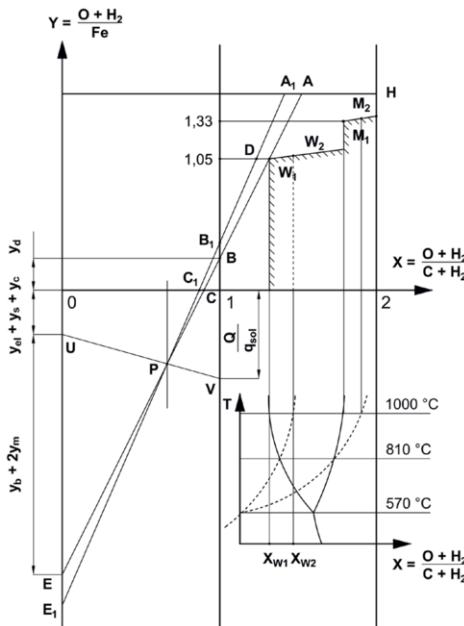


Figure A8.2 The Rist diagram, for explanation see text

Review Rist diagram

If we look on the x-axis, the point where the x-axis = 1 is the point, where $(O+H_2)/(C+H_2)=1$, which means that all the gas is in the form of CO and H_2 , so in the lower part of the furnace.

Let us follow the operation line in Figure A8.2 from left under (point E) to the right at the top point A. In the area between $X=0$ to $X=1$ the line shows generation of reducing gas (CO and H_2) in the blast furnace during coke combustion at the tuyeres (segment UE on the axis of ordinates), reduction of silicon, manganese, phosphorus and other elements by carbon (segment 0U on the Y-axis) and direct iron reduction (segment B0 on the Y-axis).

The quantity of reduction gas (mol/atom Fe) is determined by the blast oxygen consumption (y_b , atom O/atom Fe), oxygen of the direct reduction of elements (y_{el} , atom O/atom Fe) and of desulphurization reaction (y_s , atom O/atom Fe) and hydrogen of coke volatile matters (y_c , mol H_2 /atom Fe).

On point B the gas is in the form of CO and H_2 and the line BA shows the gas reduction of the burden and thus the oxidation of CO to CO_2 and H_2 to H_2O . The value of point W₁ on the x-axis shows the equilibrium composition for reaction of FeO reduction by carbon monoxide in chemical reserve zone, while value of point W₂ on the x-axis shows equilibrium composition for reaction of FeO by hydrogen in the same zone (Figure A8.2).

When CO and H_2 are both present in the gas, point W can be determined as a

weighted average $W = W_1 + h(W_2 - W_1)$, where h – is the share of hydrogen in $H_2 + CO$ mixture.

The operational straight line passes through point W only, if gas reaches the equilibrium composition. This is the ideal case, where the reduction process is executed with the minimum reduction gas and therefore with minimum fuel rate.

In a less than ideal case, gas equilibrium is not reached and the operational straight line passes to the left of point W (for example through point D in Fig Annex 8.2). The closer the operational straight line comes to point W, the more efficient the reduction processes in the blast furnace.

This is indicated by $\delta = (X_D/X_W)$. The X-value of point W (X_W) depends on the gas temperature in thermal reserve zone of the blast furnace and hydrogen content in the reducing gas. The operation line for an operating furnace can be drawn through 2 of the 3 points available i.e. A, E, P.

For point A: The Y value of point A (Y_A) is determined by oxidation of materials charged into blast furnace (atom O/atom Fe) and its X-value (X_A) – by oxidation of the top gas (atom O + mol H_2)/(atom C + mol H_2) or $(2CO_2 + 2H_2O + CO + H_2)/(CO_2 + H_2O + CO + H_2) = 1 + (CO_2 + H_2O)/(CO_2 + H_2O + CO + H_2)$.

For point E: The Y-value of point E (Y_E) is determined by quantity of oxygen (atom O/atom Fe) of the blast (segment E_U) and oxygen of direct reduction of elements (segment from 0 to point U on the axis of ordinates). The X-value of point E is 0.

For point P (X_p) is determined by temperature and moisture content of the blast, while the Y-value (Y_p) depends on quantity of oxygen for direct reduction of elements and heat requirement of lower part of the furnace, i.e. it is determined by the y-values of points U and V. Y_V in the diagram is determined as the ratio of heat requirement of the lower part (Q , kcal /atom Fe) and the heat for (q_s , kcal /atom C) the gasification reaction (reaction of solution loss $CO_2 + C \leftrightarrow 2CO$): $Y_V = Q/q_s$, atom C/atom Fe .

The heat required to run the process in the lower part is needed for all endothermic reactions, smelting/heating of hot metal and slag as well as heat losses.

The equation of the operation line at maximum efficiency ($\delta = 1$) can be derived from the temperature of the thermal reserve zone and coordinates of points W and P.

In this case the Y-value of point W is equal to oxidation of wustite $Y_W = 1,05$ atom O/atom Fe, while the X-value is equal to the equilibrium condition of the gas as can be determined from reference data for a given temperature in the thermal reserve zone. In this case the equation of operation line allows the

operator to find theoretically minimum possible fuel rate.

The equation of the operation line for an operating furnace makes it possible to assess the efficiency of reduction processes in the furnace and the degree of direct reduction (Y_B), or to forecast the furnace performance (fuel rate) for changes in blast furnace operating conditions.

The position of point P can be calculated from the temperature and moisture content of the blast as well as from the heat and mass balance of the BF lower part.

When injecting hydrocarbons into blast furnace tuyeres (Figure A8.3), blast furnace gas contains additional hydrogen and CO gas from injected fuel, for example from coal injected with PCI. The additional amount of H_2 and CO gas ($y_j(a+b)$ in Figure A8.3) is determined by the injection rate (y_j , mol/atom Fe) as well as by the content of H_2 and oxygen in it (a, mol H_2 /mol and b, atom O/mol). Heat required in the lower part of the furnace then increases with the amount of heat, required for dissociation of hydrocarbons of injected fuel (q_j/q_{sol}). In this case the Y-value of the point W depends on the share of hydrogen in the reduction gas and the gas temperature in the thermal reserve zone $W = W_1 + h(W_2 - W_1)$.

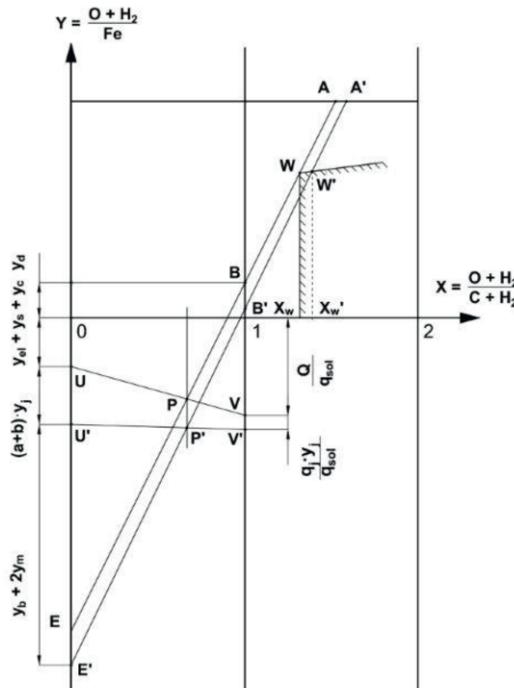


Figure A8.3 The Rist diagram including injected fuels, for explanation see text.

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Index

- alkali, reactions 169
- alkali, recirculation 169
- Banded Iron Formation 26
- basicity 138
- blast furnace profile 10
- blow-down 181
- blow-in 185
- blowpipe failures 179
- Boudouard equilibrium 116
- briquettes 39
- burden calculation 78
- burden components, blending 131
- burden components, interaction 41
- burden descent, vertical forces 87
- burden descent, voidage 85
- burden distribution 96
- burden distribution, “ideal” 104
- burden tracking 202
- carbon and iron oxides 115
- cast, not dry 157
- casthouse 145
- casthouse operation 145
- casting delays 153
- channelling 96
- charging rate, variability 166
- charging systems 8
- chills 187
- coal ash deposits 66
- coal blending 65
- coal blends for coke 45
- coal gasification 66
- coal hardness 64
- coal injection equipment 62
- coal volatile matter 64
- coal-gas coinjection 73
- coal-oxygen working area 67

cohesive zone 90
coke cold strength 52
coke degradation 48
coke hot strength 53
coke layer 101
coke making, coal types 196
coke mess 189
coke quality 46, 57
coke rate, minimum 102
coke residence time 43
coke, quality tests 198
cold compression strength 36
cooling plates 12
cooling systems 11
CRI 53
CSR 53

deadman 54
deadman cleanliness index 54
decrepitation 39
desulphurization 137
direct reduction accompanying elements 117
direct reduction estimate 83
direct reduction increase, example 83
Direct reduction iron oxides 116
DRI 39

efficiency 111
efficiency, monitoring 112
expert system 202

fines and bin levels 161
fines, handling 161
fines, segregation 161
fluidization 96
freestanding 8

gas cleaning 7
gas flow 89
gas flow daily control 106
gas reduction 118
gas utilization 20
glossary 191
goethite 27
greenhouse gas emissions 189

- hanging 167
- harmonic mean size 50
- HBI 39
- hearth, liquid level 151
- heat flux 90
- hematite 26
- hot blast stoves 7
- hot metal composition 127
- hot metal composition and temperature 132
- hydrogen 121

- incomplete combustion 59, 75
- injectants 59
- inner volume 9
- instrumentation 106
- iron and slag separation 145

- K-factor 126

- layer structure 2
- lintel 8
- lump ore 39

- magnetite 26
- mass balance 79
- metallic charge 39
- mixed layer 98
- models 202
- moisture, input with charge 164
- moisture, recirculation 164

- natural gas, working area 71

- ore layer 103
- oxygen lancing 160

- pellets 35
- permeability 126
- pressure difference 87
- process, daily control 106
- production rate, variability 166
- productivity 110

- quality of ferrous burden 28
- quenched blast furnace 5

- raceway 15
- RAFT (Raceway Adiabatic Flame Temperature) 113
- Reducibility 31
- reduction-disintegration 31, 32, 128
- replacement ratio 59, 63
- Rist diagram 208
- rules of thumb 195
- scabs, at peepsites 179
- shutdown, procedure 181
- silicon reduction 133
- sinter 32
- slag composition 127
- slag flooding 132
- slag granulation 7
- slag properties 139
- slag properties at stops and starts 143
- slipping 166
- softening and melting 129
- start-up, after short stops 182
- staves 12
- steel consumption 1
- stockhouse 7
- stops and starts 180
- sulphur in hot metal 136
- swelling 37
- symmetry, circumferential 173
- taphole 147
- taphole operation, one side 156
- taphole operation, two sides 148
- temperature profile 124
- top gas composition 79
- top gas, formation 124
- total energy 71
- tuyeres 176
- tuyeres, burnt 177
- tuyeres, tipped 178
- water discipline 177
- whisker 37
- working volume 9
- wustite 26
- zinc, recirculation 172