CONCEPTUAL ASPECTS CONCERNING THE PELLETIZING OF IRON ORES

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Vitória ES – Brazil

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I dedicate this book to my wife Mirna and son Renato, who both encouraged me in writing it. I also dedicate it to the technicians, who I worked with and learned a lot about pelletizing, during more than 30 Years of professional life at the Brazilian mining company Vale SA.

I extend my thanks to all who, direct or indirectly, contributed to the preparation of this book. In special, to my colleagues, partners on the development of several engineering projects at the Brazilian mining company Vale SA.

The Author:

José Murilo Mourão was born in Divinópolis, state of Minas Gerais - MG, Brazil. He began his studies in this city, in 1961, at the elementary school named Grupo Escolar Miguel Couto. After completion of the primary step, migrated to the Ginásio São Geraldo, where, for four years, studied the secondary stage of his education. After that, moved to the city of Ouro Preto, state of Minas Gerais, joining to the Escola Técnica Federal, where got his diploma as Technician in Metallurgy, in 1971. From 1972 on, began his studies on Metallurgical Engineering at the Escola de Minas of the Federal University of Ouro Preto, from where was graduated in 1976.

While studying engineering, he acted as professor of the disciplines of physics, mineral beneficiation, and technical drawing, at the Escola Técnica Federal.

At the end of 1976, he started his professional career at Companhia Vale do Rio Doce SA, now Vale SA, at the division of iron ore pelletizing, in the city of Vitória, state of Espírito Santo – ES. There, during more than 30 years, played various roles in the areas of production, quality, planning, R&D, technology, and business management. He participated actively in the establishment and development of a complex of pelletizing plants of the company. He made many trips abroad and stayed in permanent contact with customers, aiming to the development of iron ore and agglomerates for ironmaking in blast furnace and direct reduction.

In programs of continuing education, took many technical and business courses at Brazilian and international universities.

He participated in many conferences and seminars worldwide, having written and published more than 50 technical papers related to the technologies of pelletizing and ironmaking. In special, is an active member of the Brazilian Association for Metallurgy, Materials and Mining (ABM), having contributed to the establishment of the annual seminar on agglomeration of iron ores.

Currently, has been collaborating with projects for the development of ABM and acting as an independent consultant in the fields of iron ore, agglomeration, and ironmaking, conducting studies for public and private institutions.

TABLE OF CONTENTS

PREFACE

This is a unique work in the area of pelletizing, with a wealth of details and extremely useful quantitative analyses, which serves as a reference for professionals and researchers in the subject. The conceptual aspects are associated with the technological issues of the process, allowing an easy assimilation. It meets the increasingly critical trend of the scarcity of iron ore lumps and the increase of availability of fine concentrates (pellet feed). It is a legacy of knowledge that the author, with extensive experience in the area, leaves in favor of the development of the pelletizing process to contribute to the improvement of competitiveness.

The approach is quite comprehensive and serves to understand the pelletizing processes from the preparation of the mixture of materials, the balling, the physical, mechanical, and chemical properties of the pellets, cold and after firing, including the behaviors, during the reduction of the downstream pellets.

The drying -burning stage is the one that involves the highest cost in pelletizing, with consequences in the greenhouse effect, when fossil fuel is used. Thus, the author analyzes with quantitative data how to obtain better energy efficiency.

The effects of quality, especially of the gangue and the nature of the ore, on productivity and energy consumption are also analyzed, both in pelletizing and in the subsequent stage of reduction-melting.

As has been said, the text presents a wealth of data with technological details not found in the literature in a consolidated way.

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São Paulo – SP, Brazil - October 27th, 2021

INTRODUCTION

The history of the pelletizing of iron ores started in 1912, in Sweden, with A.G. Andersson, who invented this process of agglomeration. The first industrial plants were installed in the decade of 1950, in United States of America (USA). (1) Many pelletizing processes were studied and tested in laboratory and pilot plants. Along the time, some of them have become obsolete. Nowadays, there are two processes, the so-called by Traveling Grate (TG) and Grate-Kiln (GK), the latter also known by Rotary-Kiln (RK).

Since then, the number of pelletizing plants has expanded in the world and the growth in production of pellets has accelerated in recent times as shown in the figure 1. (2) This evolution is mainly due to the depletion of the reserves of rich iron ore, which makes the process of preparation of ferrous burden for the steel industry more complex and expensive. The lower grade iron ores have to be beneficiated, involving grinding and concentration, what generates significant quantities of fines below 0.15 mm, suitable for pelletizing. In Brazil, this activity is well developed prevailing the use of the Traveling Grate technology (TG).

The pelletizing process was developed for agglomerating the pellet feed or concentrate of iron ore, a very fine material, no proper to directly charge the blast furnaces or direct reduction reactors in ironmaking.

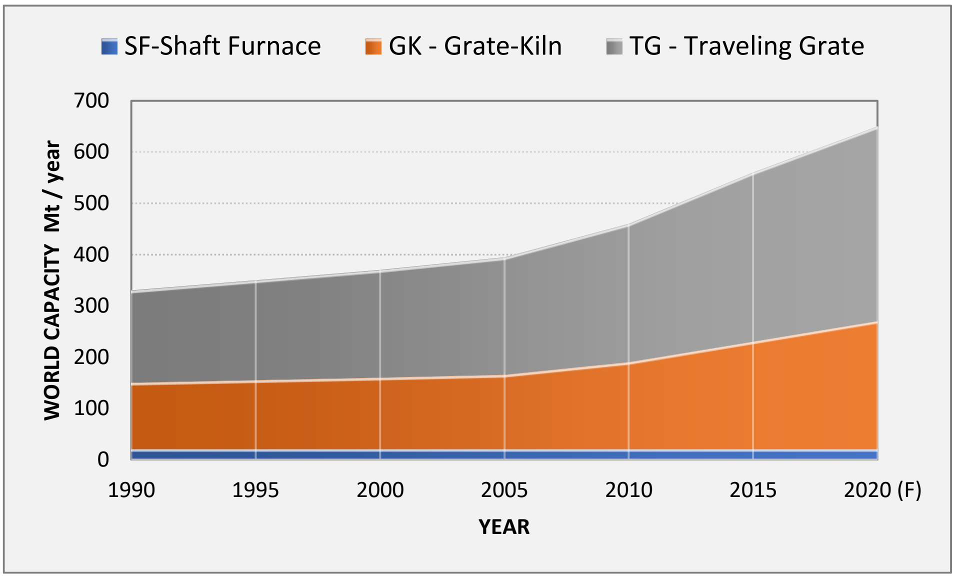


Figure 1 - Evolution of the world capacity of iron ore pelletizing. (2)

The pelletizing technology consists in agglomerating such fine iron ore particles into spheres with mean diameter around 12 mm, which have chemical, physical and metallurgical properties suitable for the use in steel industry.

This book was not written with focus on covering a wider extent of pelletizing technology, already explored by other important authors. It presents some conceptual themes, which have not been the object of further consideration or are scattered in treatises without the due clearness.

During more than 30 years working in this activity, there were evident the difficulties of the *technicians* and *junior engineers* in understanding some concepts here considered, aiming to develop their activities in operation and control of pelletizing plants.

It is expected that this contribution may cover a gap in this area and facilitate the continuing education and the development of those who militate in the art of pelletizing.

CHAPTER 2

BRIEF DESCRIPTION OF THE PELLETIZING PROCESSES (3)

Currently, two technologies dominate the market of pelletizing, accounting for more than 95% of the world production: Traveling Grate (TG) and Grate-Kiln (GK). The old and obsolete low-capacity process, called Shaft Furnace (SF), has also some furnaces in operation, especially in China. However, gradually, they tend to disappear, bearing in mind the higher operating costs.

The TG furnaces are reaching the capacity of 10 Mt/year, while those of GK type are being improved to achieve 6 Mt/year. The capacity of the SF furnace is limited to only 0.5 Mt/year. The TG process is indicated for processing any type of iron ore. The GK is more appropriate for magnetite ores or mixtures with hematite, although there are some GK plants operating with 100% of hematite ores.

Independently of the pelletizing technology considered, there are 3 stages of treatment of iron ore, aiming the production of pellets (Figure 2.1).

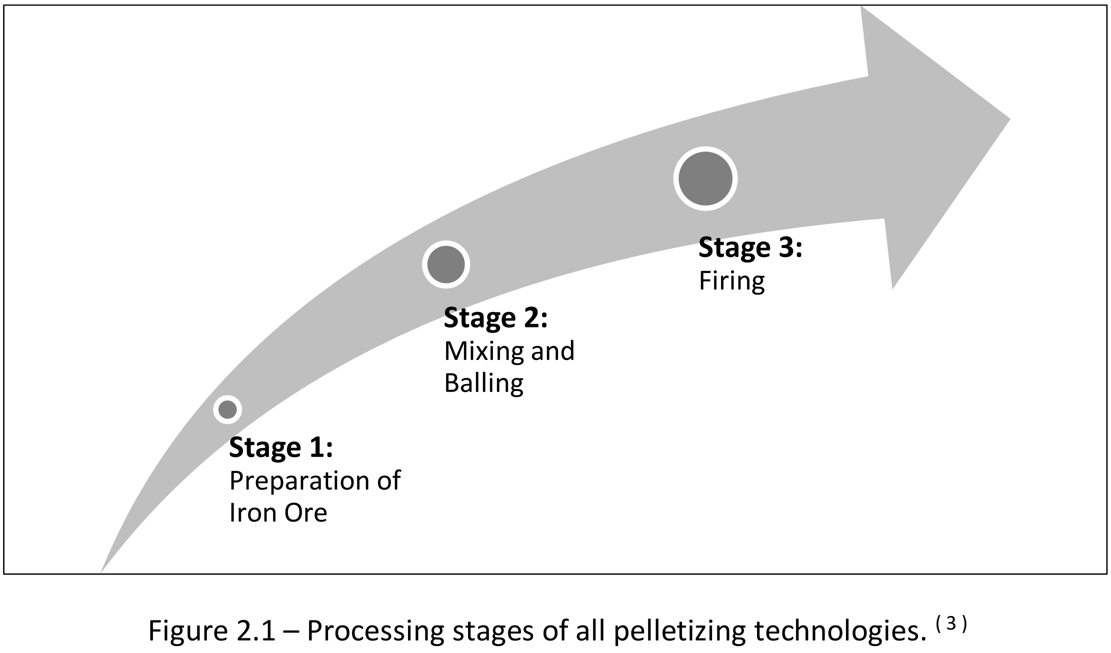


Figure 2.1 – Processing stages of all pelletizing technologies. (3)

2.1 - Stage 1 - Preparation of Iron Ore

The treatment of iron ore is necessary in order to achieve the fineness needed to the formation of green pellets in the following stage, as shown in figure 2.2, which reflects the mentioned 3 stages of pelletizing: wet grinding, balling with discs and firing, according to the Traveling Grate technology. (4)

This fineness depends on the characteristics of the ore. In general, specific surface area in the range of 1700 to 2200 cm2/g and a diameter of less than 0.074 mm are required for the formation of green pellets in balling section. These conditions are met by grinding the iron ore and the additives. Normally, the pellet feed obtained by concentration of poor ores is coarser than the specification. Dry and wet grinding in ball mills can be used for the comminution, depending on the type of ore and the conditions existing at the site of a pelletizing plant. The grinding is very costly, both in energy as well as in other wearing materials (balls, linings, etc.).

The capital and operating costs of the grinding mill are very important in a circuit of pelletizing. When the grinding is performed in wet basis, the operation of filtering is included in this stage 1, in order to obtain a filter cake with moisture in the range of 8 to 10%, proper to balling. If running dry way, a complex system of drying and classification of the ore is considered. In this latter case, the consumption of thermal energy for drying the ore is high and significantly impacts the operational cost.

In this case, the ore is dried for grinding and in the following stage 2, is wetted again in an amount of 8 to 10% of water, in order to give it the moisture content required for balling. However, the type of grinding mill used depends on several factors, including the type of ore, the energy and water sources, prices of consumables, etc. There are pelletizing plants where the stage 1 is not present in the flowchart of operations. This condition means that the plant receives the ore already ground and with fineness suitable for balling.

2.2 - Stage 2 - Mixing and balling

After obtaining the iron ore through the grinding and filtering units, as shown in the present case of figure 2.2, a binder (bentonite, hydrated lime, organic compounds, etc.), some fluxes (limestone, dolomite, dunite, magnesite, etc.) and, possibly, a solid fuel (anthracite, coke breeze, etc.), are added to it in order to produce the specified pellet.

The fluxes are important to adjust the basicity of the pellets, giving them the chemical, physical and metallurgical properties, suitable for their use in ironmaking. Preferably, these additives should be beneficiated separately and dosed to iron ore at the entrance of the mixer. However, each plant makes its way, according to the local conditions and the availability of raw materials. In certain plants, the fluxes are added to the pile of iron ore and ground together.

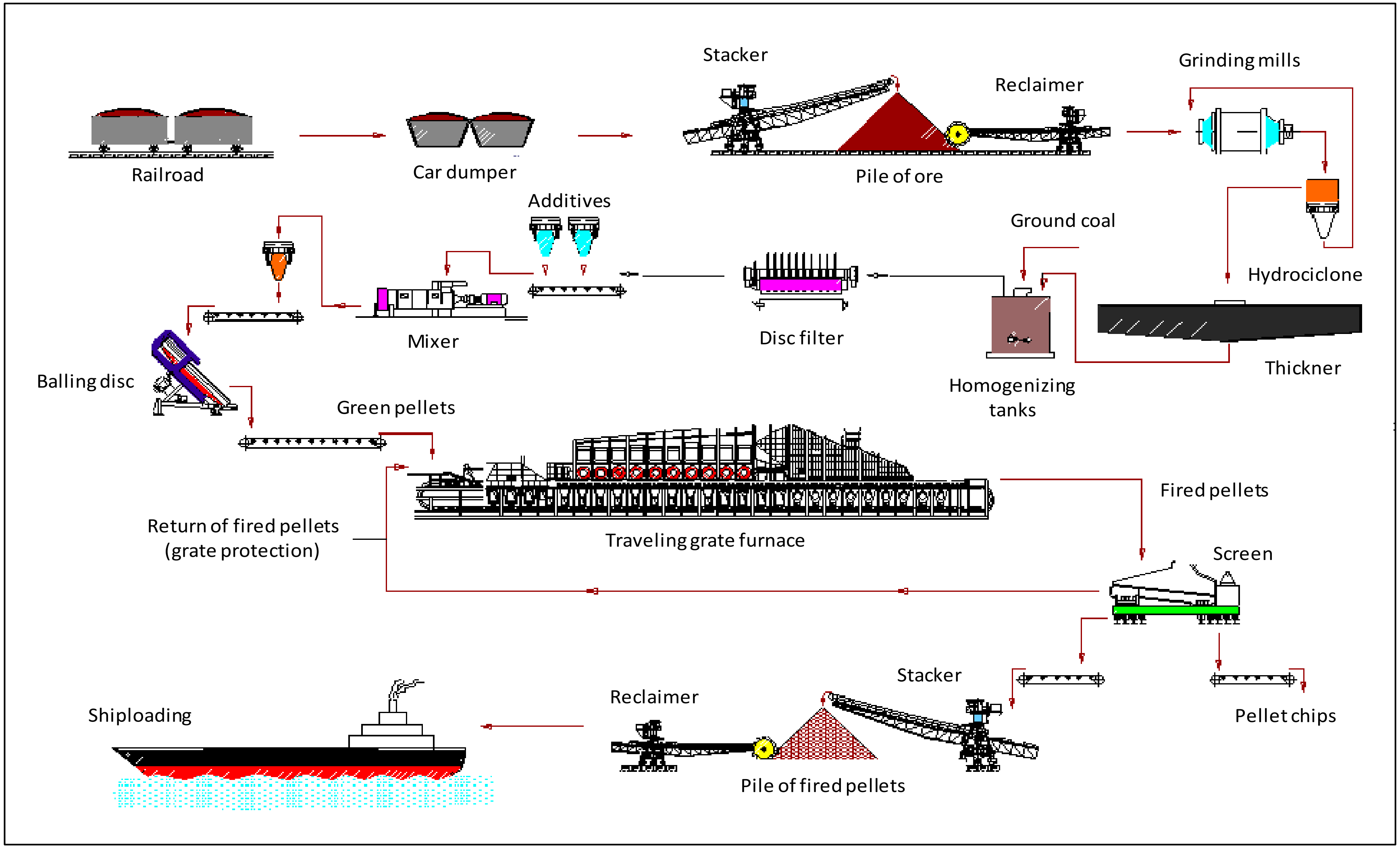


Figure 2.2 - Typical flowchart of a pelletizing plant with wet grinding, balling discs and Traveling Grate. (1, 4)

After filtration and the addition of additives, if necessary, adjustment is done in moisture content of the mixture to be balled. The materials are fed to the mixer after what the mixture is sent to balling unit. Discs or drums can be used for the production of green pellets. The pelletizing discs are used more often, in view of their greater simplicity. The size of the green pellets can be adjusted through the control of some process variables like: production rate, slope and speed of the disc, moisture content of the mixture, etc. Before entering the furnace, the green pellets are sieved (8-18 mm) and removed the bigger and smaller balls which are recirculated in the process.

2.3 - Stage 3 - Firing

This is the main stage of any process of pelletizing plant and the step of largest capital and operational costs. In fact, when one says in pelletizing technology, normally, is speaking about the firing, since the other two stages involve common operations at any one of the existing technologies. The Traveling Grate (TG) and Rotary-Kiln (RK) furnaces are used to burn (thermal processing) of green pellets obtained in balling section. To produce strong pellets and with adequate metallurgical properties, the green pellets are subjected to high temperatures (1250 to 1350 °C) inside of the furnaces, during a convenient and strictly controlled residence time. These furnaces have different zones, which perform the drying, preheating, burning and cooling of the product. The flows of hot gases (updraught and downdraught), the temperatures and pressures of the gases in each zone are controlled, in function of each type of pellet produced. The recirculation of hot gases from the cooling zone can recover part of the heat, improving the thermal efficiency of the furnaces.

The TG technology consists in a tubular shaped furnace, where, internally, there is a mobile grate, as shown in figure 2.3. All stages of firing are conducted in this single device, and there is no relative movement of the pellets among themselves or with the grate. The TG furnaces have been shown to be very effective in treating any type of iron ore or mixtures of different kinds of ores.

On the other hand, RK technology has three independent reactors connected in series: a movable grate, a rotary reactor (cylindrical and inclined) and a cooler in circular shape. Many plants of these two processes were built in different places of the world. However, the acquired experience has shown that the furnaces RK type have some limitations, to treat hematite ores and those hydrated (limonite and or goethite). The main reason is that, after the steps of drying and preheating of green pellets on the mobile grate, the material, yet little resistant, has to be transferred to the rotary reactor. Thus, fines can be generated in this transfer, thus reducing the efficiency of the process (in addition to other possible fines, due to the movement of the rotary reactor and transfer to the cooler). On the other hand, the green pellets of magnetite ore have a behavior very differently in those areas of the furnace, mainly in the pre-heating step.

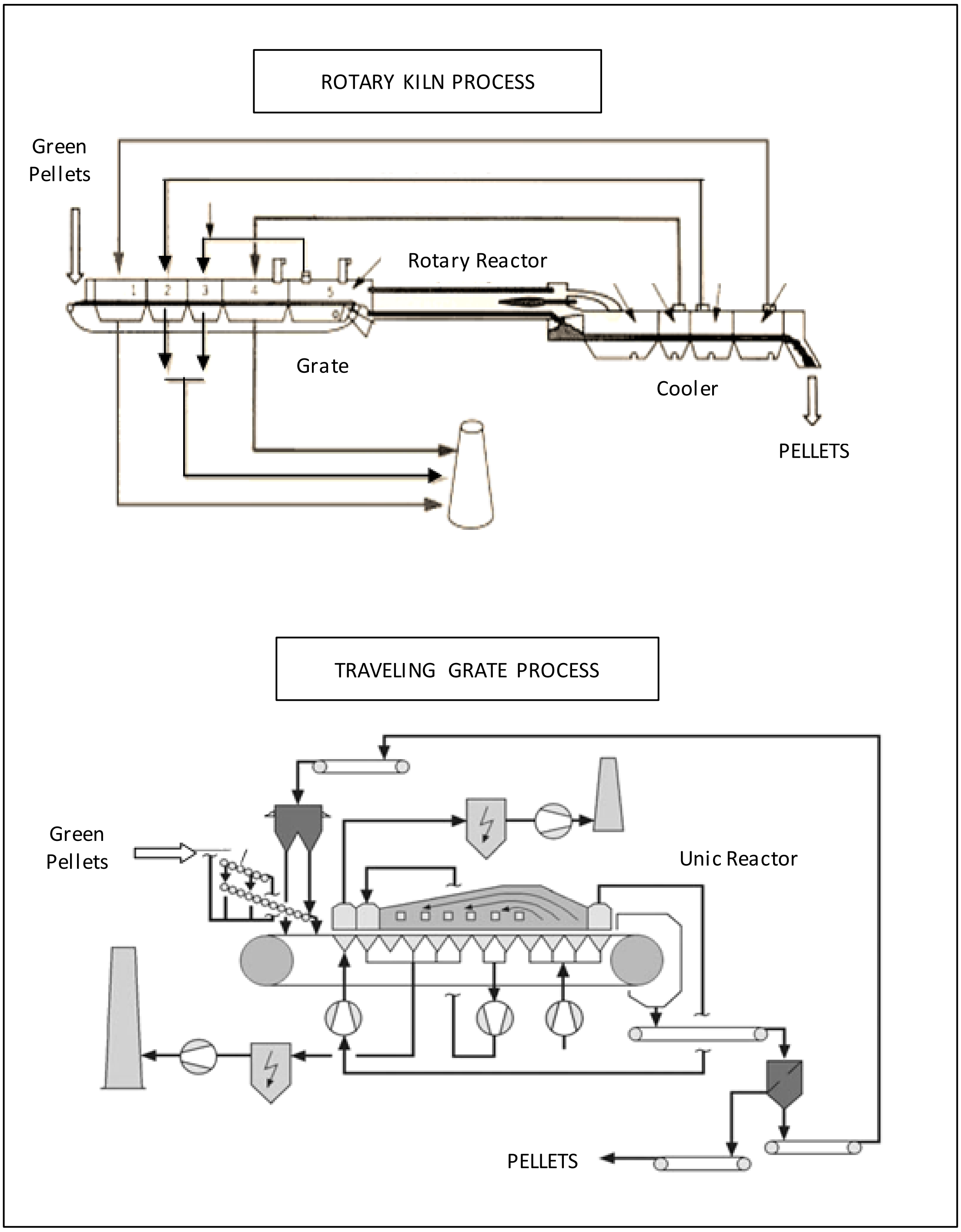


Figure 2.3 - Basic diagrams of Traveling Grate and Rotary-Kiln processes. (3)

In this kiln section, the oxidation of magnetite crystals occurs, accompanied by a recrystallization (transformation in hematite). Then, there is the formation of bridges involving the grains, which confers resistance to the agglomerate. This resistance is sufficient to support the operations in the following two reactors of the RK process. Most recent experiences on the use of hematite ore in RK have shown that it is necessary to raise, significantly, the temperature of the gases in the area of preheating zone, to compensate the low resistance of the hematite pellets in this region, at the end of which there is a transfer to the rotary reactor. This implies, usually, at a lower thermal efficiency of this RK process.

In any case, what will define the use of one or another technology is the study of technical and economic feasibility of the project, considering a certain type of ore. This iron ore should be tested at bench and pilot plant, highlighting the difficulty of simulating the transfers between the three reactors of the RK, and the problems relative to scale up of industrial reactors.

Normally, the output of the furnace is screened, and the pellet chips (< 5 mm) are eliminated from the final product (fired pellets).

CHAPTER 3

PELLETIZING AND IRONMAKING (5)

The iron mining is experiencing a dramatic expansion in recent years, fruit of the great escalation of production and consumption of steel in Asian countries, especially in China. The accelerated growth of wealth in this region, expressed by gross domestic product, leads to greater consumption of steel, in view of large government investment in infrastructure, development of the capital goods industry, consumption of durable goods, etc.

Iron ore is the basic raw material of steelmaking, responding for the metallic units (Fe) supply to the reducing reactors, as the blast furnace and the modules of direct reduction. Processed in these installations, the ore gives rise to the primary iron, Pig iron and DRI/HBI. The subsequent treatment of the primary iron in the steelworks converts it into steel. It is important to note that the iron and steel scrap also has an important role in the steel industry, considering its use as direct charging of electric arc furnaces. However, scrap has a smaller importance in steelmaking than the iron ore, accounting for something like 25 to 30% of the total supply of iron units.

In figure 3.1, one can see the interdependence of iron ore mining and steelmaking. In mining operations, there are the exploitation of the ore body and the transportation of the raw product (run of mine) to be beneficiated. This material is subjected to a series of operations involving fragmentation, classification by size, concentration, dewatering etc. The chemical, physical and metallurgical properties are adjusted in order to meet the demands of the steel processes thereafter. A very important point to characterize the mineral products that come out from the ore treatment plant is the global particle size distribution of the products. During mining, comminution and handling operations, a lot of fines is generated. Such fines are inadequate to direct use in reducing reactors of the steel mills. To this end, it is necessary to agglomerate them, through sintering or pelletizing technologies. Generally, it could be nominated the following mineral products, depending on the size distribution:

Product	Size Range	Basic Application
Lump ore	6.3 to 31.7 mm	blast furnace and direct reduction
Sinter Feed	0.15 to 6.3 mm	agglomeration via sintering
Pellet Feed	< 0.15 mm	agglomeration via pelletizing

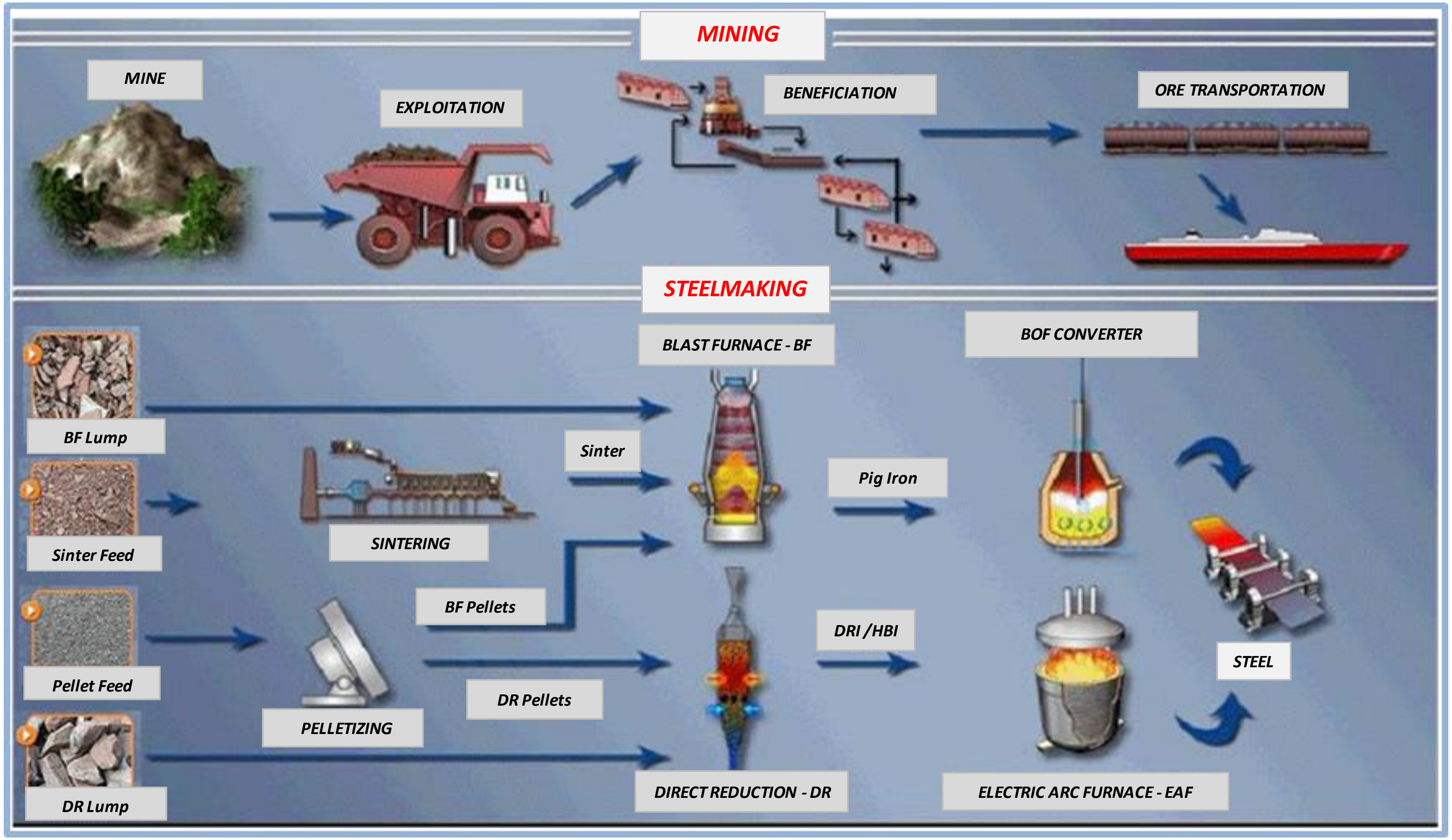


Figure 3.1 – Pelletizing and Steelmaking (1, 4, 5)

The activity of iron mining is as much more profitable as more lump ore is possible to generate. In this aspect, the quality of the mineral reserves is a determining factor of business success.

The processes of concentration and agglomeration are of high operational costs, require large amounts of investment and have significant environmental impact. Ore fines, like sinter feed and pellet feed, need to be modified for using in the steel industry. The great advantage of agglomerating them is the possibility of adding significant value to the products, sinter, and pellets, through adjustments and adequacy of their chemical, physical and metallurgical properties.

In this respect, it was developed the concept of increasing the use of agglomerates in ironmaking and, at the same time, decreasing the consumption of lump ores. As a consequence, the iron burdens became more stable and pushed productivity of the blast furnaces and DR reactors. This subject was quite discussed and published in world literature.

Lump ore, sinter and pellet constitute the basic elements for steel industry supply, according to the two routes of steelmaking:

Blast Furnace (BF) – Oxygen Converter (BOF) and

Direct Reduction (DR) - Electric Arc Furnace (EAF).

While in the first route the primary iron is obtained in liquid form in the blast furnace and feeding the BOF, in the second one, the iron is generated in solid form in the reactors of direct reduction to feed the EAF. These features, among others, are fundamental in defining the metallic burdens of these reduction furnaces. In mining the iron ore, measures are taken to ensure that the products have adequate properties to be used in such reactors. Generally, the typical metallic burdens of blast furnaces and direct reduction reactors are as follows:

Reactor	Typical Burden (particle size)
Blast Furnace	Lump, Sinter (4 to 50 mm), Pellet (8 to 18 mm)
Direct Reduction	Lump, Pellet (8 to 18 mm)

New reduction processes are under development being in different stages. Among them, the COREX is the most advanced, and there is already a large number of plants in industrial operation. Produces liquid pig iron and uses the pellets and lump as metallic burden. In turn, the FINEX process, which also produces liquid pig iron, uses iron ore fines as the feedstock (sinter feed and pellet feed).

The pelletizing technology was developed for recovering the iron ore fines of mining operations, generating high quality agglomerates to be used in the production of steel.

CHAPTER 4

RAW MATERIALS FOR PELLETIZING OF IRON ORE (1, 4, 5, 6)

The basic raw materials for pelletizing are iron ore, fluxes, solid fuels, and binders. Iron ore is the main component of the mixture to be pelletized. Iron is an abundant element in the earth's crust, representing on average 4.2% of the lithosphere. According to the United States Geological Survey (USGS), the iron ore resources in the world are in the order of 800 billion tons, with an average iron content of 28.8%. Typically, a deposit should contain at least 25% Fe to be economically recoverable, within the current state of the art. However, this cut-off content depends on other factors, such as geographical location and logistics.

More than 300 minerals contain iron, but only 5 are primary sources of iron:

- Hematite (Fe_2O_3),
- Magnetite (Fe_3O_4),
- Goethite/Limonite ($Fe_2O_3.nH_2O$),
- Siderite ($FeCO_3$),
- Pyrite (FeS_2).

Hematite, magnetite, and siderite are the most important iron minerals due to their occurrence in large deposits and in concentration that allows economic extraction. Iron ores originate from different geological environments, with prevailing rock formations of sedimentary, metamorphic, and igneous natures.

In Brazil, the most important iron mineral is hematite, and the main rock that contains it is the itabirite, as the formations of the iron quadrangle area in the state of Minas Gerais. Occurrence of magnetite is rare. Goethite/limonite, siderite and pyrite appear as contaminants of itabirite resources or in small deposits with no economic value.

Regardless of the type of iron ore, the final crystalline phase contained in the pellet is hematite (Fe_2O_3), in view of the high temperatures and the oxidizing environment inside the furnaces. The chemical reactions that occur inside this furnace are as follows:

Magnetite: $2Fe_3O_4 + \frac{1}{2} O_2 \longrightarrow 3Fe_2O_3 + \text{Heat}$

Goethite/limonite: $Fe_2O_3.nH_2O \longrightarrow Fe_2O_3 + nH_2O$

Siderite: $2FeCO_3 + \frac{1}{2} O_2 \longrightarrow Fe_2O_3 + 2CO_2$

Pyrite: $2FeS_2 + 11/2\ O_2 \longrightarrow Fe_2O_3 + 4SO_2$

It is worth noting that the reaction of oxidation of the magnetite is highly exothermic, which contributes to accelerate the burning of green pellets and for the reduction of consumption of thermal and electrical energy in pelletizing furnace. In turn, the hematite enters and exits the kiln with the same composition.

Different iron ores, due to the various genesis, have entirely distinct properties. As a result, have different behavior, not only in the mining, beneficiation, and agglomeration operations, but also in the reduction reactors. In the same way, the agglomerates, pellets and sinter, present distinct characteristics, and performance in ironmaking, depending on the type of ore used in their production.

The concentrates used for production of pellets for blast furnace and direct reduction present different characteristics, being a mixture of ores of different origins and aiming to meet the chemical, physical and metallurgical specifications of the pellets. Such concentrates have SiO2 + Al2O3 content ranging from 1.0 to 5.0%. In the case of direct reduction pellets, the iron ore should contain the lowest possible SiO2 + Al2O3 content because these acidic compounds are incorporated into the DRI / HBI, which feeds the electric arc furnaces, generating excessive volume of slag. For the blast furnace pellets, the content of these compounds will depend on the percentage of use in the furnace; whether as a complement of the burden or at the level close to 100% (all pellet burden). In turn, the Al2O3 content should be strictly controlled and at low concentrations, due to its influence on the characteristics of the slag (melting point and viscosity).

In relation to the other materials for the pelletizing of iron ore, they could be classified in accordance with their function as follows:

- Dosers of CaO: limestone and hydrated lime;
- Dosers of MgO: dolomite, dunite and magnesite;
- Binders: bentonite, hydrated lime (dual function), organic binders such as carboxy or hydroxy-methyl-cellulose, etc.;
- Solid fuels: anthracite, coke breeze and pet coke.

The limestone is the main source of CaO and the most demanded material in the market, there existing great reserves, and availability. The CaO corrects the acidity of the pellet giving it suitable physical and metallurgical properties for the use in ironmaking. The control of the quantity to be added to the pellet depends on the specified binary basicity index; this last characterized by the relation %CaO/%SiO2 in the fired pellet.

The hydrated lime has a dual function in the pelletizing. Inputs CaO and acts as a binder. Its use is quite limited due to the high costs involved in its production. Has the advantage of not adding significant amount of SiO2 to the pellet and presenting a high agglomeration power, when in production of green pellets. In Brazil, there is great availability of this material, and the demand is for the use in higher value-added industrial activities.

The dolomite, although it has a high percentage of CaO, is employed with the function of dosing MgO, this last compound also being of basic character. The MgO is very important to set the metallurgical properties of the pellets, particularly in relation to sticking in the direct reduction and to softening and melting in the case of blast furnace. Being a refractory compound, increases the melting point of the slag phase of pellets, requiring additional quantity of heat and electrical energy during the firing step of the pellets. The availability and reserves are also very large.

The Dunite is a source of MgO but contains high content of SiO2. For this reason, has limited application in pelletizing, being used in special cases. Adding silica to the pellet means increasing the volume of slag in reduction reactors, what is undesirable for obvious reasons.

The Magnesite is an excellent source of MgO. Chemically, is very pure. However, the availability and reserves are limited. This material is more used for production of refractories and products with higher added value. The use in the pelletizing plant is in very special cases.

The bentonite is the main binder used throughout the world. The availability and the reserves in Brazil are limited, however the offer in the international market is quite considerable. Although it is an excellent binder, has the disadvantage of adding to the pellets a significant amount of acid oxides (SiO2+Al2O3) and to a lesser extent some deleterious compounds such as Na2O and K2O. For this reason, there is a limitation of its application on the production of pellets for direct reduction, which requires high level of iron content and low percentage of SiO2+Al2O3.

The organic binder, usually derived from cellulose, has been developed over the past 30 years. Currently, there is a good number of suppliers. This material came in order to substitute the bentonite in the production of pellets for direct reduction. Being organic, this binder is completely burnt inside the pellets during the firing stage in the pelletizing furnace. Then, there is no pellet contamination with acid elements or any other deleterious compound. It is used alone or in combination with hydrated lime or even bentonite. Due to the complex process of producing this organic material, its price is high and its application in pelletizing of iron ore depends on economic feasibility factors. The first tests with organic binders were conducted by Vale SA, in the beginning of the 1980's, at its Pelletizing Complex located in Vitória ES, Brazil.

The anthracite is the most used solid fuel in pelletizing plant, in view of the wide availability on the world market. In Brazil, there is no production of this material, so it is acquired via imports. This fuel, added to the green pellets, has a role of great relevance. Accelerates the process of burning, replaces external fuel of higher cost, and improves the efficiency of the furnace, with a significant reduction in the consumption of heat and electricity. In addition, as it is burned inside the pellets, increases the porosity, with benefits for the reducibility and performance in the reactors of ironmaking. The levels of ash and sulfur content must be strictly controlled because they add undesirable elements and oxides to the pellet and degrade the environment (emission of SOx).

The coke breeze and petcoke have a high degree of purity, meaning low aggregation of undesirable compounds to the pellet. Their performance in the pelletizing plant is excellent, considering the high level of fixed carbon (Cfix), and low of ash content (CZ) and volatile matter (MV). In this condition, allow a smoother process control with benefits for the operation. However, the availability of such fuels for using in pelletizing is low and the prices are too high.

The use of one or other of such raw materials will depend on several factors, such as: specification of the pellets, availability and logistics for supply, price, performance in the process, etc.

In Brazil, where the iron ore is hematite, several of these raw materials are used in pelletizing activities. The main typical characteristics of them are shown in the table 4.1.

For a better understanding on the need of using these various additives, different types of pellets for blast furnace and direct reduction, traded in the seaborne market, are specified in the table 4.2.

Table 4.1 – Typical characteristics of additives used in pelletizing. (4)

TYPE / CHARACTERISTIC		SiO_2 %	Al_2O_3 %	CaO %	MgO %	LOI %	FINENESS (% - 0.074 mm)
FLUXES	➲ LIMESTONE	1.4	0.7	53.0	0.3	43.0	> 90
	➲ DOLOMITE	2.2	0.2	37.0	15.0	44.0	> 90
	➲ DUNITE	39.0	3.5	2.5	35.0	14.0	> 90
	➲ MAGNESITE	0.7	1.5	1.1	47.0	50.0	> 90
BINDERS	➲ BENTONITE	60.0	16.0	1.0	2.6	10.0	> 90
	➲ HYDRATED LIME	0.9	0.1	73.0	0.3	25.0	> 95
	➲ ORGANIC BINDER	Carboxy or Hydroxy-Methyl-Cellulose					-
SOLID FUELS	TYPE / CHARACTERISTIC	Fixed Carbon (%)	Volatile Matter (%)	Ash (%)	S (%)	Heat Value (kcal/kg)	-
	➲ ANTHRACITE	69.0	11.0	20.0	1.2	6600	> 90
	➲ PETCOKE	90.0	9.4	0.6	1.8	7890	> 90
	➲ COKE BREEZE	90.0	5.0	5.0	1.5	7800	> 90

LOI - loss on ignition

Table 4.2 – Typical chemical characteristics of pellets for using in blast furnace and direct reduction. (1, 4)

CHARACTERISTICS		BLAST FURNACE PELLETS			DIRECT REDUCTION PELLETS		
	TYPE	FLUXED	FLUXED/ HIGH SiO2	ACID	STANDARD	LOW SILICA	FLUXED
CHEMICAL ANALYSIS (%)	Fe	65.80	63.00	64.51	67.85	68.10	66.70
	SiO_2	2.35	4.00	5.00	1.20	1.00	1.40
	Al_2O_3	0.55	0.60	0.70	0.50	0.48	0.50
	CaO	2.60	4.00	1.00	0.70	0.65	1.60
	MgO	0.05	1.00	0.48	0.30	0.28	0.60
	P	0.030	0.027	0.030	0.025	0.025	0.025
	S	0.003	0.006	0.003	0.002	0.002	0.002
	Mn	0.10	0.23	0.13	0.12	0.12	0.12
	CaO/SiO_2	1.11	1.00	0.20	0.58	0.65	1.14

CHAPTER 5

ASPECTS OF THE CHEMICAL COMPOSITION OF IRON ORE PELLETS

The iron ore pellets are composed of oxides of important elements in order to be used in iron and steelmaking. Other compounds occur in lower concentrations and have deleterious effects to the processes and quality of steel. The most important compounds are: $Fe2O3$, FeO (traces), $SiO2$, $Al2O3$, CaO, MgO and MnO. Those considered deleterious are the followings which depend on the origin of the iron ore and type of additives used to produce the pellets: $P2O5$, CaS, $NaCl$, $Na2O$, $K2O$, etc.

Then, in terms of mass percentage, it could be written for the whole pellets that:

$Fe2O3 + SiO2 + Al2O3 + CaO + MgO + (FeO + P2O5 + CaS + Na2O + K2O + \text{etc.}) = 100\%$.

Or:

$Fe2O3 + SiO2 + Al2O3 + CaO + MgO + \Sigma\ (\text{other compounds}) = 100\%$ Eq. 5.1

The sum of the other compounds of the pellet is controlled and kept very low. As mentioned before, it depends on the origin of the iron ore. On the other hand, it is known from the chemistry that the hematite, $Fe2O3$, contains 70% of iron, Fe. Furthermore, most of Brazilian concentrates for pellet production is very pure and it can be estimated that Σ (other compounds) = 0.5 %. Thus, the equation Eq. 5.1 is written as follows:

$Fe/0.7 + SiO2 + Al2O3 + CaO + MgO = 100 – 0.5$.

Or by simplifying:

$Fe + (0.7).(SiO2 + Al2O3 + CaO + MgO) = 69.65$. Eq. 5.2

On the other hand, the basicity (B4) of a pellet is defined by:

$B4 = (CaO + MgO)/(SiO2 + Al2O3)$ or $(CaO + MgO) = (B4).(SiO2 + Al2O3)$.

Substituting ($CaO + MgO$) in equation Eq. 5.2 it comes:

$Fe + (0.7).(SiO2 + Al2O3) + (0.7).(B4).(SiO2+Al2O3) = 69.65$. By simplifying, it results:

$Fe + (SiO2 +Al2O3).(0.7).(1 + B4) = 69.65$ Eq. 5.3

Assigning the values 1, 3, and 5% to the sum of acid oxides ($SiO2 + Al2O3$), it could be made simulations of all possible chemical specifications in the production of iron ore pellets. Thus, the following equations are developed with the pertinent math simplifications:

For (SiO2 + Al2O3) = 1 .... Fe + (0.7).B4 = 68.95 Eq. 5.4

For (SiO2 + Al2O3) = 3 .... Fe + (2.1).B4 = 67.55 Eq. 5.5

For (SiO2 + Al2O3) = 5 .... Fe + (3.5).B4 = 66.15 Eq. 5.6

On the other hand, when designing and planning the production of a certain type of pellet, it is very important to know the relationship between the volume of slag (SiO2 + Al2O3 + CaO + MgO) and the correspondent iron content (Fe) of the pellet. This knowledge allows evaluating the impact of these oxides in the composition of the slag produced in iron and steelmaking.

In this context and considering the stoichiometry, it could be said that the relation slag/Fe is equal to:

RE = (SiO2 + Al2O3 + CaO + MgO)/Fe . Eq. 5.7

From the equation Eq. 5.2, one can write that:

Fe + (0.7).(SiO2 + Al2O3 + CaO + MgO) = 69.65. Dividing all terms by Fe, it results:

1 + (0.7).(SiO2 + Al2O3 + CaO + MgO)/Fe = 69,65/Fe

In accordance to the equation Eq. 5.7, it is possible to simplify as follows:

[1 + (0.7).RE] = 69.65/Fe . Or:

Fe = 69.65/[1 + (0.7).RE] . Eq. 5.8

Considering the value of Fe in equation Eq. 5.8 and inputting it in equation Eq. 5.3, it comes:

69,65/[1 + (0.7).RE] + (SiO2 +Al2O3).(0.7).(1 + B4) = 69.65 . Eq. 5.9

Then, it was obtained this equation, Eq. 5.9, with 3 variables RE, (SiO2 + Al2O3) and B4, allowing simulations and case studies, as in the previous example. Thus, assigning the values 1%, 3%, and 5% to the variable (SiO2+Al2O3) and doing the mathematical simplifications, the following 3 equations, Eq. 5.10; 5.11 and 5.12, can be written:

For $(SiO_2 + Al_2O_3) = 1$: $RE = (0.7).(1+B4)/(48.27-0.49.B4)$ in t of slag/t Fe.

Or finally: $RE = 700.(1+B4)/(48.27-0.49.B4)$ kg slag/t Fe . Eq. 5.10

For $(SiO_2 + Al_2O_3) = 3$: $RE = 2100.(1+B4)/(47.28-1.47.B4)$ Kg slag/t Fe . Eq. 5.11

For $(SiO_2 + Al_2O_3) = 5$: $RE = 3500.(1+B4)/(46.31-2.45.B4)$ kg slag/t Fe . Eq. 5.12

Figure 5.1 shows a graphical representation of equations Eq. 5.4, 5.5 and 5.6. It is clear that, using the presented methodology, it could be made other simulations with other variables.

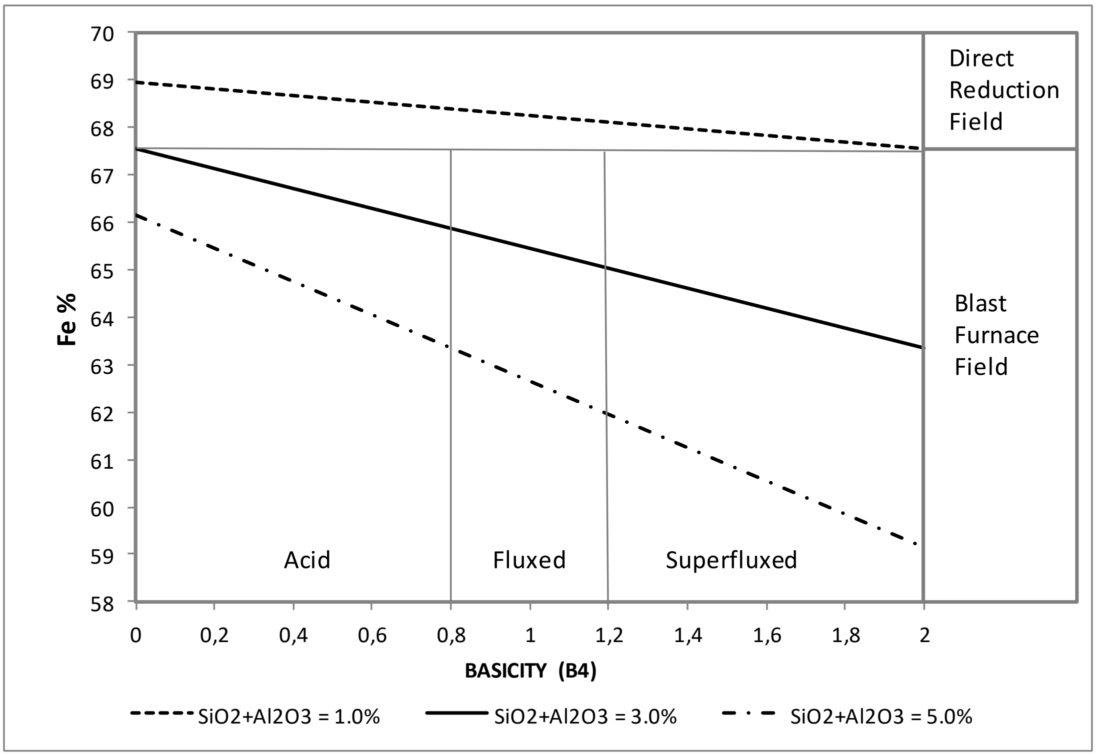


Figure 5.1 - Variation of iron content (Fe) with the basicity (B4) and with the percentage of acid slag $(SiO_2 + Al_2O_3)$ of the pellet. (1)

The field of pellets for direct reduction is quite restricted, compared to that of the pellets for blast furnace. The market for direct reduction requires the greatest iron content as possible, which implies use of pellet feed with a high degree of purity. Normally, the cut-off point between the two types is 67.5 % of Fe. In special cases, when there is no viability to concentrate the iron ore, or if one specifies a superfluxed pellet, the iron content of the pellet falls below 67.5%. The reason for this preference is that in the direct reduction all gangue goes to the final product, DRI/HBI, and this reduced material is sent to the electric arc furnaces (EAF), aiming at the production of steel. If containing low iron content (or high slag volume) the performance in the FEA is inadequate and there is an increase in production cost (higher consumptions and worse iron yield).

The field of pellets for blast furnace is much more ample. In this aspect, the pellet takes a great advantage over the sinter because it is possible to vary in a large extent its chemical composition thus obtaining good quality products.

In general, the pellets are classified as: acid, with basicity less than 0.8; fluxed, with intermediate basicity, in the range from 0.8 to 1.2; and superfluxed, where the quaternary basicity of the pellets is over 1.2.

In the pelletizing plant, increasing the basicity of the pellets implies an increase in operational costs and a decrease in iron content (drop in selling price, based on % Fe). To this end, it is necessary to make higher dosage of fluxes to the concentrate (CaO + MgO). Beyond the extra cost of such materials, there is a worse performance of the pelletizing furnace, with increase in energy consumption and decrease in productivity (t/m2/h). On the other hand, as this basic slag (CaO + MgO) can favor the performance of pellets in ironmaking, there is a value added to them which must be exploited by the producers (value-in-use).

Figure 5.2 shows another simulation, considering as variables the iron content (Fe) and the concentration of acid slag (SiO2 +Al2O3), for different values of the basicity (B4). It is also possible to draw other simulations, considering constant values of percentage of Al2O3, CaO and MgO.

Figure 5.3 shows a graphical representation of equations Eq. 5.10, 5.11 and 5.12. There are highlighted the impacts of the basicity (B4) and the volume of acid gangue (SiO2 + Al2O3) on the content of slag in the pellet (CaO + MgO + SiO2 + Al2O3), expressed in kg per t of Fe. In fact, this relationship (RE) indicates the contribution of slag that the pellets lead to the reactors of reduction and ironmaking, in relation to the desired element, the iron (Fe). It is clear that the steel industry would always operate with the lowest amounts of slag as possible because the production of slag leads to operational costs and higher equivalent CO2 emissions. However, the operation mode will be set taking in view that each steel plant has its standards concerning to costs, amount of slag generation and environmental emissions.

Under strictly technical point of view, the ideal pellet for use at 100% in the blast furnace (all pellet burden) would be one in which the slag could neutralize the acid slag of coke and act as de-S agent. Still, it should generate the lowest amount of total slag as possible, compatible with the smooth operation of the blast furnace. The BF area, shown in the figure, is quite limited and indicative of this condition.

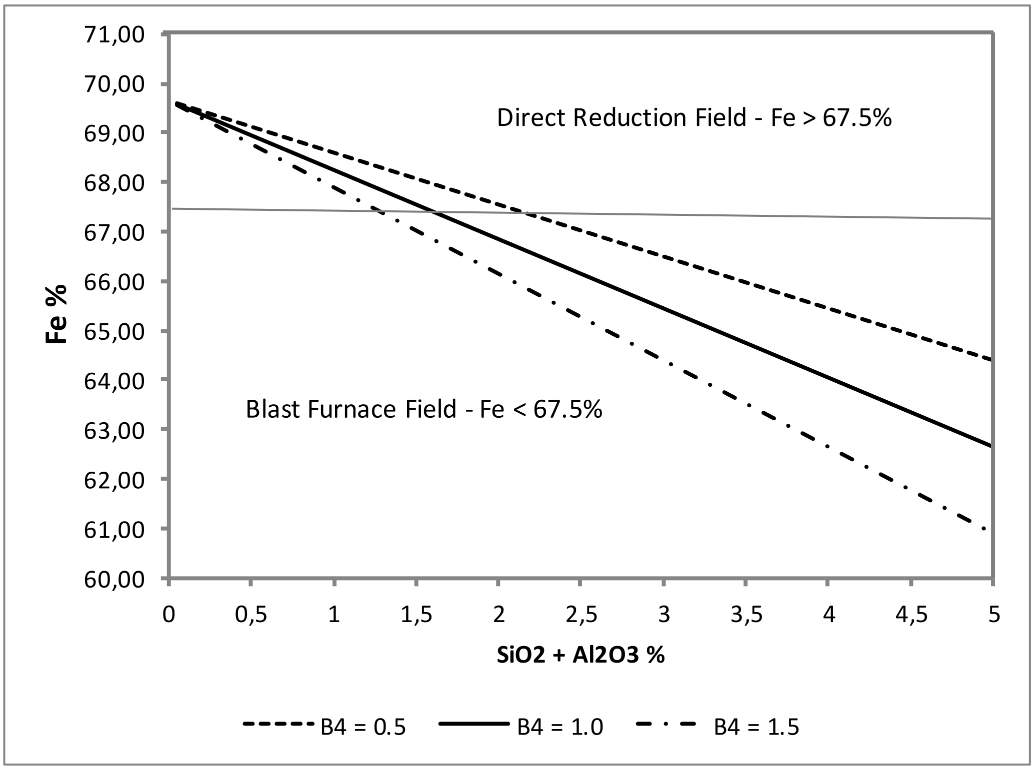


Figure 5.2 - Variation of iron content of the pellet (Fe) with the percentage of acid slag (SiO2 + Al2O3) and with the basicity (B4). (1)

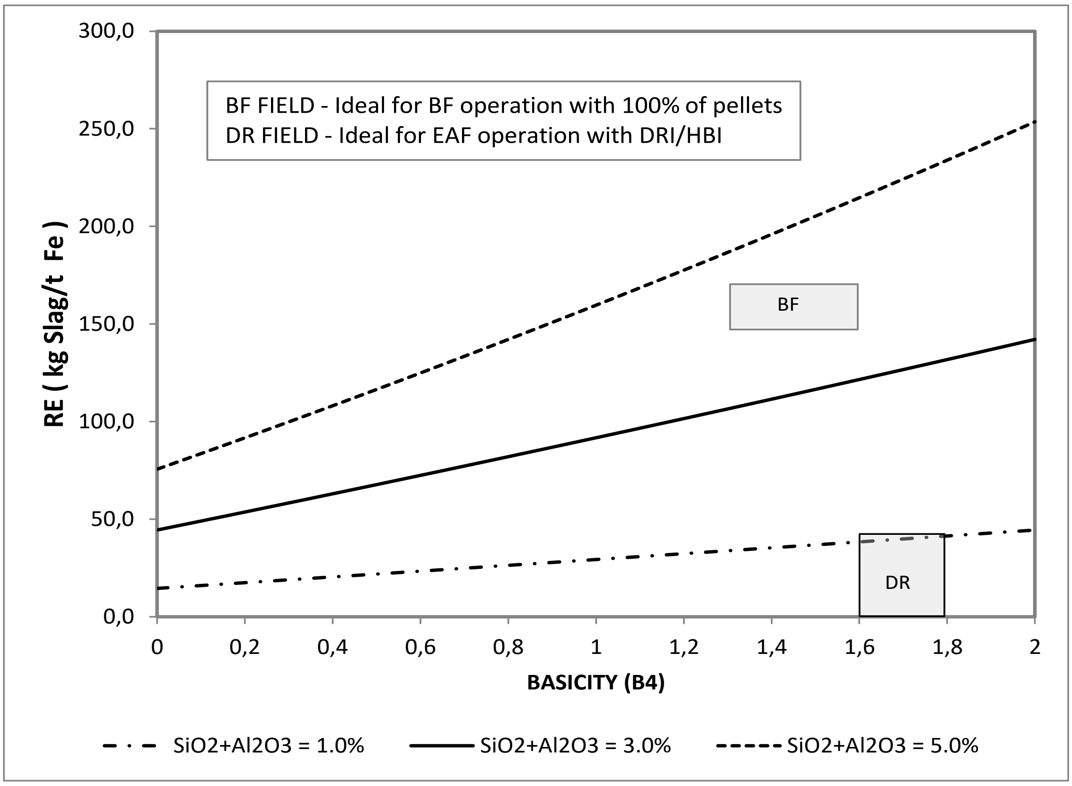


Figure 5.3 - Variation of the relation slag/Fe in the pellet (RE) with the basicity (B4) and with the percentage of acid slag (SiO2 + Al2O3). (1)

The technical reason that would justify this option is that the blast furnace would receive a material with the fluxes already calcined and in appropriate proportions to make a good slag in the blast furnace operation.

However, because of economic and logistics reasons, where freight is a decisive factor, this type of pellet of the BF field is not traded in the seaborne market.

It is more economical to transport iron (Fe) than calcined fluxes (CaO and MgO). From the point of view of the pelletizing plant, the production of pellets of BF field would have an additional and significant cost, in view of the higher consumptions and productivity loss. In the case of a specific kind of this product had commercial appeal, It should be examined the losses in pellet production and the added value to the product for setting the selling price (value-in-use).

The pellets for complement of the burden of blast furnaces, in combination with sinter and lump, have chemical composition which depends on many technical and economic factors. Normally, in this case, the binary basicity (CaO/SiO2) is in the range from 0.8 to 1.2. Although it is important for the blast furnace the addition of some MgO to the pellets, this is a problem for the producers of pellets. MgO increases the melting point of the slag phase of the pellet, interfering negatively in energy consumption and productivity of the pelletizing kiln. Also, it has deleterious effects on some physical and metallurgical properties.

In the case of direct reduction, it is important to consider the integrated operation of the direct reduction reactor with the electric arc furnace (EAF). This EAF is designed to melt scrap and produce steel. With the advent of the direct reduction, the EAF was adapted to work with DRI/HBI, replacing part or all of the scrap. With the use of HBI/DRI, the volume of slag grew, and the basicity was maintained at a high level for refining the steel. In a technical perspective only, it was indicated the DR area in the figure 5.3, as that ideal for DR pellets. As in the case of blast furnace pellets, this condition (DR) has many restrictions of technical-economical order. Some studies need to be developed in order to move forward toward the DR area, such as:

- Production of iron ore concentrate with very low content on acid gangue, (SiO2 + Al2O3) < 1.0 %;

- Production of DR pellets with high basicity levels without worsening the reducibility and metallization.

The MgO is also very important in the direct reduction route. In the direct reduction reactor, it prevents the sticking of pellets (clustering). In the EAF, minimizes the wear of the refractory lining (magnesian) in the region of contact with the molten slag.

In fact, the production cost of the pellets of the shaded DR area would be very high, or even prohibitive.

Finalizing the discussion on this subject, in figure 5.4 below is shown a graphical representation of RE x (SiO2 + Al2O3) x B4.

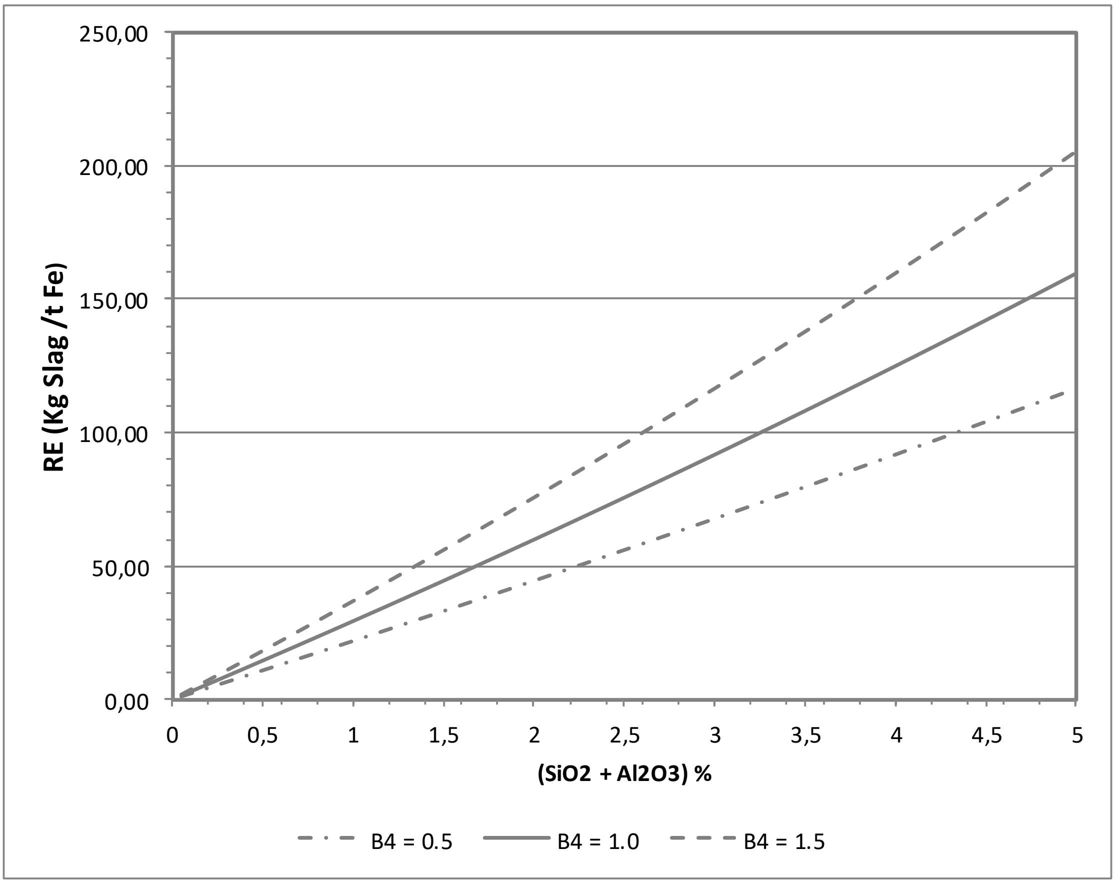


Figure 5.4 - Variation of the relation slag/Fe in the pellet (RE) with the percentage of acid slag (SiO2 + Al2O3) and with the basicity (B4). (1)

CHAPTER 6

MASS BALANCE IN PELLETIZING OF IRON ORES (1)

As shown before, in the pelletizing of iron ore are used different materials which have specific functions and typical chemical characteristics. The development of a generic mass balance concerning the pelletizing process is very important for planning and controlling the production of pellets, as well as for the development of projects and studies on technical and economic viability. As the number of variables involved is very large, it will be used the software SOLVER, from Microsoft Excel, to solve the system of linear equations, arising from the development.

Basically, all the iron ores and their mixtures, treated in a pelletizing plant, are transformed into Hematite-Fe2O3, bearing in mind the atmosphere of oxidizing gases and the high temperatures that exist within the pelletizing furnaces. The case of magnetite ores is special, considering that the mineral Fe3O4 is oxidized by the gases in the furnace with high heat release (exothermic reaction) and gain in weight gain, this last equivalent to the incorporation of oxygen in the structure of the ore. This process of oxidation occurs in accordance with the following chemical reaction:

2Fe3O4 + ½ O2 ----------> 3Fe2O3 and by stoichiometry can be written as follows.

463.1g + 16.0g ----------> 479.1g

Thus, in the oxidation of magnetite, there is a gain of mass equivalent to 16.0/463.1, or the same as 3.5%. So, when treating magnetite ore in a pelletizing furnace, there is a productivity gain of this order of magnitude, (productivity of iron oxide and not of metallic iron).

Due to this fact, it is important to emphasize that, in accordance with the standards and analytical methodology, the loss on ignition of the ore with prevalence of magnetite always presents negative value. This will not occur whether the magnetite ore contains considerable quantities of other materials and or mineral components with loss on ignition of positive value and at higher levels than the magnetite. Either way, for the calculation of the mass balance of a mixture of iron ore and additives for pelletizing, consideration should be given to the sign of the loss on ignition of the ore and constituents: *plus-sign (+)*, which means weight loss during burning of the pellets in the pelletizing kiln, and *negative-sign (-)*, meaning, in contrary, gain in mass.

Thus, it will be developed a general mass balance, which can be used for all types of iron ore, considering that in the pelletizing plant there is the full oxidation and the calcination of all mineral phases, arising as a result an iron agglomerate (pellet) with a matrix composed basically of hematite ($Fe2O3$). The balance sheet could also be used for calculations in the sintering process, making small adjustments in the parameters of the equations.

It is known that, in this process of sintering, occurs a slight reduction of hematite and generation of some amount of FeO, with loss of mass of oxygen ($O2$).

For the development of the mass balance sheet, it is important to make some observations as the following:

a) In order to have a system of linear equations, the solution less complex, it was considered that the initial weight of iron ore in the mixture to be pelletized (M) was known, a reference weight.

b) As it is considered in the chemical analysis of materials, in a general way, the mass balance was also developed on a dry basis. Thus, the moisture of the different materials was treated separately.

c) In the case of the binder, the dosage was considered in relation to the total weight of the mixture of materials under pelletizing, with a previously defined value (x %). It is important to remember that the hydrated lime, beside the function of binder, gives to the pellets considerable amount of CaO. In turn, the magnesian lime adds to the mixture important quantities of CaO and MgO. By the other side, the organic binders, due to their low dosages and loss on ignition close to 100%, don’t need to be considered because they don’t add waste or mineral components to the pellets.

d) In many pelletizing plants, especially in those where the ores are hematite, solid fuel is frequently used, as it was reported previously. The level of addition of that fuel is stipulated in advance and, in an indirect way, through the dosage of fixed carbon (Cfix %). As done in the case of the binder, the addition of solid fuel was considered in relation to the total weight of the pelletizing mixture.

e) In all pelletizing processes, dust and pellet chips are generated and lost by emissions to the environment or mainly recovered in the final screening stage of pellets. Some plants recycle these fines in the process. Other plants give another destination to this material. For the latter case, it is important to consider in the balance this apparent loss of material (the mass of and pellet chips which is removed from the circuit of pelletizing plant). In the development of the mass balance, it is considered a loss of pellet fines equal to z %, in relation to the net weight of produced pellets after the screening (free of such fines).

f) To extend the application of the mass balance, it was considered the possibility of doing the dosage of a special element or compound in the pellets, QO, unlike the traditional CaO, MgO and Cfix.

g) The chemical analyses of all materials involved in the mixture of pelletizing plants were considered as known data for the development of the mass balance. In the case of the solid fuel, beyond the proximate analysis (fixed carbon, volatile matter, and ash), it is of fundamental importance to know the complete chemical analysis of the ashes.

h) The chemical specifications of the target pellets were considered as known data and, in fact, constituted the main focus of the solution of the problem.

And what is it desired to calculate by the use of a mass balance?

i) The silica content that must have the iron ore, or mixture of different types of it, in order to get the silica content specified for the pellet to be produced.

j) The net weight of fired pellets, after screening and elimination of z% of pellet fines, the so-called pellet chips, unwanted by customers. If the quantity of such fines be insignificant, then z = 0.

k) The weight of the doser of CaO.

l) The weight of the doser of MgO.

m) The weight of the doser of special element or compound QO.

n) The weight of the doser of binder.

o) The weight of the doser of solid fuel.

After the determination of the values from items **(i)** to **(o)**, it is possible to calculate the percentage of other elements and compounds of the fired pellets, including estimate very accurately the content of total iron (Fe %). Also, can be defined the compositions of the mixtures of iron ore and additives, in dry and wet basis, as well as can be calculated the mass yields, costs, etc.

6.1 – Mass Balance Characterization

Let's consider the following data and basic variables of input and output (results) of a mass balance for a mixture of ingredients to be pelletized, as expressed in the tables 6.1, 6.2 and 6.3.

Table 6.1 – Input Data – Specification of the Fired Pellets

TYPE OF PELLETS	PELLET CHIPS	CHEMICAL ANALYSIS - TARGET %			
(Symbol)	(%)	SiO2	CaO	MgO	QO
XXXXXXX	z	s'p	c'p	g'p	q'p

It is important to note that these data on the chemical analysis of the target pellets, s'p, c'p, g'p and q'p, are input data of the system controlling the dosage of the elements or compounds.

Thus, if any of the elements and / or compounds, CaO, MgO and QO is not specified and demanded, the corresponding input content value must be zero (c'p and / or g'p and / or q'p = 0).

Table 6.2 – Input data – Characteristics of the Iron Ore and Additives

IRON ORE / ADDITIVES	BRAND OR SYMBOL	MIXTURE/ DOSAGE	CHEMICAL ANALYSIS (%)										MOISTURE
			Cfix	VM	ASH	SiO2	Al2O	CaO	MgO	QO	P	PF	%
Iron Ore	xxxxxxxx	Weight - M	-	-	-	-	am	cm	gm	qm	pm	fm	um
Doser of CaO	xxxxxxxx	-	-	-	-	sc	ac	cc	gc	qc	pc	fc	uc
Doser of MgO	xxxxxxxx	-	-	-	-	sg	ag	cg	gg	qg	pg	fg	ug
Doser of QO	xxxxxxxx	-	-	-	-	sq	aq	cq	gq	qq	pq	fq	uq
Binder	xxxxxxxx	x %	-	-	-	s a	aa	ca	ga	qa	pa	fa	ua
Solid Fuel	xxxxxxxx	y %	cfix	mv	cz	ss	as	cs	gs	qs	p s	fs	us

Remarks: 1 - The silica content of the iron ore is a variable to be determined, an output of the mass balance.
2 - Proximate Analysis of the solid fuel: Cfix = fixed carbon, VM = volatile matter and ASH. PF = loss on ignition.
3 - The loss on ignition of the solid fuel is calculated by knowing its proximate analysis: fs = (100-cz)%.
4 - As mentioned before, and in order to simplify the calculation, the weight of the iron ore is known and equals M.

Table 6.3 – Output Data – Mass Balance Results

CALCULATED VARIABLE	SYMBOL
Silica content of the iron ore	sm
Weight of the doser of CaO	C
Weight of the doser of MgO	G
Weight of the doser of QO	Q
Weight of binder	A
Weight of solid fuel	S
Net weight of fired pellets	P

Where:

- The sílica content of the iron ore (sm) is a variable to be determined; an output data and no input one. It is dependent on the amounts of silica, added by the additives to the pellet.
- Proximate analysis of the solid fuel: Cfix = fix carbon; VM = volatile matter e cz = ash. On the other side, PF = loss on ignition.
- The loss on ignition of the solid fuel, (fs), is determined as a function of its proximate analysis: fs = (100-cz) %.
- As mentioned before, for simplification, the iron ore weight is known, a reference value denoted by M.
- As P is the net weight of fired pellets (free of fines or chips), *the total weight of product with z% of fines/chips, P'*, can be written as follows:

P' = P + z.P/100, or

P' = P.(100+z)/100.

This total final weight of fired pellets (P'), which represents the sum of the net weight (P) plus the wheight of fines/chips, is an important variable for the development of the equations´ system involving the other variables. Included in such fines, one may consider other particulate, dust, and powder losses, some of them released to environment along the production process.

Considering such statements, the development of the mass balance is now possible to be done, through the use of a system of 7 linear equations with 7 variables to be calculated.

6.2 – <u>SiO2 content of Fired Pellet</u> (s'p)

The silica content of the pellet (%) is defined through the sum of the weighs of silica introduced by the different materials into the pelletizing mix, divided by the total final weight of pellets (fines included), P', being this result multiplied by 100. It´s important to mention that the weight of silica added by the solid fuel is relative to the ash content.

Thus, considering the variables characterized in the tables 6.1, 6.2 and 6.3, it´s possible to write that:

$$s'p = (sm.M/100 + sc.C/100 + sg.G/100 + sq.Q/100 + sa.A/100 + cz.S.ss/10.000).100 /[P.(100+z)/100]. \qquad \text{Eq. 6.1}$$

Simplifying, it results:

$$M.sm + sc.C + sg.G + sq.Q + sa.A + (cz.ss/100).S - [(100.s'p + z.s'p)/100].P = 0 \qquad \text{Eq. 6.2}$$

6.3 – CaO content of Fired Pellet (c'p)

The calculation of CaO content of fired pellet can be done similarly as developed for the determination of SiO2 content. Then, it results the following simplified equation:

$$cc.C + cg.G + cq.Q + ca.A + (cz.cs/100).S - [(100.c'p + z.c'p)/100].P = - M.cm \qquad \text{Eq. 6.3}$$

6.4 – MgO content of Fired Pellet (g'p)

Similarly, as before, it results this simplified equation for MgO content:

$$gc.C + gg.G + gq.Q + ga.A + (cz.gs/100).S - [(100.g'p + z.g'p)/100].P = - M.gm \qquad \text{Eq. 6.4}$$

6.5 – QO content of Fired Pellet (q'p)

Similarly, it results this simplified equation for QO content:

$$qc.C + qg.G + qq.Q + qa.A + (cz.qs/100).S - [(100.q'p + z.q'p)/100].P = - M.qm \qquad \text{Eq. 6.5}$$

6.6 – Equation relative to Binder Dosage (x %)

As a premise, the dosage of Binder was considered as done in relation to the total weight of the pelletizing mix, in dry basis. Then, it results the following equation:

$$x = A.100/(A+M+C+G+Q+S).$$

Developing, it follows:

$$x.C + x.G + x.Q + (x-100).A + x.S = - M.x \qquad \text{Eq. 6.6}$$

6.7 – Equation relative to Solid Fuel dosage (y %).

The dosage of solid fuel is normally done in terms of adding a defined fixed carbon amount, (y %), into the pelletizing mix. Thus, the total weight of the pelletizing mix, (A + M + C + G + Q + S), need to be considered as the reference in calculation.

By the other side, the weight of fixed carbon (Sfix) of the total amount of solid fuel (S) added to the pelletizing mix is given by:

$$Sfix = (cfix/100).S$$

Then, it results:

$$y = [(cfix.S/100)/(M+C+G+Q+A+S)].100$$

Simplifying, it follows:

$$y.C + y.G + y.Q + y.A + (y-cfix).S = - M.y \qquad \text{Eq. 6.7}$$

6.8 – Equation Relative to the Net Weight of Fired Pellet (P).

In this development, it's important to mention that the net weight of fired pellet (P) is free of fines/chips, these last in the amount of z %. The total weight of produced material, pellets + fines/chips, resulted from processing the pelletizing mix, was denoted as P´. Then, such total weight P' is the output of pelletizing furnace, because of firing the raw mixture contained into the green balls. Under high temperatures, inside the kiln, the raw materials are calcined and lose weight (loss on ignition). As mentioned before, the exception are the magnetite ores, which instead of losing they gain weight due to the oxidation process inside the pelletizing furnace.

Thus, the final total weight of fired pellets (P') is equal to the sum of the weights of calcined raw materials, in other words, equal to the sum of original weights less the corresponding losses on ignition.

Before, it was seen that:

$$P' = P.(100+z)/100$$

By the other side, considering the losses of ignition of the raw materials, one can write that P' is equal to:

$$P' = M.(100-fm)/100 + C.(100-fc)/100 + G.(100-fg)/100 + Q.(100-fq)/100 +$$
$$+ A.(100-fa)/100 + S.cz/100$$

Concerning with the solid fuel, it's noticeable that is burned inside the pelletizing kiln, remaining into the pellets only the ashes.

Making equal the two expressions in P´ and simplifying, it results:

$$(100-fc).C + (100-fg).G + (100-fq).Q + (100-fa).A + cz.S – (100+z).P =$$
$$= - M.(100-fm) \qquad \text{Eq. 6.8}$$

6.9 – The System of 7 Equations which solves the Mass Balance Problem:

The seven linear equations, Eq. 6.2 to Eq. 6.8, involving the 7 variables of the mass balance, **sm, C, G, Q, A, S e P**, (see table 6.3), can be listed as follows:

$$M.\mathbf{sm} + sc.\mathbf{C} + sg.\mathbf{G} + sq.\mathbf{Q} + sa.\mathbf{A} + (cz.ss/100).\mathbf{S} - [(100.s'p + z.s'p)/100].\mathbf{P} = 0 \qquad \text{Eq. 6.2}$$

$$cc.\mathbf{C} + cg.\mathbf{G} + cq.\mathbf{Q} + ca.\mathbf{A} + (cz.cs/100).\mathbf{S} - [(100.c'p + z.c'p)/100].\mathbf{P} = - M.cm \qquad \text{Eq. 6.3}$$

$$gc.\mathbf{C} + gg.\mathbf{G} + gq.\mathbf{Q} + ga.\mathbf{A} + (cz.gs/100).\mathbf{S} - [(100.g'p + z.g'p)/100].\mathbf{P} = - M.gm \qquad \text{Eq. 6.4}$$

$$qc.\mathbf{C} + qg.\mathbf{G} + qq.\mathbf{Q} + qa.\mathbf{A} + (cz.qs/100).\mathbf{S} - [(100.q'p + z.q'p)/100].\mathbf{P} = - M.qm \qquad \text{Eq. 6.5}$$

$$x.\mathbf{C} + x.\mathbf{G} + x.\mathbf{Q} + (x-100).\mathbf{A} + x.\mathbf{S} = - M.x \qquad \text{Eq. 6.6}$$

$$y.\mathbf{C} + y.\mathbf{G} + y.\mathbf{Q} + y.\mathbf{A} + (y-cfix).\mathbf{S} = - M.y \qquad \text{Eq. 6.7}$$

$$(100-fc).\mathbf{C} + (100-fg).\mathbf{G} + (100-fq).\mathbf{Q} + (100-fa).\mathbf{A} + cz.\mathbf{S} - (100+z).\mathbf{P} = - M.(100-fm) \qquad \text{Eq. 6.8}$$

This system makes possible the determination of the 7 variables, bringing a solution for the mass balance, where:

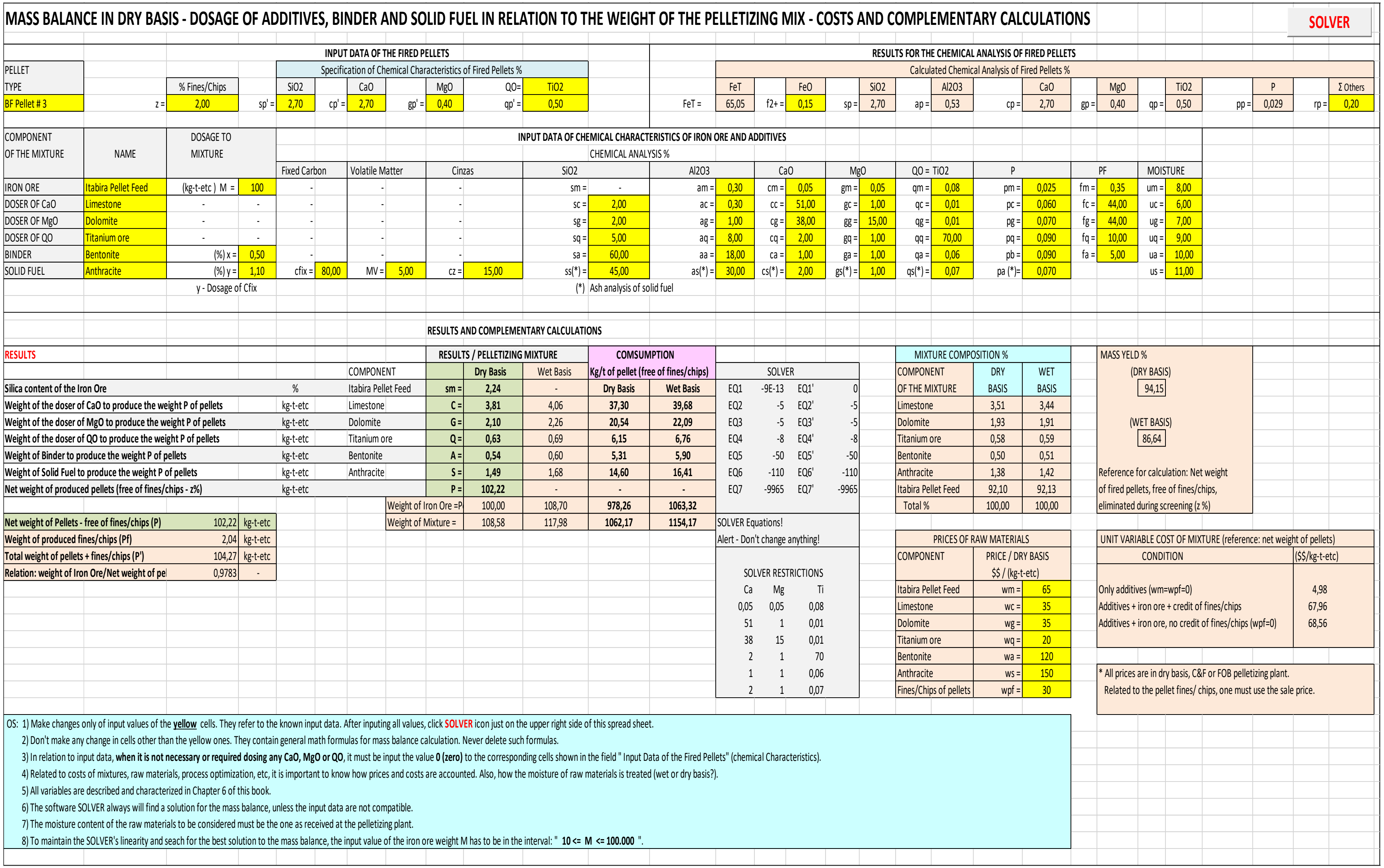
- **sm** - silica content of the iron ore % ;
- **C** - weight of the doser of CaO;
- **G** - weight of the doser of MgO;
- **Q** - weight of the doser of QO;
- **A** - weight of binder;
- **S** - weight of solid fuel;
- **P** - net weight of produced pellets, free of fines/chips.

The classic mathematical method used to solve a system of linear equations like this involves the use of matrices and determinants. These calculations are really complexing and take too much time. Then, a better option was to make use of the SOLVER tool of the Microsoft's Excel software to solve the problem.

It was prepared an Excel Spreadsheet with all the input data and the 7 referred linear equations, developing a software for calculation and or mass balance simulation. Through this system, it´s easily possible to develop feasibility studies and prepare strategic planning on raw materials for pelletizing. Such spreadsheet is available in electronic form in author's files. Someone with knowledge of Excel application can take the 7 equations and solve the system with the use of solver tool.

A general view of the spreadsheet can be seen in table 6.4.

Table 6.4 – Excel Spreadsheet for calculation of pelletizing mass balance with the use of SOLVER tool.



MASS BALANCE IN DRY BASIS - DOSAGE OF ADDITIVES, BINDER AND SOLID FUEL IN RELATION TO THE WEIGHT OF THE PELLETIZING MIX - COSTS AND COMPLEMENTARY CALCULATIONS

SOLVER

INPUT DATA OF THE FIRED PELLETS

Specification of Chemical Characteristics of Fired Pellets %

PELLET TYPE		% Fines/Chips		SiO2		CaO		MgO	QO=	TiO2
BF Pellet # 3	z =	2,00	sp' =	2,70	cp' =	2,70	gp' =	0,40	qp' =	0,50

RESULTS FOR THE CHEMICAL ANALYSIS OF FIRED PELLETS

Calculated Chemical Analysis of Fired Pellets %

	FeT		FeO		SiO2		Al2O3		CaO		MgO		TiO2		P		Σ Others
FeT =	65,05	f2+ =	0,15	sp =	2,70	ap =	0,53	cp =	2,70	gp =	0,40	qp =	0,50	pp =	0,029	rp =	0,20

INPUT DATA OF CHEMICAL CHARACTERISTICS OF IRON ORE AND ADDITIVES

CHEMICAL ANALYSIS %

COMPONENT OF THE MIXTURE	NAME	DOSAGE TO MIXTURE		Fixed Carbon		Volatile Matter		Cinzas		SiO2		Al2O3		CaO		MgO		QO = TiO2		P		PF		MOISTURE	
IRON ORE	Itabira Pellet Feed	(kg-t-etc) M =	100	-		-		-		sm =	-	am =	0,30	cm =	0,05	gm =	0,05	qm =	0,08	pm =	0,025	fm =	0,35	um =	8,00
DOSER OF CaO	Limestone	-	-	-		-		-		sc =	2,00	ac =	0,30	cc =	51,00	gc =	1,00	qc =	0,01	pc =	0,060	fc =	44,00	uc =	6,00
DOSER OF MgO	Dolomite	-	-	-		-		-		sg =	2,00	ag =	1,00	cg =	38,00	gg =	15,00	qg =	0,01	pg =	0,070	fg =	44,00	ug =	7,00
DOSER OF QO	Titanium ore	-	-	-		-		-		sq =	5,00	aq =	8,00	cq =	2,00	gq =	1,00	qq =	70,00	pq =	0,090	fq =	10,00	uq =	9,00
BINDER	Bentonite	(%) x =	0,50	-		-		-		sa =	60,00	aa =	18,00	ca =	1,00	ga =	1,00	qa =	0,06	pb =	0,090	fa =	5,00	ua =	10,00
SOLID FUEL	Anthracite	(%) y =	1,10	cfix =	80,00	MV =	5,00	cz =	15,00	ss(*) =	45,00	as(*) =	30,00	cs(*) =	2,00	gs(*) =	1,00	qs(*) =	0,07	pa (*)=	0,070			us =	11,00

y - Dosage of Cfix

(*) Ash analysis of solid fuel

RESULTS AND COMPLEMENTARY CALCULATIONS

RESULTS

		COMPONENT		RESULTS / PELLETIZING MIXTURE Dry Basis	Wet Basis	COMSUMPTION Kg/t of pellet (free of fines/chips) Dry Basis	Wet Basis
Silica content of the Iron Ore	%	Itabira Pellet Feed	**sm =**	**2,24**	-		
Weight of the doser of CaO to produce the weight P of pellets	kg-t-etc	Limestone	**C =**	**3,81**	4,06	**37,30**	**39,68**
Weight of the doser of MgO to produce the weight P of pellets	kg-t-etc	Dolomite	**G =**	**2,10**	2,26	**20,54**	**22,09**
Weight of the doser of QO to produce the weight P of pellets	kg-t-etc	Titanium ore	**Q =**	**0,63**	0,69	**6,15**	**6,76**
Weight of Binder to produce the weight P of pellets	kg-t-etc	Bentonite	**A =**	**0,54**	0,60	**5,31**	**5,90**
Weight of Solid Fuel to produce the weight P of pellets	kg-t-etc	Anthracite	**S =**	**1,49**	1,68	**14,60**	**16,41**
Net weight of produced pellets (free of fines/chips - z%)	kg-t-etc		**P =**	**102,22**	-	-	-
		Weight of Iron Ore =P		100,00	108,70	**978,26**	**1063,32**
		Weight of Mixture =		108,58	117,98	**1062,17**	**1154,17**

Net weight of Pellets - free of fines/chips (P)	102,22	kg-t-etc
Weight of produced fines/chips (Pf)	2,04	kg-t-etc
Total weight of pellets + fines/chips (P')	104,27	kg-t-etc
Relation: weight of Iron Ore/Net weight of pel	0,9783	-

SOLVER

EQ1	-9E-13	EQ1'	0
EQ2	-5	EQ2'	-5
EQ3	-5	EQ3'	-5
EQ4	-8	EQ4'	-8
EQ5	-50	EQ5'	-50
EQ6	-110	EQ6'	-110
EQ7	-9965	EQ7'	-9965

SOLVER Equations!
Alert - Don't change anything!

SOLVER RESTRICTIONS

Ca	Mg	Ti
0,05	0,05	0,08
51	1	0,01
38	15	0,01
2	1	70
1	1	0,06
2	1	0,07

MIXTURE COMPOSITION %

COMPONENT OF THE MIXTURE	DRY BASIS	WET BASIS
Limestone	3,51	3,44
Dolomite	1,93	1,91
Titanium ore	0,58	0,59
Bentonite	0,50	0,51
Anthracite	1,38	1,42
Itabira Pellet Feed	92,10	92,13
Total %	100,00	100,00

MASS YELD %

(DRY BASIS)
94,15

(WET BASIS)
86,64

Reference for calculation: Net weight of fired pellets, free of fines/chips, eliminated during screening (z %)

PRICES OF RAW MATERIALS

COMPONENT	PRICE / DRY BASIS $$ / (kg-t-etc)	
Itabira Pellet Feed	wm =	65
Limestone	wc =	35
Dolomite	wg =	35
Titanium ore	wq =	20
Bentonite	wa =	120
Anthracite	ws =	150
Fines/Chips of pellets	wpf =	30

UNIT VARIABLE COST OF MIXTURE (reference: net weight of pellets)

CONDITION	($$/kg-t-etc)
Only additives (wm=wpf=0)	4,98
Additives + iron ore + credit of fines/chips	67,96
Additives + iron ore, no credit of fines/chips (wpf=0)	68,56

* All prices are in dry basis, C&F or FOB pelletizing plant.
Related to the pellet fines/ chips, one must use the sale price.

OS: 1) Make changes only of input values of the **yellow** cells. They refer to the known input data. After inputing all values, click **SOLVER** icon just on the upper right side of this spread sheet.
2) Don't make any change in cells other than the yellow ones. They contain general math formulas for mass balance calculation. Never delete such formulas.
3) In relation to input data, **when it is not necessary or required dosing any CaO, MgO or QO**, it must be input the value **0 (zero)** to the corresponding cells shown in the field " Input Data of the Fired Pellets" (chemical Characteristics).
4) Related to costs of mixtures, raw materials, process optimization, etc, it is important to know how prices and costs are accounted. Also, how the moisture of raw materials is treated (wet or dry basis?).
5) All variables are described and characterized in Chapter 6 of this book.
6) The software SOLVER always will find a solution for the mass balance, unless the input data are not compatible.
7) The moisture content of the raw materials to be considered must be the one as received at the pelletizing plant.
8) To maintain the SOLVER's linearity and seach for the best solution to the mass balance, the input value of the iron ore weight M has to be in the interval: " **10 <= M <= 100.000** ".

6.10 – Complementary Calculations of the Mass Balance.

These complementary calculations, derived from the resolution of the mass balance, enable a broader analysis and evaluation of the iron ore pellet production process. Thus, these calculations were also introduced in the Excel spreadsheet, as shown in table 6.4.

(a) *weight of pellet fines (Pf).*

According to the previous definition, the percentage of these fines (z %) was referred to the final weight of pellets, free of these fines (P).

It is evident that in pelletizing plants where there is no sieving of the burned pellets, z = Pf = 0. Thus, it can be written that:

$Pf = z.P/100$ Eq. 6.9

(b) *total weight of pellets produced (P')*.

According to the previous definitions, the total weight of pellets produced, at the exit of the furnace, would be:

$P' = P + Pf$. Replacing the value of Pf, obtained in the equation Eq. 6.9 and simply put, it comes:

$P' = (100+z). P/100$ Eq. 6.10

(c) *the composition of the mixture to be pelletized on a dry basis (%)*.

The weights of the components of the mixture to be pelletized were calculated by the mass balance according to the following nomenclature:

- M - weight of iron ore (any value - input data)
- C - weight of CaO dosing material;
- G - weight of MgO dosing material;
- Q - weight of the QO dosing material;
- A - weight of the binder;
- S - weight of solid fuel.

These variables were calculated in relation to the weight M, stipulated for iron ore and considered as input data for the mass balance. Thus, the total weight of the mixture to be pelletized (Pm) can be written as:

- $Pm = M+C+G+Q+A+S$ Eq. 6.11

From these considerations, one can deduce the percentage composition of the mixture on a dry basis by writing the following equations:

- $Ms\ \% = 100.M/Pm$ - percentage of iron ore, on a dry basis; Eq. 6.12

- Cs % = 100.C/Pm - percentage of CaO feeder, on a dry basis; Eq. 6.13
- Gs % = 100.G/Pm - percentage of MgO feeder, on a dry basis; Eq. 6.14
- Qs % = 100.Q/Pm - percentage of QO feeder, on a dry basis; Eq. 6.15
- The % = 100.A/Pm - percentage of binder, on a dry basis; Eq. 6.16
- Ss % = 100.S/Pm - percentage of solid fuel, on a dry basis. Eq. 6.17

(d) *the composition of the mixture to be pelletized on a wet basis (%)*.

The materials and components of the mixtures to be pelletized, iron ore and additives, are received in the companies, containing some moisture content (see table 6.2). For the definition of the mixing pile, or even the weights of additives to be demanded, it is important to know the wet weights of these materials and, therefore, the composition of the mixture on a wet basis.

Knowing the dry weights of these components, defined or calculated by the mass balance, M, C, G, Q, A and S and the respective moisture contents, um, uc, ug, uq, ua and us, the wet weights can be calculated, through the generic equation of definition of moisture content (u%):

u % = 100.(Pu – Ps)/Pu

Where: u – moisture content (%), Pu – wet weight and Ps – dry weight.

Unfolding the equation comes:

u.Pu = 100.Pu – 100.Ps or, Pu. (100-u) = 100.Ps, resulting,

Pu = 100.Ps/(100-u) Eq. 6.18

Using the equation Eq. 6.18 and the data of the components of the mixture already mentioned (dry weights and moisture contents), the corresponding wet weights may be calculated by analogy, resulting in:

- M' = 100.M/(100-um) - wet weight of iron ore; Eq. 6.19
- C' = 100.C/(100-uc) - wet weight of the CaO dosing material; Eq. 6.20
- G' = 100.G/(100-ug) - wet weight of the MgO dosing material; Eq. 6.21
- Q' = 100.Q/(100-uq) - wet weight of the QO dosing material; Eq. 6.22
- A' = 100.A/(100-ua) - wet weight of the binder; Eq. 6.23
- S' = 100.S/(100-us) - wet weight of solid fuel. Eq. 6.24

The total wet weight of the mixture to be pelletized (Pm') is given by:

Pm’ = M’+C’+G’+Q’+A’+S’ Eq. 6.25

Then, as done for the dry components, one can calculate the percentage composition of the mixture on a wet basis, knowing the wet weights, calculated above. The knowledge of wet mixing greatly assists the planning of ore piles for pelletizing. From these considerations, then, follows:

- Mu' % = 100.M'/Pm' - % iron ore, on a wet basis; Eq. 6.26
- Cu' % = 100.C'/Pm' - % of CaO dosing material, wet base; Eq. 6.27
- Gu' % = 100.G'/Pm' - % of MgO dosing material, wet base; Eq. 6.28
- Qu' % = 100.Q'/Pm' - % of QO dosing material, wet base; Eq. 6.29
- Au' % = 100.A'/Pm' - % binder, on a wet basis; Eq. 6.30
- Su' % = 100.S'/Pm' - % solid fuel, on a wet basis. Eq. 6.31

(e*)* *mass pelletizing yields (%)*.

During the burning of raw pellets in pelletizing furnaces, there is a considerable loss of mass, associated with several factors, such as:

- Evaporation of moisture from raw pellets (loss of H2O),
- Dehydration of compounds with water content in the crystalline state (loss of H2O),
- Calcination of calcareous components (loss of CO2) and other components of the mixture (extinction of volatiles),
- Combustion of solid fuel added to the pelletizing mixture (loss of CO2, H2O, SO2),
- Other losses in the form of dust, fines from sieving the burnt pellets, etc.

In this aspect, it is important to remember that, in the case of pelletizing of magnetite ore, there is a weight gain of 3.5%, associated with the oxidation of the magnetite present in the pelletizing mixture, as already mentioned in chapter 6. In this case, the total mass loss is less than that of hematite ore in pelletizing.

The mass yield can be defined as the percentage ratio between the net weight of pellets produced, free of fines, P, already calculated by the mass balance and the total weight of the mixture to be pelletized. Thus, mass yields can be calculated on the dry basis, Rm, and on the wet basis, Rm'. For the representation of the 2 equations that express these mass yields, one must consider the expressions Eq. 6.11 and Eq. 6.25, which show the dry and wet weights of the pelletizing mixtures, respectively: Pm and Pm'.

Within these considerations, it can be written that:

Rm % = 100. P/Pm - Mass Yield, on the dry basis, and Eq. 6.32

Rm' % = 100.P/Pm' – Mass yield, on wet basis. Eq. 6.33

Mass yields shall be calculated with the net weight of pellets (P), without fines, which represents the amount of marketable product. In the case of pelletizing plants in which there is no sieving of the burned pellets, and in which other mass losses are irrelevant (powders, dust, etc.), " z = 0 " must be made in the input data table of the mass balance (table 6.1). And in this case, P = P', as shown in the equation Eq. 6.10.

(f) *specific consumption of iron ore and additives*.

When studying the mass balance of pelletizing, it is very important to determine the specific intakes of iron ore and additives, whether on a dry or wet basis. These consumptions should be referenced to the net weight of pellets produced (P), free of fines and powders, known and calculated by the system of equations. Typically, these consumptions are calculated and expressed by the unit: kgf of ore or additive / t of pellets.

Considering that the unit of weight of the mass balance, for all materials, is the ton (t), it is necessary to make the conversion t → kgf for the consumption of ore and additives (factor = 1,000).

- *Calculation of Specific Consumptions on a dry basis*.

Previously, it was evidenced that the weights of the components of the mixture to be pelletized were calculated by the mass balance, according to the following nomenclature:

- M - *dry weight* of iron ore (any value - input data);
- C - *dry weight* of CaO dosing material;
- G - *dry weight* of MgO dosing material;
- Q - *dry weight* of the QO dosing material;
- A - *dry weight* of the binder;
- S - *dry weight* of solid fuel.

Also, the final net weight of pellets produced was calculated, expressed by the variable P. Considering the conversion t → kgf for the weights of ore and additives, the following specific consumptions result on a dry basis and expressed by kgf/t:

- EMF = 1000.M/P - Specific consumption of iron ore; Eq. 6.34
- CEC = 1000.C/P - Specific consumption of CaO dosing material; Eq. 6.35
- CEG = 1000.G/P - Specific consumption of MgO dosing material; Eq. 6.36
- CEQ = 1000.Q/P - Specific consumption of QO dosing material; Eq. 6.37
- CEA = 1000.A/P - Specific binder consumption; Eq. 6.38
- CES = 1000.S/P - Specific solid fuel consumption. Eq. 6.39

- *Calculation of Specific Consumptions on a wet basis*.

Similarly, to the specific consumptions on a dry basis, the wet equivalents will be calculated, considering the wet weights of ore and additives, previously determined in item 6.10 – d. These wet weights were expressed by:

- M' = 100.M/(100-um) - wet weight of iron ore;
- C' = 100.C/(100-uc) - wet weight of the CaO dosing material;
- G' = 100.G/(100-ug) - wet weight of the MgO dosing material;
- Q' = 100.Q/(100-uq) - wet weight of the QO dosing material;
- A' = 100.A/(100-ua) - wet weight of the binder;
- S' = 100.S/(100-us) - wet weight of solid fuel.

It follows that the specific consumptions, wet basis, in kgf/t, would be calculated by:

- CEM' = 1000.M'/P - Specific consumption of iron ore; Eq. 6.40
- CEC' = 1000.C'/P - Specific consumption of CaO feeder; Eq. 6.41
- CEG' = 1000.G'/P - Specific intake of MgO feeder; Eq. 6.42
- CEQ' = 1000.Q'/P - Specific consumption of QO feeder; Eq. 6.43
- CEA' = 1000.A'/P - Specific binder consumption; Eq. 6.44
- CES' = 1000.S'/P - Specific solid fuel consumption. Eq. 6.45

g) *Evaluation of the unit variable cost of the pelletizing mixture*.

For the evaluation of the variable cost of the mixture to be pelletized, calculated by the mass balance, it is necessary to know the prices of the various components on a C&F basis - Pelletizing Plant. The dry base must be considered in the calculations and the appropriate conversion must be carried out if any component of the mixture is acquired on a wet basis. Bentonite, for example, because it is hygroscopic characteristic, is always acquired with a moisture content in the range of 10 to 12%. Solid fuel, limestone and other additives are also purchased with a certain moisture content. Hence, the need for knowledge by the areas of control and planning of the form of acquisition of the various materials of the mixture. In the marketing of these materials, including iron ore, the dry base is preferentially adopted. After all, consumers do not want to pay for water freight (humidity) or establish additional controls on receipt.

Another important point in this analysis of the cost of the pelletizing mixture impacted by the sieving fines of the burned pellets (< 5 mm). Such fines have the same chemical composition as pellets and carry part of the raw materials of the pelletizing mixture. Some companies recirculate them in the process, such as feeding the grinding mills. However, they have very high hardness and reduce the efficiency of grinding. Others prefer to sell them as sinter feed or send them to other areas of the company at agreed transfer prices. In these last two conditions, a credit for the mixing/balling section can be considered, with consequent reduction of the variable cost of the pelletizing mixture.

For the evaluation of the variable unit cost of the mixture, it is necessary to take the weights of the components on a dry basis, already calculated by the mass balance, M, C, G, Q, A and S, in addition to the weights of burnt pellets, free of fines, P, and of the sieving fines, Pf. Also, it is necessary to consider the purchase prices of the components of the mixture, on a dry basis, and the price of transfer and/or sale of the screening fines (credit). Table 6.5 shows these known data in a consolidated form.

Table 6.5 – Weights and unit prices of the components of the mixture and pellet fines (Wpf is the selling or transfer price of the sieving pellet fines).

MATERIAL (dry basis)	WEIGHT	UNIT PRICE
Iron ore	M	Wm
Doser of CaO	C	Wc
Doser of MgO	G	Wg
Doser of QO	Q	Wq
Binder	A	Wa
Solid fuel	S	Ws
Pellets (without fines)	P	-
Pellet fines (< 5 mm)	Pf	Wpf

Considering the values of table 6.5, it can be said that the total cost of the materials components of the mixture, CT, for the manufacture of the pellet specified and calculated by the mass balance (see Excel spreadsheet – table 6.4), could be written by:

$$CT = M.Wm + C.Wc + G.Wg + Q.Wq + A.Wa + S.Ws \qquad \text{Eq. 6.46}$$

However, if the fines of burnt pellets are sold, it can be considered a credit to the mixing/balling section, reducing the total cost of the mixture, CT', which could be expressed by:

$$CT' = M.Wm + C.Wc + G.Wg + Q.Wq + A.Wa + S.Ws - Pf.Wpf \qquad \text{Eq. 6.47}$$

Considering the equations of CT and CT', the variable unit costs of the mixture to be pelletized, CTu and CTu', added to the Excel spreadsheet of the mass balance, would be given by:

$$CTu = (M.Wm + C.Wc + G.Wg + Q.Wq + A.Wa + S.Ws)/P, \text{ (without credit of pellet fines),} \qquad \text{Eq. 6.48}$$

And $CTu' = (M.Wm + C.Wc + G.Wg + Q.Wq + A.Wa + S.Ws - Pf.Wpf)/P$, with fine credit. Eq. 6.49

If it were desired to calculate only the unit cost of the additives dosed to iron ore, CAu, the following equation could be used:

$$CAu = (C.Wc + G.Wg + Q.Wq + A.Wa + S.Ws)/P \qquad \text{Eq. 6.50}$$

(h) *estimation of the iron content and final chemical analysis of the burnt pellet.*

Consider the input data of the mass balance, shown in tables 6.1 and 6.2. They were used to calculate the 7 fundamental variables, broken down in table 6.3. As already mentioned, the data of the desired chemical analysis for the burned pellet cp', gp' and qp' guide the calculations of the system and the *dosages of the compounds CaO, MgO and QO,* respectively. When one does not want to dose one of these compounds, one should make the corresponding value equal to zero (*cp' and or gp' and or qp' = 0*). In the latter case, the content of the corresponding compound in the burned pellet will be residual, as a result of the concentrations of this compound in the components of the mixture to be pelletized.

The FeO content of the burned pellet (f2+), broken down in table 6.4, is usually low and could be disregarded. Industrially, the atmosphere prevailing in pelletizing furnaces is very oxidizing, so that the FeO content assumes values of the order of 0.5%. This is also an *input data of the mass balance*, and one must take a historical value, characteristic of the type of pellets being analyzed. In the case of sintering, where there is partial reduction of iron ore, the FeO content in the sinter is much higher, being in the range of 5 to 10%. If someone want to use this mass balance for sintering have to make small adjustments to the equations involving the final weight of the product produced (P), since the reduction is accompanied by weight loss (oxygen loss: $Fe_2O_3 \rightarrow Fe_3O_4 \rightarrow FeO$).

In addition to the elements and compounds highlighted in the tables, there are other residuals in the burnt pellet, coming from the components of the mixture. Adding to these the loss on ignition, also residual, it was adopted the symbology *Σ Others (rp)*, to designate this complement of the final chemical analysis of the pellets. Normally, this complementary content is low and should be considered on the basis of historical data relating to the type of pellet under study. This is an input data, as highlighted in table 6.4.

Having made these considerations and considering the results calculated by the mass balance, table 6.3, it can be written for the composition of the burned pellets that:

$$Fe_2O_3 + FeO + SiO_2 + Al_2O_3 + CaO + MgO + QO + P_2O_5 + \Sigma \text{ Others} = 100\ \% \qquad \text{Eq. 6.51}$$

The following points must be considered for the unfolding of this equation:

- As reported, FeO (fe2+) and Σ Others (rp) are known, historical data.
- The total iron of the burnt pellet (FeT) equal to the sum of the iron contents contained in hematite, Fe_2O_3, and FeO: FeT = Fe (hematite) + Fe (FeO).
- Through stoichiometry, it is verified that Fe_2O_3 and FeO contain 70% and 78% of Fe, respectively.

- In the chemical analysis of the pellet, the content of the element phosphorus, P. However, in the mineralogical constitution of the pellet, it appears in the form of oxide, P_2O_5. Stoichiometric calculations show that P_2O_5 has 44% P.
- With the data in tables 6.1, 6.2 and 6.3, the contents of SiO_2 (sp), Al_2O_3 (ap), CaO (cp), MgO (gp), QO (qp) and P (pp) of the burned pellet can be calculated, calculations that will also prove the accuracy of the output data of the mass balance, as shown in table 6.4.
- These contents of the burned pellet can be calculated using the general equation Eq. 6.1, which, in this specific case, allows the calculation of the silica content (sp):
  - ✓ sp = (sm. M/100+sc. C/100+sg. G/100 +sq. Q/100+ sa. A/100 + cz.S.ss/10.000).100 /[P.(100+z)/100].
  - ✓ Thus, the other contents, ap, cp, gp, qp and pp can be calculated by analogy, replacing in this equation the silica contents of the various components of the mixture by the alumina contents, then by the CaO contents, and thus proceeding, successively, until the phosphorus contents.

Having then calculated all these data, the equation Eq. 6.51 could be written as:

Fe_2O_3 + (fe2+) + sp + ap + cp + gp + qp + pp/0.44 + rp = 100 Eq. 6.52

On the other hand, the total iron content of the pellet is given by:

FeT = (Fe_2O_3). 0.70 + (fe2+).0.78. From this, it results:

Fe_2O_3 = [FeT – (f2+).0.78]/0.70. Replacing itself in Eq. 6.52, we have:

FeT – (f2+).(0,78) + (fe2+).(0.70) = [100 - sp - ap - cp - gp - qp - pp.(2,27) – rp].(0,70)

Simplifying, it turns out:

FeT = (0.70). [100 - sp - ap - cp - gp - qp – (2.27).pp – rp] + (0.08). (Fe2+) Eq. 6.53

With this equation, the total iron content of the pellet is calculated, concluding the mass balance of pelletizing, as shown in the Excel spreadsheet of table 6.4.

i) *Considerations for the SOLVER tool and the Microsoft Excel spreadsheet*.

All the calculations and considerations discussed in this chapter 6 have been incorporated into the Excel spreadsheet in table 6.4. The *Solver* option of the spreadsheet was used to solve the system of linear equations, previously developed.

The spreadsheet is self-explanatory for those who have a minimal knowledge of Excel and who have become familiar with the general concepts explained in this chapter.

However, in order to make better use, some points should be highlighted:

- The mass balance can cater to any type of iron ore. In the case of magnetite, due to the mass gain in oxidation, the negative sign must be introduced in the loss on ignition value.

- With less precision, the model can be used for the Sintering of iron ore. The inaccuracy is due to the small reduction of the ore during the process (generation of FeO), with consequent weight loss.

- All *input data* is indicated on the spreadsheet, table 6.4, by *cells shaded in yellow*. Only these cells can be modified, in studies and simulations.

- The *7 results* of the system of linear equations are indicated in the *cells shaded in green*.

- The results of the *complementary calculations* to the mass balance are indicated in the *cells shaded in sand color*.

- The linear equations and imposed constraints are placed in a gray-shaded area of the worksheet. Under no circumstances may they be changed.

- As previously discussed, after analyzing and defining the balance sheet input data, one should go to the *Data Icon* of the spreadsheet, click on the *Solver* and *Run Options*. From there, it should be clicked *OK,* and all the results will be plotted in the corresponding cells of the spreadsheet (shaded in green and sand colors).

CHAPTER 7

CONSIDERATIONS ON THERMAL ENERGY CONSUMPTION IN PELLETIZING

The pelletizing of iron ore is an energy-intensive activity, whether electrical or thermal (heat). To obtain good quality pellets, green pellets need to be heated to average temperatures of the order of 1300 oC, so several physicochemical reactions occur, and the consolidation of the agglomerate is obtained. Thermal energy accounts for an appreciable portion of the transformation cost associated with pelletizing. In a production circuit that includes the grinding of iron ore, the cost of thermal energy can reach values of 30-40% of the cost of transformation ore-pellet, (hematite ore).

Hematite ores consume much more heat than magnetitic ores. As previously mentioned, magnetite ores are oxidized inside pelletizing furnaces through a highly exothermic reaction. The heat generated in this reaction, inside the pellets, accelerates the thermal processing and reduces the external heat input, (additional demands of fuel oil, natural gas, solid fuel, etc.). To minimize heat consumption in pelletizing of hematite ores, many companies add solid fuel to green pellets. The burning of this fuel inside the pellets directly provides part of the heat needed for the process, with a higher thermal efficiency.

The thermal balance of pelletizing furnaces has been extensively studied and is available in the technical literature. (8) When analyzing the subject, it is important to characterize the type of iron ore and the pelletizing technology, since these factors significantly affect the results. The thermal balance of pelletizing furnaces will not be the subject of this approach. It is thought that it is more interesting to characterize the *theoretical minimum heat consumption to produce 1 ton of pellets*, from *hematite* and *magnetite ores*. Taking the theoretical consumptions as a reference, it is possible to analyze, generically, the performance of a pelletizing furnace and the thermal efficiency of the pellet firing process.

For the formation of green pellets, in the discs or drums, the moisture content of the mixture of iron ore and additives must be adjusted to the *optimum point*. A value below this point leads to the formation of green pellets of inferior quality and the loss of productivity. On the other hand, an excess of moisture impairs the next step of burning the pellets, because this surplus has to be evaporated, causing increased consumption of thermal energy in the furnace. In fact, the optimal moisture content should be determined for each mixture composition to be pelletized. But there are still no standardized laboratory tests for this. However, there is an experimental methodology developed by Japanese Steel Mills, which is still under evaluation by pelletizing researchers. (1)

In the thermal processing of pellets inside pelletizing furnaces, the following physicochemical phenomena must be considered, among other possible ones:

- Heating and evaporation of moisture from green pellets;
- Dehydration of any compounds with water in the crystalline state, [limonite, Goethite, Ca(OH)2, Mg(OH)2, etc.];
- Calcination of carbonates, [CaCO3, MgCO3, Ca.Mg(CO3)2, etc.];
- Scoring reactions for ceramic consolidation, (CaO.SiO2, 2CaO.SiO2, CaO.Fe2O3, etc.);
- In the case of magnetite ore (Fe3O4), oxidation reaction to hematite (Fe2O3), with heat release;
- Heating of the pellets to 1300 oC;
- Cooling of the pellets from 1300 to 100 oC, and recovery of all this heat to the pellet burning system (heat released by the pellets in cooling from 1300 to 100 oC).

At the very least, these are the processes that must be considered in calculations of theoretical amount of heat for the pelletizing of iron ore.

The reactions of solid fuel (anthracite, coke breeze, etc.), which may be added to the green pellets, are not considered in the present calculation. This additive gives thermal energy to the pellets, acting as an external source of heat, similar to fuel oil, natural gas, pulverized coal, etc., which are burned in the burners of the pelletizing furnaces. Of course, when calculating the thermal balance of the pelletizing furnace, solid fuel would have to be considered.

For the development of the proposed calculation, it is necessary to define some assumptions and determine the composition of the mixture of constituents, for the production of 1 ton of burned pellets. This mixture can be determined by the mass balance, discussed in chapter 6, using the Excel Solver tool.

7.1 – *HEMATITE ORE: Calculation of the theoretical amount of heat for the production of 1 ton of pellets.* (1,8,9)

The following premises were considered for this case:

- Hematite ore, with 0.5% loss on ignition;
- The heat quantities were calculated in relation to the standard state: 25 oC or 298 °K and 1 atm;

- Pellets containing 1%, 3% and 5% SiO_2. These types of pellets are representative of the industrial reality, as they cover the fields of products for use in blast furnace and direct reduction. Once the calculations for these pellets have been developed, it is possible to perform similar calculations for any other type;
- For simplification, we considered the binary basicity $CaO/SiO_2 = 0.93$, equal to that of the compound $CaO.SiO_2$. Thus, all the silica reacts with CaO, stoichiometrically, with no leftovers of these compounds. This was the only slag reaction considered in the calculations. The other possible ones were snubbed;
- CaO doser: Limestone. To simplify the calculations of heat amounts, the limestone was considered pure, without contamination, and containing 100% $CaCO_3$. This hypothesis has no significant influence on the results;
- Binder: bentonite, in the amount of 0,5% in relation to the weight of the mixture to be pelletized;
- Moisture of green pellets: equal to 9%, regular value in industrial plants;
- The temperature of the green pellets was set at 35 oC, or 308 °K, a typical temperature of that observed in industrial pelletizing plants;
- It was established that the pellets should be heated until they reached the temperature of 1300 oC, or 1573 °K. In the Traveling Grate process, unlike the Grate-Kiln process, at the end of the firing zone, there is a significant difference between the temperatures of the top and bottom layers of the pellet bed. A good burning condition is one in which the average value of these temperatures is of the order of 1300 oC;
- To simplify the calculations related to the heating of the pellets to 1300 oC, it was established that they were composed of 100% Fe_2O_3. It was also considered that the predominant phase at 35 oC, at the beginning of the heating, was *the alpha Hematite* and that at 1300 oC it was the *gamma Hematite phase*;
- The processed pellets were cooled from 1300 to 100 oC (1573 to 373 °K) and, *hypothetically*, *all this heat* was recovered to the system and *fully incorporated into the pellets in the firing stage*. The final temperature of the pellets was considered equal to 100 oC, with predominance of the *Fe_2O_3 alpha phase*. Such a temperature can be reached in modern pelletizing furnaces.

Taking the values of these assumptions to the Excel program of the mass balance, the compositions of the green pellets resulted for the production of 1 ton of burnt pellets, with 1%, 3% and 5% of SiO_2, as shown in table 7.1.

From the data presented in table 7.1, it is verified that when the silica content of the pellet increased from 1% to 5%, the amount of limestone increased significantly.

Similarly, even with constant humidity at 9%, there was a progressive increase in water weight. As a result, the total weight of raw pellets to produce 1 t of burnt pellets jumped from 1,112.76 to 1,147.81 kg. Thus, when the silica content of the pellet is increased, keeping the binary basicity constant, there is a progressive loss of mass yield for the production of 1 t of pellet.

This, of course, leads to higher heat consumption for the calcination of the additional limestone and for the vaporization of the excess water weight. There is also a progressive reduction in the consumption of iron ore, given the increasing values of silica and the addition of limestone.

Table 7.1 – Composition of hematite green pellets for the production of 1 ton of burned pellets.

COMPONENT OF GREEN PELLET (Assumptions - mass balance)	1% SiO2		3% SiO2		5% SiO2	
	Weight (kg)	%	Weight (kg)	%	Weight (kg)	%
Hematite ore - Fe2O3	990,38	89,0	969,76	85,8	949,15	82,7
Limestone - CaCO3	17,17	1,5	53,65	4,7	90,14	7,9
Bentonite	5,06	0,5	5,14	0,5	5,22	0,5
Moisture - H2O	100,15	9,0	101,74	9,0	103,30	9,0
Total	1112,76	100,0	1.130,29	100,0	1.147,81	100,0

Table 7.2 shows the results regarding the theoretical amount of heat for the production of 1t of hematite pellets, within the established assumptions. The calculations were performed using a Microsoft Excel spreadsheet, where the model was built, based on the principles of thermodynamics, thermochemistry, and the premises previously established. The steps and equations for calculating the amounts of heat are broken down in this spreadsheet, which is part of this document, and is available in electronic media. Through this spreadsheet, it is possible to develop other studies and simulations, just by changing the cells shaded in yellow.

Table 7.2 – Calculation of the theoretical amount of heat for the production of 1 t of hematite pellets

Silica content of pellet - SiO2 %	1,00	3,00	5,00
Pellet basicity - CaO/SiO2	0,93	0,93	0,93
STAGES OF HEAT TREATMENT OF PELLETS			
1 - HEATING AND WATER VAPORIZATION OF GREEN PELLETS WITH 9% MOISTURE			
Weight of water in green pellet - from the mass balance (g)	100.150	101.740	103.310
Mol H2O (g)	18,00	18,00	18,00
# Moles H2O	5.563,89	5.652,22	5.739,44
Heating from 35 to 100 oC or 308 to 373 oK; Enthalpy of water = f (T), where T in oK:			
ΔH(T) = (HT - H298) = 18,04.T - 5379 cal/mol			
ΔH(308) = (18,04.308 - 5379).(no. moles)/1000 kcal	986,59	1.002,25	1.017,72
ΔH(373) = (18,04.373 - 5379).(no. moles)/1000 kcal	7.510,80	7.630,05	7.747,79
ΔH1 water heating from 35 to 100 oC = ΔH373 - ΔH308 Kcal	6.524	6.628	6.730
Evaporation			
ΔH (boiling) - kcal/mol	9,82	9,82	9,82
ΔH2 of water vaporization = ΔH (boiling) . (no. Moles) kcal	54.637	55.505	56.361
2 - LIMESTONE CALCINATION - Endothermic reaction			
Calcination reaction: CaCO3 = CaO + CO2			
Enthalpies of formation of these compounds - ΔHo(298) - kcal/mol			
ΔHCaCO3	-288,40	-288,40	-288,40
ΔHCaO	-151,60	-151,60	-151,60
ΔHCO2	-94,10	-94,10	-94,10
ΔH of limestone calcination reaction = (ΔHCaO + ΔHCO2) - ΔHCaCO3 - Kcal (endothermic)	42,70	42,70	42,70
Mol of CaCO3 (g)	100,00	100,00	100,00
CC - Heat consumption per gram of limestone = (ΔH da reação)/(mol do CaCO3) kcal/g	0,43	0,43	0,43
PC - Weight of limestone - from the mass balance (g)	17.170	53.650	90.140
ΔH3 of limestone calcination = CC.PC kcal	7.332	22.909	38.490
3 -SLAGGING REACTION - CaO.SiO2 - Exothermic reaction			
Slagging reaction: CaO + SiO2 = CaO.SiO2			
Enthalpies of formation of these compounds - ΔHo(298) - kcal/mol			
ΔHCaO	-151,60	-151,60	-151,60
ΔHSiO2	-217,00	-217,00	-217,00
ΔHCaO.SiO2	-390,00	-390,00	-390,00
ΔH of slagging reaction = ΔHCaO.SiO2 - (ΔHCaO + ΔHSiO2) - Kcal - Exothermic reaction	-21,40	-21,40	-21,40
Mol do SiO2 (g)	60,09	60,09	60,09
CL =Heat released by the slagging reaction per g de SiO2 = (ΔH reaction/mol of SiO2) - kcal/g	-0,36	-0,36	-0,36
% of SiO2 in the burned pellets - premises	1,00	3,00	5,00
PSiO2 = Weight of SiO2 in 1t of pellets (g)	10.000	30.000	50.000
ΔH4 of slagging = (CL.PSiO2) / heat released to the firing system in kcal	-3.561	-10.684	-17.807
4 - HEATING OF PELLETS (HEMATITE - Fe2O3) FROM 35 to 1300 oC (308 a 1573 oK)			
Weight of hematite - premise - 1 t of pellets (g)	1.000.000	1.000.000	1.000.000
Mol Fe2O3 (g)	160,00	160,00	160,00
# Moles Fe2O3	6.250,00	6.250,00	6.250,00
Heating from 308 to 1573 oK; Alpha phase --> Gamma phase - Enthalpy of Fe2O3 = f (T), where T in oK:			
ΔH Alpha hem (T) = (HT - H298) = 23,49.T + (9,30.T.T/1000) + (3,55.100000/T) -9021 cal/mol			
ΔH Gamma hem (T) = (HT - H298) = 31,71.T + (0,88.T.T/1000) - 8446 cal/mol			
ΔH Alpha hem (308) = {[(23,49.308) + (9,30.308.308/1000) + (3,55.100000/308) -9021]/1000}.(# Moles) kcal	1.554,70	1.554,70	1.554,70
ΔH Gamma hem (1573) = {[31,71.1573 + (0,88.1573.1573/1000) - 8446]/1000}.(# moles) kcal	272.570,25	272.570,25	272.570,25
ΔH5 of pellet heating = ΔH Gamma hem (1573) - ΔH Alpha hem (308) kcal	271.016	271.016	271.016
5 - HEAT RECOVERY OF PELLETS - COOLING FROM 1300 to 100 oC (1573 a 373 oK)			
Pellet cooling from 1573 to 373 oK; Gamma phase --> Alpha phase:			
ΔH Gamma hem (1573) = {[31,71.1573 + (0,88.1573.1573/1000) - 8446]/1000}.(# moles) kcal	272.570,25	272.570,25	272.570,25
ΔH Alpha hem (373) = {[(23,49.373) + (9,30.373.373/1000) + (3,55.100000/373) -9021]/1000}.(# Moles) kcal	12.415,08	12.415,08	12.415,08
ΔH6 of pellet cooling /recovered heat = ΔH Alpha hem (373) - ΔH Gamma hem (1573) kcal	-260.155	-260.155	-260.155
6 - THEORETICAL AMOUNT OF HEAT FOR BURNING 1 t OF PELLETS			
6.1 - Without heat recovery from burned pellets (kcal/t): ΔH = ΔH1 + ΔH2 +ΔH3 +ΔH4 +ΔH5	**335.947**	**345.373**	**354.790**
6.2 - WITH HEAT RECOVERY OF THE HOT PELLETS AT THE END OF BURNING STAGE (kcal/t): ΔH' = ΔH1 + ΔH2 + ΔH3 + ΔH4 + ΔH5 + ΔH6	**75.792** (Referência)	**85.218** (+ 12 %)	**94.635** (+ 25 %)

Considering the *established assumptions* and analyzing the data in table 7.2, the following comments could be made:

- The <u>theoretical</u> consumption of heat from external sources for the production of 1 ton of pellets, considering the heat recovery, grows progressively with the increase of the silica content. For 1% SiO2, the consumption is close to 76,000 kcal/t and for 5% it is around 95,000 kcal/t (+25%), as shown in Figure 7.1. Within the premise of constant basicity, established for the three types of pellets, (CaO/SiO2=0.93), the growth of the silica content implies a progressive increase in the amount of CaO added to the pellets. In other words, in the need for greater heat input for the decomposition of limestone equivalent to increased basicity.

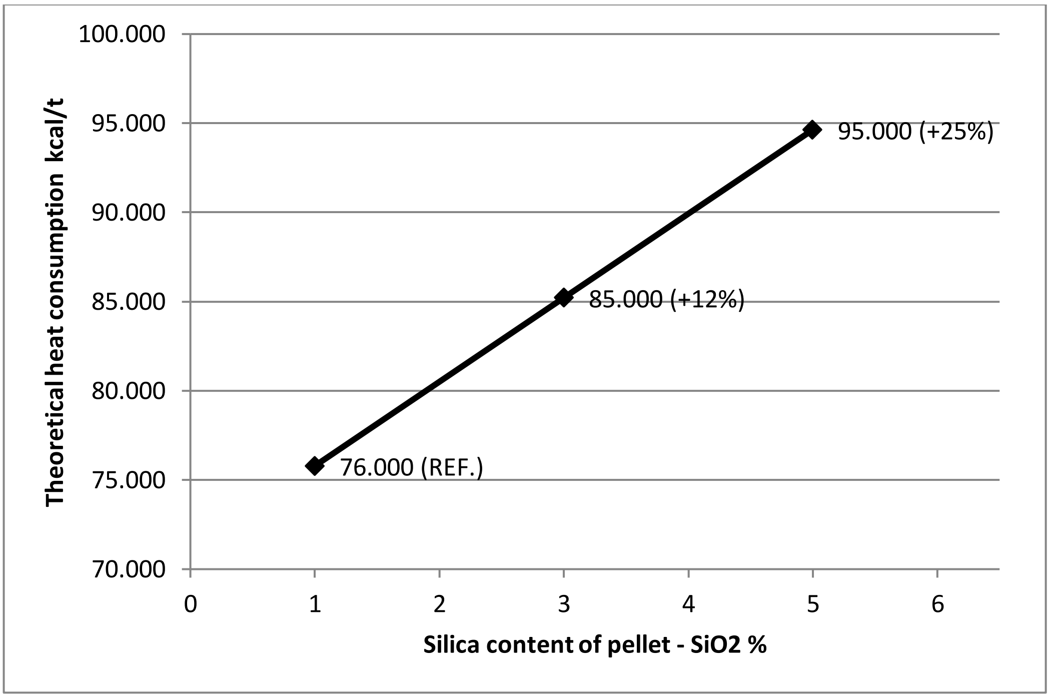


Figure 7.1 – Theoretical heat consumption for the production of 1 t of hematite pellet.

- If there was no heat recovery in the cooling of the pellets, the theoretical heat consumption would rise considerably, from 76,000 kcal/t to 336,000 kcal/t for the pellets with 1% SiO2, and from 95,000 kcal/t to 355,000 kcal/t in the case of pellets with 5% SiO2. Figure 7.2 shows how the heat recovery in the cooling of the pellets is fundamental, so that there is less heat demand from external sources (solid fuel, fuel oil, natural gas, etc.). This shows the importance of heat recovery in the industrial process. It should be noted that in conventional *sintering* of iron ore fines there is no heat recovery, and, for this reason, thermal energy consumption is very high, compared to pelletizing.

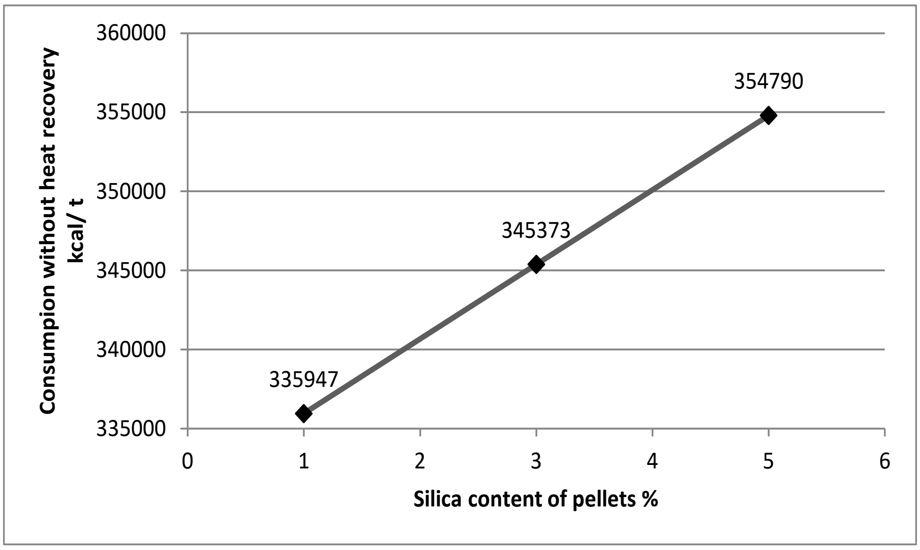


Figure 7.2 – Theoretical heat consumption without recovery in the cooling stage of the pellets.

- To vaporize the moisture from the green pellets, a large amount of heat is required. This portion represents more than 60% of the theoretical heat required for the production of pellets, considering the *integral recovery of heat in cooling and its total incorporation into the pellets*. The data in Table 7.2 and Figure 7.3 show this fact. For this reason, in industrial plants, a strict control of the moisture content of raw pellets should be established, seeking to work at the optimum point. As iron ores are different and there are several types of pellets, this optimal moisture content must be established for each mixture to be pelletized. It is worth mentioning that for every additional 1% of green pellet moisture, there is an increase in heat consumption of about 6,000 kcal.

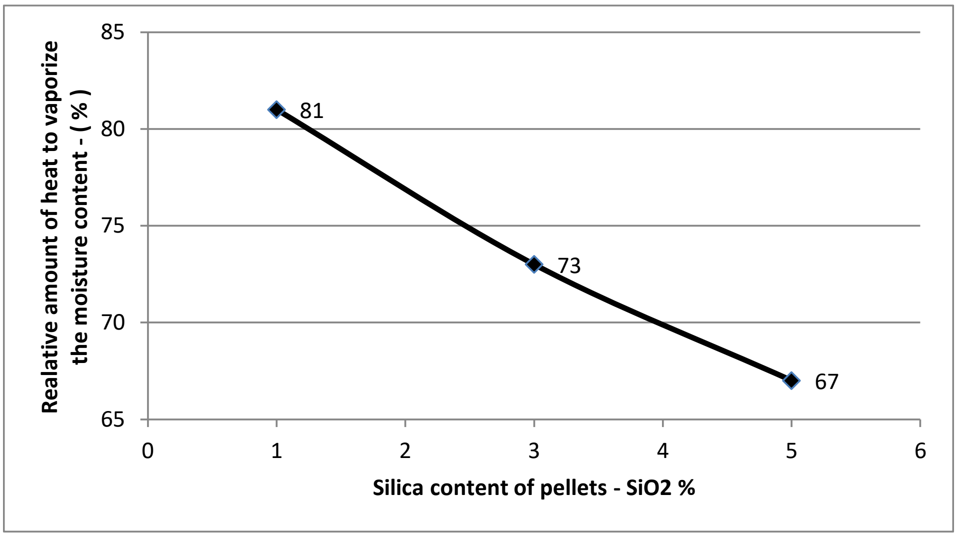


Figure 7.3 - % of the theoretical heat required to vaporize moisture from green pellets.

- The industrial furnaces of pelletizing of hematite ores present very high heat consumption, in relation to the minimum theoretical, with heat recovery. Depending on the efficiency of the furnaces, the consumptions can vary in the range of 180,000 to 350,000 kcal/t of pellets. The *efficiency in the use of thermal energy* (μ) **could be evaluated, indirectly, by the percentage relationship between the theoretical consumption of heat (**with recovery) and the total consumption, verified in an industrial plant. Within this concept, efficiency curves, as a function of the silica content of the burned pellets, could be constructed, as shown in Figure 7.4. It is important to emphasize that *the Thermal Efficiency*, so designated here, can be understood as *an indication* of the use of heat in pelletizing furnaces.

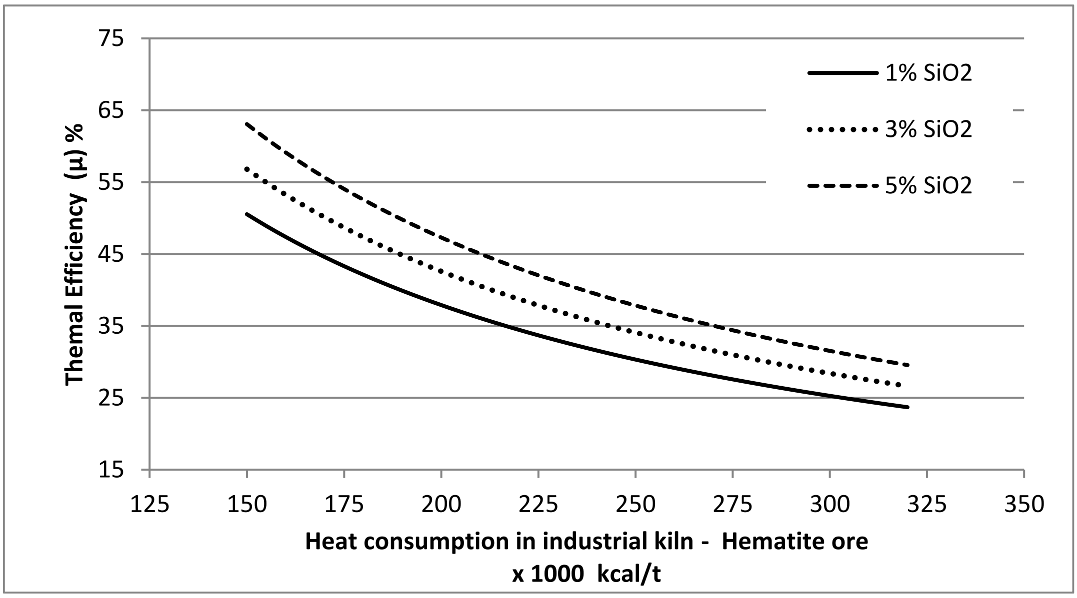


Figure 7.4 – Efficiency in the use of thermal energy for hematite ore.

For the determination of the thermal yield of industrial furnaces (8) it is necessary to calculate the thermal balance, considering all heat flows, materials, thermal losses, etc. Known industry data (1) have shown that the thermal yield is relatively low, being in the range of 25 to 40 % for hematite ores, depending on the operating conditions. There are reports of world-class furnaces that produce hematite pellets, direct reduction type (1 % SiO2), with heat consumption in the order of 170,000 kcal/t. Plotting this value in the graph of Figure 7.4, it is verified that the *thermal efficiency* of these furnaces is in the order of 45% (benchmark). Extrapolating this same efficiency of 45% for the pellets with 3 and 5% SiO2, it is observed that the corresponding heat consumption would have to be equal to 190,000 kcal/t and 210,000 kcal/t, respectively (values not yet reached in industrial operations with hematite).

Thus, there is a lot of opportunity for process improvement in the operation of pelletizing furnaces. From the general analysis of the topic of thermal energy, the question remains:

Would it be possible to achieve *__thermal efficiency__* *of 50% in industrial furnaces, operating with hematite?*

μ % = (theoretical heat with recovery / heat consumed in the furnace).100
Eq. 7.1

- An important point to be observed, and which has a great impact on thermal efficiency, is the *__final temperature of the pellets__*, after their cooling in the pelletizing furnaces.

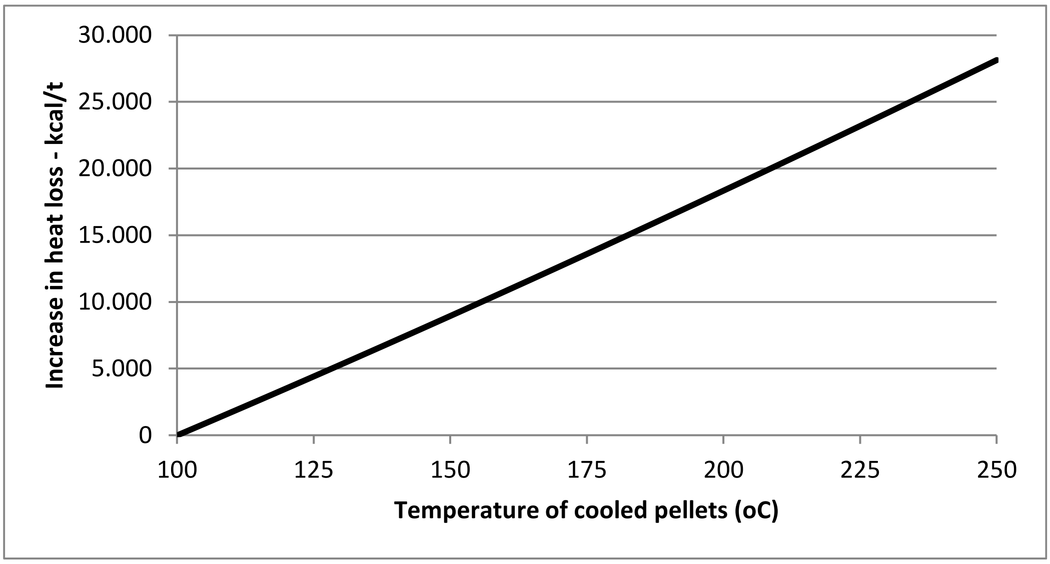


Figure 7.5 – Increase in heat loss as a function of the final temperature of the pellets.

Even after cooling, the pellets carry a large amount of heat. The higher the temperature, the lower the cooling efficiency in the furnace (heat recovery) and the greater the loss of thermal energy to the environment since the pellets are stored in open yards. This parameter, the final temperature of the pellets, must be continuously monitored and minimized in the operation of industrial furnaces, to reduce heat losses and improve thermal efficiency. According to the premises adopted in this development, the final reference temperature of the pellets, after cooling, was __100 oC__. In view of this condition, calculations of *__additional heat losses__* were developed, corresponding to cases in which the pellets were cooled to temperatures above 100 oC

Figure 7.5 shows how the minimization of the final temperature of the pellets is fundamental for the control of thermal energy losses in pelletizing furnaces.

- When analyzing the industrial pelletizing furnaces of *hematite ore*, it is verified that the thermal efficiency is relatively low. Thus, a process control must be established, so that heat losses are minimized and heat recovery in the cooling of the pellets is maximized.

The dosage of solid fuel to raw pellets improves thermal efficiency since the heat of combustion is generated inside the pellets. Thus, one should try to work with the maximum dosage of this solid fuel. There are industrial pelletizing plants (1) that operate with the addition of 1.25% of fixed carbon, being a reference in this area (benchmark).

The temperature of the green pellets should also be considered in these studies to improve the thermal efficiency. Higher temperatures speed up the drying steps in the kiln.

In the Traveling Grate processes, part of the pellet production is recycled for the formation of the bottom and side layers of the grate. The height and width of these layers should be reduced as much as possible, as these pellets lose some heat to the environment on the way back to the furnace (recirculation). Some industrial plants have already eliminated the side layer and work with a bed layer of only 5 cm.

The zones of the pelletizing furnaces work with slightly negative relative pressure (vacuum), so that there is continuously a flow of cold air from the environment to the interior of the furnace. That is, also a factor of loss of thermal efficiency. Thus, these relative pressures should be monitored and maintained at values as close to zero as possible.

Grate-Kiln (GK) pelletizing furnaces have 3 reactors, rather than a single one, as in the Traveling Grate (TG) process. The total residence time in the GK is in the order of 70 min, more than double that of modern TG kilns. These two factors contribute to the fact that the thermal losses and heat consumption in the GK process are higher than the corresponding values verified in the TG process.

In studies to optimize the thermal efficiency of furnaces, it is necessary to consider the size of the pellets and the porosity of the bed as critical factors. The smaller pellets have a larger specific surface (m2/t), favoring contact with hot gases in the heating and firing phases, and with cold air in the cooling stage. Thus, they accelerate heat exchanges by convection hot gas/pellet and hot pellet/cold air, increasing the yields in heating (heat absorption) and cooling of pellets (heat recovery).

Gas flows through the pellets depend on the porosity of the bed. To improve this porosity, the size range of the pellets should be narrowed, decreasing the amplitude of the sizes. Since convection is the main form of heat transmission between the pellets and the gases, there is a lot of inefficiency in this process, which implies considerable heat losses, either through the burned pellets that leave the furnace, or through the hot exhaust gases in the chimneys (sensible heat).

Thus, these and other heat losses should be studied in more detail:

- Losses in hot exhaust gases in chimneys;
- Losses by heat transmission through the refractory walls and ducts of the furnace;
- Leakage of hot gases into the environment;
- Cold air intake due to problems in seals and negative pressure in the furnace zones (vacuum);
- Possible water entrance;
- Heat loss in the recirculation of the hearth and side layers (Traveling Grate process);
- Heat losses through the grate cars (output/re-entry temperatures in the kiln);
- Cold air intake for atomization of burners;
- And so on.

In addition to the technical-economic aspects of these thermal losses, it is worth mentioning that all the heat lost in the pelletizing furnaces goes to the environment and that greater losses mean higher emissions of CO2, SOx, NOx, etc.

To summarize what was discussed about thermal energy in *pelletizing furnaces of hematite ores* and about heat losses to the environment, the diagram shown in Figure 7.6 was drawn.

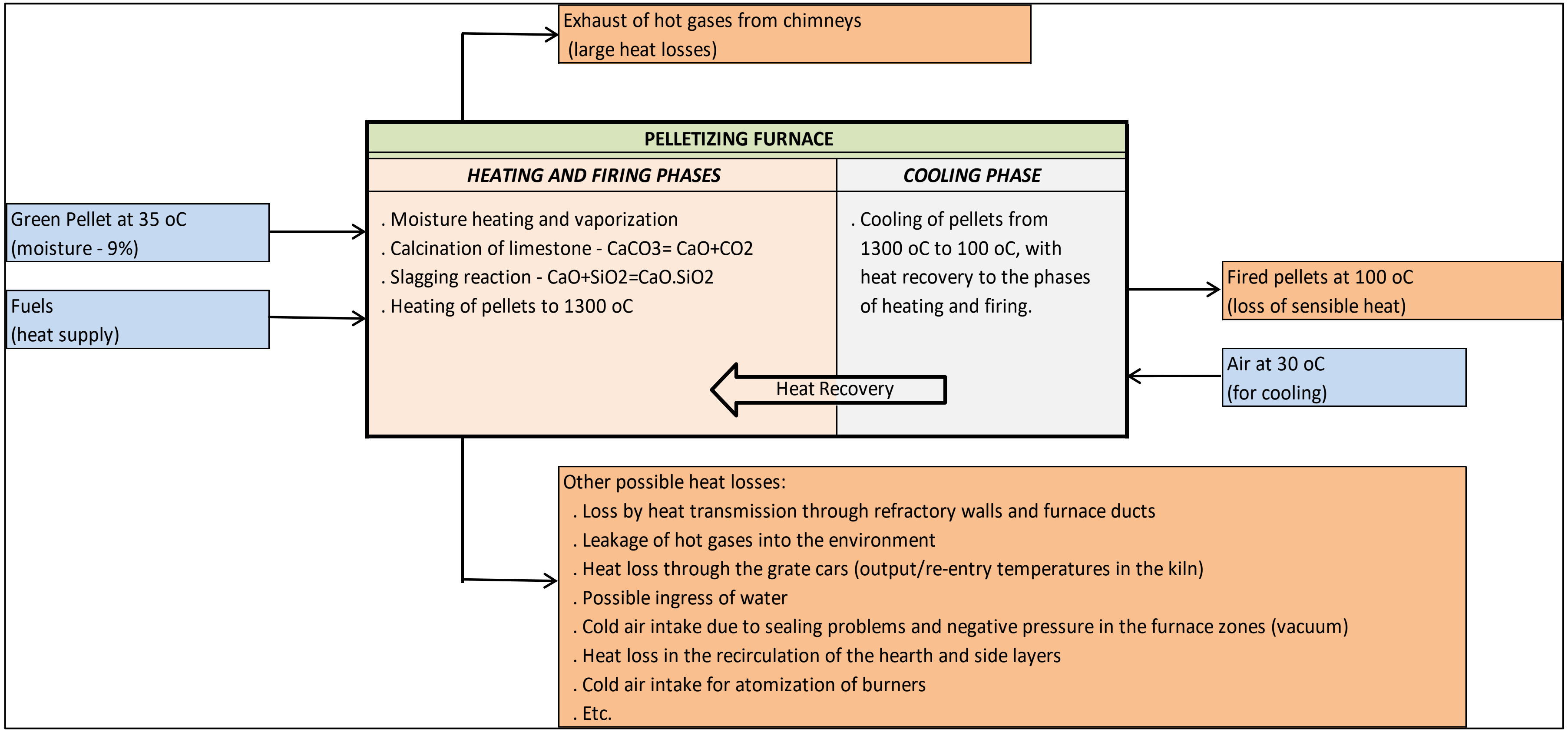


Figure 7.6 - Generic diagram of a pelletizing furnace, showing the heat flows and heat loss factors.

7.2 – *MAGNETITE ORE*: *Calculation of the theoretical amount of heat for production 1 ton of pellets*. (1,8,9)

Similarly, to the case of hematite ore, some premises were adopted for the development of the proposed calculation, using magnetite ore. As already reported, the biggest difference that occurs with magnetite is its oxidation inside the pelletizing furnace, with release of large amounts of heat, which drastically reduces the heat demand from external sources. In this oxidation, the magnetite, Fe3O4, is transformed into hematite, generating a product in which the predominant phase is composed of Fe2O3, according to the following *exothermic chemical reaction*:

$$2Fe3O4 + 1/2\ O2 \rightarrow 3Fe2O3 \qquad \Delta H = -\ 54\ Kcal$$

Iron ores designated as *magnetite* are usually composed of mixtures of ferrous mineral phases, with a predominance of magnetite (Fe3O4). Purest magnetitic ores, with Fe3O4 > 90%, are found in some countries, especially Sweden.

For the development of the proposed calculation, the following assumptions were adopted, like to what was done for the hematite ore and introducing some characteristic parameters of magnetite ore:

- Magnetite ore, with 100% *Fe3O4* and -3.0% of loss on ignition (negative);
- The heat quantities were calculated in relation to the standard state: 25oC or 298 oK and 1 atm;
- Pellets with three levels of SiO2: 1%, 3% and 5%;
- Binary basicity CaO/SiO2 = 0.93, equal to that of the compound CaO.SiO2. Thus, all the silica reacts with CaO, stoichiometrically, with no leftovers of these compounds. This was the only slag reaction considered in the calculations. The other possible ones were snubbed;
- CaO doser: limestone. To simplify the calculations of heat amounts, the limestone was considered pure, without contamination, and containing 100% CaCO3;
- Binder: bentonite, in the amount of 0,5% in relation to the weight of the mixture to be balled;
- Moisture of green pellets: equal to 9%, regular value in industrial plants;
- The temperature of the green pellets was set at 35 oC, or 308 oK, temperature;
- It was established that the pellets should be heated until they reached the temperature of 1300 oC, or the same as 1573 oK;
- The magnetite, Fe3O4, was fully oxidized to Fe2O3. In this process, to simplify the calculations, the ore was considered to be without impurities;
- In the heating of the pellets to 1300 oC, it was considered that the pellets were composed *of Fe3O4 alpha phase, at the initial temperature of 35 oC, and 100%*

of Fe2O3 gamma phase (hematite resulting from the oxidation of magnetite), at the temperature of 1300 oC;

- The processed pellets were cooled from 1300 to 100 oC (1573 to 373 oK) and, *hypothetically*, *all this heat* was recovered to the system, and *fully incorporated into the pellets in the firing stage.* The final temperature of the pellets was considered equal to 100 oC, with predominance of *the Hematite alpha phase*.

Taking the values of these assumptions to the Excel program of the mass balance, the compositions of the green pellets resulted for the production of 1 ton of burnt pellets, with 1%, 3% and 5% of SiO2, as shown in table 7.3.

From the data presented in table 7.3 and compared to the hematite pellets shown in table 7.1, it is verified that the amounts of magnetite ore to produce 1 t of pellets are smaller, as well as the amounts of water of the pelleting mixtures and the total weights of the related raw pellets.

This is because, during burning, there is incorporation of mass (oxygen) to the pellets, due to the oxidation of magnetite. When increasing the silica content of the pellet from 1% to 5%, the trends were the same as those observed for the corresponding hematite pellets, and no other comments were appropriate.

Table 7.3 – Composition of magnetite green pellets for the production of 1 ton of burned pellets.

COMPONENT OF GREEN PELLETS (Assumptions - mass balance)	1% SiO2		3% SiO2		5% SiO2	
	Weight (kg)	%	Weight (kg)	%	Weight (kg)	%
Magnetite ore - Fe3O4	956,86	88,9	936,95	85,6	917,03	82,4
Limestone - CaCO3	17,20	1,6	53,69	4,9	90,18	8,1
Bentonite	4,89	0,5	4,98	0,5	5,06	0,5
Moisture - H2O	96,82	9,0	98,47	9,0	100,11	9,0
Total	1075,77	100,0	1.094,09	100,0	1.112,38	100,0

Table 7.4 shows the results regarding the theoretical amount of heat for the production of 1 t of pellets, from magnetitic ore with 100% Fe3O4, within the established premises. The calculations were performed using a Microsoft Excel spreadsheet, where the model was built.

The model was based on the physical, chemical, thermodynamic, and thermochemical principles that govern the subject, as well as on the set of established premises.

This spreadsheet is part of this document and available electronically. Through this spreadsheet, it is possible to develop other studies and simulations, just by changing the cells shaded in yellow.

Analyzing the data in table 7.4 and in summarized table 7.5, compared to those in table 7.2, constructed for the case of hematite pellets, it is important to highlight the following points:

- The theoretical heat consumption for the magnetite pellets, *without heat recovery*, were significantly lower than those verified for the corresponding hematite pellets of the same silica content. The differences were of the order of 110,000 kcal/t, the amount of heat associated with the oxidation of the magnetite present in the iron ore, figure 7.7.

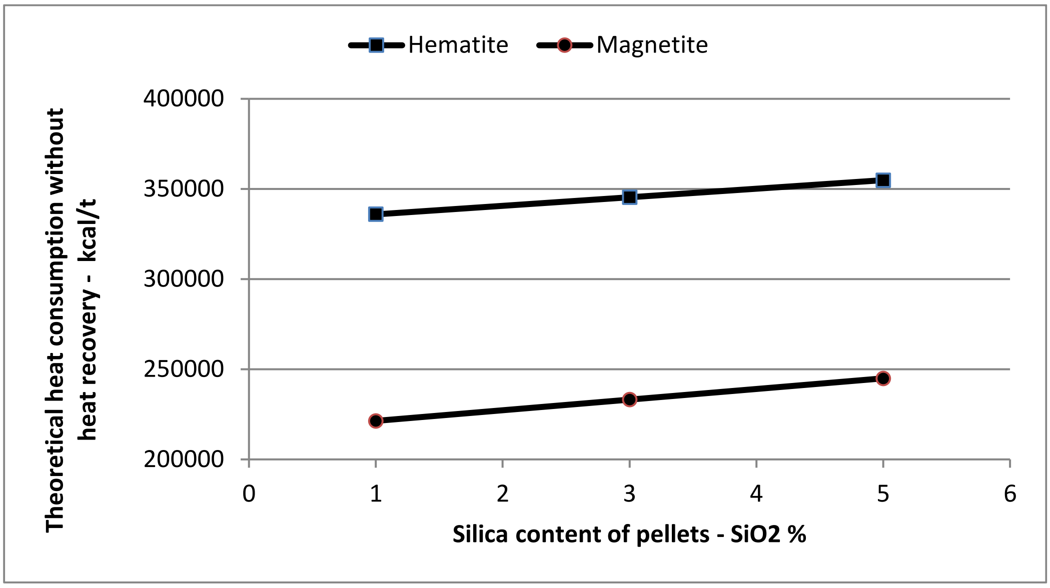


Figure 7.7 - Theoretical heat consumption for hematite and magnetite pellets (without heat recovery in cooling).

- Considering the recovery of heat in cooling according to the established premise, it is verified that the theoretical heat consumption, for pellets with 100% of magnetite ore, are negative, as shown in tables 7.4 and 7.5. This means that if all the premises could be met in practice, and the activation energy was inserted into the pelletizing system, only the heat resulting from the oxidation of magnetite would be sufficient to cover all the demands and there would still be a good portion of heat leftover.

Table 7.4 – Calculation of the theoretical amount of heat for the production of 1 t of magnetite pellets

Silica content of pellet - SiO2 %	1,00	3,00	5,00
Basicity of pellets - CaO/SiO2	0,93	0,93	0,93
STAGES OF HEAT TREATMENT OF PELLETS			
1 - HEATING AND WATER VAPORIZATION OF GREEN PELLETS WITH 9% MOISTURE			
Weight of water in green pellet - from the mass balance (g)	96.820	98.470	100.110
Mol H2O (g)	18,00	18,00	18,00
# Moles H2O	5.378,89	5.470,56	5.561,67
Heating from 35 to 100 oC or 308 to 373 oK; Enthalpy of water = f (T), where T in oK:			
ΔH(T) = (HT - H298) = 18,04.T - 5379 cal/mol			
ΔH(308) = (18,04.308 - 5379).(# moles)/1000 kcal	953,78	970,04	986,19
ΔH(373) = (18,04.373 - 5379).(# moles)/1000 kcal	7.261,07	7.384,81	7.507,81
ΔH1 water heating from 35 to 100 oC = ΔH373 - ΔH308 Kcal	6.307	6.415	6.522
Evaporation			
ΔH (boiling) - kcal/mol	9,82	9,82	9,82
ΔH2 of water vaporization = ΔH (boiling) . (no. Moles) kcal	52.821	53.721	54.616
2 - LIMESTONE CALCINATION - Endothermic reaction			
Calcination reaction: CaCO3 = CaO + CO2			
Enthalpies of formation of these compounds - ΔHo(298) - kcal/mol			
ΔHCaCO3	-288,40	-288,40	-288,40
ΔHCaO	-151,60	-151,60	-151,60
ΔHCO2	-94,10	-94,10	-94,10
ΔH of limestone calcination reaction = (ΔHCaO + ΔHCO2) - ΔHCaCO3 - Kcal (endothermic)	42,70	42,70	42,70
Mol of CaCO3 (g)	100,00	100,00	100,00
CC - Heat consumption per gram of limestone = (ΔH da reação)/(mol do CaCO3) kcal/g	0,43	0,43	0,43
PC - Weight of limestone - from the mass balance (g)	17.200	53.690	90.180
ΔH3 of limestone calcination = CC.PC kcal	7.344	22.926	38.507
3 - SLAGGING REACTION - CaO.SiO2 - Exothermic reaction			
Slagging reaction: CaO + SiO2 = CaO.SiO2			
Enthalpies of formation of these compounds - ΔHo(298) - kcal/mol			
ΔHCaO	-151,60	-151,60	-151,60
ΔHSiO2	-217,00	-217,00	-217,00
ΔHCaO.SiO2	-390,00	-390,00	-390,00
ΔH of slagging reaction = ΔHCaO.SiO2 - (ΔHCaO + ΔHSiO2) - Kcal - Exothermic reaction	-21,40	-21,40	-21,40
Mol do SiO2 (g)	60,09	60,09	60,09
CL =Heat released by the slagging reaction per g de SiO2 = (ΔH reaction/mol of SiO2) - kcal/g	-0,36	-0,36	-0,36
% of SiO2 in the burned pellets - premises	1,00	3,00	5,00
PSiO2 = Weight of SiO2 in 1t of pellets (g)	10.000,00	30.000,00	50.000,00
ΔH4 of slagging = (CL.PSiO2) / heat released to the firing system in kcal	-3.561	-10.684	-17.807
4 - HEATING OF PELLETS (Magnetite - alpha)) FROM 35 to 1300 oC - 308 to 1573 oK (Hematite - gamma)			
Weight of hematite - premise - 1 t of pellets (g)	1.000.000	1.000.000	1.000.000
Mol Fe2O3 (g)	160,00	160,00	160,00
# Moles Fe2O3	6.250,00	6.250,00	6.250,00
Heating from 308 a 1573 oK; Mag. alpha --> Hem. gamma - Enthalpy = f (T), being T in oK:			
ΔH Mag alpha (T) = (HT - H298) = 21,88.T + (24,10.T.T/1000) - 8666 cal/mol			
ΔH Hem gamma (T) = (HT - H298) = 31,71.T + (0,88.T.T/1000) - 8446 cal/mol			
ΔH Mag alpha (308) = {[21,88.308 + (24,10.308.308/1000) - 8666]/1000} .(# moles) kcal	2.245,39	2.245,39	2.245,39
ΔH Hem gamma (1573) = {[31,71.1573 + (0,88.1573.1573/1000) - 8446]/1000}.(# moles) kcal	272.570,25	272.570,25	272.570,25
ΔH5 - Heating of pellets = ΔH Hem gamma (1573) - ΔH Mag alpha (308) kcal	270.325	270.325	270.325
5 - OXIDATION OF MAGNEITE - Exothermic Reaction - Ore with 100% Fe3O4			
PM = 100% of the weight of magnetite ore from the mass balance (g)	956.860	936.950	917.030
ΔHo(298): Fe3O4 = -267 kcal/mol; O2 = 0 kcal/mol; Fe2O3 = -196 kcal/mol			
Reaction of oxidation: 2Fe3O4 + 1/2 O2 = 3Fe2O3; ΔH of reaction = 3.(-196) - 2.(-267) = -54 kcal			
Heat released per gram of magnetite = ΔH Mag = -54/463,1 = -0,117 Kcal/g of magnetite			
ΔH6 - Magnetite oxidation / Heat for the system = PM.(-0,117) Kcal	-111.953	-109.623	-107.293
6 - HEAT RECOVERY FROM PELLETS - COOLING from 1300 to 100 oC (1573 to 373 oK)			
Cooling of pellets from 1573 to 373 oK; Hem gamma --> Hem alpha:			
ΔH Hem gamma (1573) = {[31,71.1573 + (0,88.1573.1573/1000) - 8446]/1000}.(# moles) kcal	272.570,25	272.570,25	272.570,25
ΔH Hem alpha (373) = {[(23,49.373) + (9,30.373.373/1000) + (3,55.100000/373) -9021]/1000}.(# Moles) kcal	12.415,08	12.415,08	12.415,08
ΔH7 - Cooling of pellets / Heat recovery = ΔH Hem alpha (373) - ΔH Hem gamma (1573) kcal	-260.155	-260.155	-260.155
7 - THEORETICAL AMOUNT OF HEAT FOR BURNING 1 t OF PELLETS			
7.1 - Without heat recovery from burned pellets (kcal/t): ΔH = ΔH1 + ΔH2 + ΔH3 + ΔH4 + ΔH5 + ΔH6	**221.283**	**233.079**	**244.870**
7.2 - WITH HEAT RECOVERY OF THE HOT PELLETS AT THE END OF BURNING STAGE (kcal/t): ΔH' = ΔH1 + ΔH2 + ΔH3 + ΔH4 + ΔH5 + ΔH6 + ΔH7	**-38.872**	**-27.076**	**-15.285**

Table 7.5 – Summary of the theoretical amount of heat for burning 1 t of pellets (hematite and magnetite ores).

TYPE OF IRON ORE	*HEMATITE*			*MAGNETITE*		
SILICA CONTENT %	*1,00*	*3,00*	*5,00*	*1,00*	*3,00*	*5,00*
BASICITY - CaO/SiO2	*0,93*	*0,93*	*0,93*	*0,93*	*0,93*	*0,93*
PHASES OF FIRING PROCESS	*Heat Amount in kcal / t pellet*					
1 - HEATING AND VAPORIZATION OF WATER	61.162	62.133	63.091	59.128	60.136	61.137
2 - CALCINATION OF LIMESTONE	7.332	22.909	38.490	7.344	22.926	38.507
3 - SLAGGING REACTION - CaO.SiO2	-3.561	-10.684	-17.807	-3.561	-10.684	-17.807
4 - HEATING OF PELLETS TO 1,300 oC	271.016	271.016	271.016	270.325	270.325	270.325
5 - MAGNETITE OXIDATION- Exothermic	-	-	-	-111.953	-109.623	-107.293
6 - HEAT RECOVERY FROM PELLETS	-260.155	-260.155	-260.155	-260.155	-260.155	-260.155
7 - THEORETICAL AMOUNT OF HEAT						
7.1 - WITHOUT HEAT RECOVERY (kcal/t):	**335.947**	**345.373**	**354.790**	**221.283**	**233.079**	**244.870**
7.2 - WITH HEAT RECOVERY (kcal/t):	**75.792** (Reference)	**85.218** (+ 12 %)	**94.635** (+ 25 %)	**-38.872**	**-27.076**	**-15.285**

- The amounts of heat leftovers for the pellets with 1, 3 and 5% of SiO2 would be equal to the absolute values of the negative results found. In industrial furnaces, due to inefficiency factors and also the fact that magnetite ores have variable amounts of magnetite, Fe3O4, heat consumption from external sources varies in the range of 50,000 to 150,000 kcal/t of pellets. Known industry data (1) have shown that the thermal efficiency is relatively lower for furnaces operating with magnetite ores, compared to those with hematite ones. As magnetite furnaces require very little external thermal energy and industrial furnaces are of large size, the possibility of thermal losses, in relative terms, becomes greater, leading to lower efficiencies.

7.3 – *The consumption of thermal energy in pelletizing circuit with Dry Grinding*.

When the flowcharts of the various pelletizing plants are analyzed, it is verified that there are a number of units with *dry grinding*, in the ore preparation phase. Such circuits differ greatly from those in which the grinding is conducted in wet, where there is a need to perform many other unit operations. Among these processing operations, the following are common: thickening, homogenizing, filtering, etc., as shown earlier in Chapter 2, Figure 2.2. It turns out that, in operational terms, the pelletizing circuit with dry grinding is much simpler than the one with wet grinding. The definition of the type of grinding depends on several factors. Certainly, the characteristics of the ore to be pelletized have a determining importance in this choice. Normally, iron ores originated from geological processes by supergenic enrichment, with high levels of goethite and/or limonite, micro or cryptocrystalline and earthy aspect, present a fragmentation in the grinding with generation of appreciable amount of micro fines. Depending on the amount of these micro fines generated, the filtration of the pulp from the wet grinding circuit becomes very difficult and sometimes very costly, so dry grinding is indicated.

Although the dry grinding circuit is simpler, with less capital investment, in operation of the plant, there is a need *to dry the pellet feed,* which demands an appreciable consumption of heat, or even thermal energy, with a permanent impact on the operating cost. Depending on the origin, the moisture of the concentrated pellet feed can vary in the range of 6 to 9%. It is important to highlight that in operation of pelletizing plants with dry grinding, the ore is dried before the mills entrance and, subsequently, there is the need to again moist the ore at a level of 9 to 10%, for the balling stage. Although it seems counterintuitive to dry the ore and then, soon after, add the necessary moisture content to the balling, it is necessary to consider the technical-economic feasibility studies and the recommendations for the adoption of dry grinding.

In any case, even if in the drying operation some sensible heat from exhaust gases from chimneys of the pelletizing plant is used, the heat demand can be quite significant.

Taking the equations shown in tables 7.2 and 7.5, it is verified that this amount of heat could reach up to 60,000 kcal/t of pellets, in theoretical terms, without considering the inefficiencies of the pellet feed drying equipment. If a high efficiency of 50% were obtained, the heat consumption in grinding would be equivalent to 120,000 kcal/t of pellets. This is a fairly significant amount of energy and deserves special evaluation in the feasibility studies.

CHAPTER 8

ASPECTS RELATED TO THE CONSUMPTION OF ELECTRIC ENERGY IN PELLETIZING (1)

The pelletizing of iron ores is an energy-intensive activity, whether thermal or electrical. Electric energy accounts for a large portion of the operating cost of a pelletizing plant, sometimes accounting for 25 to 30% of this cost. In a plant that involves the three stages of the pelletizing process, *preparation-balling-burning*, the grinding and the furnace fans account for about 70% of the total consumption of electrical energy, deserving special attention from process control. This total consumption of electricity depends on several factors, among which could be cited:

- ✓ Characteristics of iron ore;
- ✓ Flowchart of the pelletizing plant;
- ✓ Characteristics and types of equipment of unit operations;
- ✓ Dosage of solid fuel to green pellets;
- ✓ Type of pellet;
- ✓ Type of pellet firing furnace;
- ✓ And so on.

- As for the *characteristics of iron ore,* it should be highlighted as fundamental: *the type of ore, whether magnetite or hematite, the content of hydrated ores, the grindability index and the degree of filterability*. The magnetite ores, due to the oxidation of magnetite inside the furnaces, favor the burning of the pellets, with consequent acceleration of the firing process and increased productivity of the furnace. Due to this, the specific electricity consumption of the furnace fans (kwh/t) is lower than that for the hematite pellets. The productivity, reference standard (benchmark) (1) traveling grate plants type, operating with 100% magnetite ore is of the order of 40 t/m2.24h and with hematite ore, with the addition of solid fuel, is around 30 t/m2.24h. These data show the difference in yields for the two types of iron ore, which justifies the lower energy consumption of the furnace fans when treating the magnetite ones. The proportion of hydrated ores, such as goethite and limonite (high loss on ignition), greatly influences the consumption of electrical energy of the furnace, since they interfere negatively in productivity. These ores have to be dehydrated during heat treatment, slowing down pelletizing reactions. On the other hand, ores have very variable grindability, depending on their origin. This has a major impact on the electrical energy consumption of the mill during the preparation for balling. Hematite ores of high grindability have smaller crystals and usually come from geological processes with supergenic enrichment. For plants that use the wet circuit, the filterability of the ore is impactful, since ores that are difficult to filter cause low productivity of the filters and impose the use of more robust equipment that sometimes operates at higher pressures, requiring greater amounts of electrical energy (use of press filters, for example).

- With regard to the *flowchart of the pelletizing plant*, the most important thing is to know whether or not the ore preparation step is included in the circuit. Also, whether the operations are dry or wet. The grinding is part of the preparation of the ore for balling, being a unit operation of high consumption of electricity. Dry grinding consumes more electrical energy than wet grinding. On the other hand, dry circuits eliminate some unit operations, such as: thickening, homogenization, and filtration, which have an influence on the consumption of electricity. If the ore was received at the pelletizing plant already ground and ready for use, the circuit would become much simpler and with less demand for energy, since it would involve only the other two stages of the pelletizing process, balling and burning.

- As for *the characteristics and types of equipment of unit operations*, it is worth mentioning that, in recent years, there has been a great evolution in the area of iron ore grinding, with the development of tower mills and roller presses, whose proper application in pelletizing circuits can result in significant gains in electricity and productivity. In the area of filtration, ceramic filters have emerged with significant gains in electrical energy. Regarding the furnace fans, the biggest event was the introduction of variable speed drives for the control of gas flow, eliminating the archaic flow control systems through dampers. With this, the pressure loss in the control of the gas flows of the furnace was reduced, obtaining gains in the consumption of electric energy of the fans.

- Solid *fuels, anthracite, coke breeze, etc*., were introduced into pelletizing in the late 1970s, when there was an accelerated increase in the price of fuel oil, drastically affecting the pelletizing activities of hematite ores. The pelletizing of magnetite ones was much less affected, given the low consumption of heat energy verified for these ores, as already discussed in chapter 7. Solid fuels were exhaustively studied for a combination with hematite ores since magnetite ores already had much more than enough internal heat. This heat comes from the oxidation of magnetite. Tests showed that the magnetite ores did not combine with solid fuel dosages, with a worsening in the quality and productivity of the furnace. In the case of hematite ores, the addition of solid fuel evolved to the level of 1.25% of fixed carbon, the limit point of dosage. The solid fuel dosage accelerated the agglomeration reactions, resulting in a great increase in the productivity of the traveling grate furnaces and a significant reduction in heat and electricity consumption. By introducing a large amount of heat into the pellets, solid fuels made it possible to reduce the volume of hot gases flowed by the furnace fans, responsible for the heat exchange between the hot gases and the pellet bed. The gains in electrical energy in the furnace grew with the increase in the dosage of solid fuel, so that each plant sought to establish the highest dosage level, which was compatible with the quality specifications of its products. In the case of hematite ores, it is verified that the dosage was in the range of 1.0 to 1.2 % of fixed carbon. (1)

- The *type of pellet* has an influence on the electrical energy consumption of the furnace. Generally, the lower the volume of basic slag (CaO + MgO) the higher the productivity of the furnace and the lower the consumption of electricity. These basic compounds usually come from limestone and other carbonates and must be calcined in the furnace and reacted to the formation of slag. MgO, in turn, is a refractory compound that raises the melting point of the slag components. Thus, these compounds delay the agglomeration reactions, being factors of productivity decrease of the furnace, with consequent increase of the consumption of electric energy.

- *The type of kiln* used, *Grate-kiln (GK) or Traveling Grate (TG)* (10), influences the consumption of electricity. In this respect, grate-kiln has an advantage over traveling grate. In the GK, the layer of pellets in the drying and pre-firing grate is much smaller than that found in the TG. Thus, the pressure loss of the hot gas flow is much lower in the GK, leading to a lower consumption of electrical energy in this type of furnace.

Having made these considerations, it is worth highlighting some typical data of specific consumption of electric energy for the various stages of the pelletizing process. (1, 10) Table 8.1 shows the main typical specific consumptions related to the various stages and unit operations of the pelletizing process.

From the data in this table, it is verified that it is very important to develop conceptual studies when designing a pelletizing project, in order to minimize the consumption of electricity in the industrial plant.

Data from plants in operation (1, 10) show total electricity consumption ranging in the range of 45 to 65 kwh/t of pellets, for circuits with all three stages of pelletizing, preparation-balling-burning, and from 25 to 40 kwh/t of pellets, where the circuits contemplate only the balling and burning stages.

Table 8.1 – Specific consumption of electricity, typical of industrial pelletizing plants.

STEPS / UNIT OPERATIONS		CONSUMPTION (kwh/t)	REMARKS
1 - ORE PREPARATION:			
1.1 - Grinding/ ball mills	. *Dry*	20 a 30	. Heavily dependent on ore grindability
	. *Wet*	10 a 20	. Combination with roller press and tower mill optimize electric consumption in ore grinding step
1.2 -Filtering	. *Vacuum Filter*	4 a 5	. Conventional filter - vertical discs
	. *Ceramic Filter*	1 a 2	. Vertical ceramic discs
2 - MIXING AND BALLING:			
2.1 - Mixing		1 a 2	. Depends on the type of mixer
2.2 - Balling		2 a 3	. Depends on return rate of balling section
3 - PELLET FIRING			
3.1 - Furnace equipment		0,5 a 1	
3.2 - Fans	. *Traveling Grate*	17 a 23	. Highly dependent on the ore type, control of gas flow, use of solid fuel, kind of pellets, etc.
	. *Grate-Kiln*	10 a 15	
4 -OTHERS			
4.1 - Additive Grinding (per t of additive)		4 a 5	. In plants where they are ground separetely
4.2 - Dedusting		2 a 4	. Depends on the type of equipment
4.5 - Auxiliary Equipment and Stockyard		5 a 7	

CHAPTER 9

ASPECTS RELATED TO WATER CONSUMPTION IN PELLETIZING (1)

Water is a fundamental element in the pelletizing of iron ore. Although other unit operations can be conducted in dry basis, it would be impossible to form the green pellets on the discs and drums without the presence of water. As is well known in the literature, (11) the surface tension of the water, acting on the capillary ducts of the pellets in formation, confers resistance to the agglomerate. Water also plays other important roles in pelletizing plants, such as: cooling agent of equipment, element for minimizing particulate emissions in environmental control, etc. It is evident that, *in wet circuits*, additional amount of water is the main agent for the formation the ore pulp, in preparation steps of pelletizing.

The water demand in an industrial plant depends on some important factors, such as:

- Flow chart;
- Type of circuit, *dry or wet*;
- And so on.

As for the flowchart, there are basically two types of plants: those that consider the 3 stages of the process, preparation-balling-burning, and *others that receive the ground ore, ready for use, and that contemplate only the steps of* balling and burning. The later, require lower amounts of water.

Regarding the type of circuit, the one with *dry ore preparation* implies the fact that the moisture of the pellet feed is lost in the drying and that the ground material is delivered to the balling section with residual moisture, close to zero. In this case, there is a need to add to the ground ore the optimal moisture content for the balling step, a value commonly in the range of 8% to 10%, which requires a lot of water from the supply source. In turn, considering that *the wet circuit* takes advantage of the water that comes with the pellet feed (6% to 8%) and that the water circuit in the preparation stages is closed, with a high level of reuse, it can be estimated that the water demand of this type of circuit is*, theoretically,* lower than that with dry operations. The major problem of wet circuits, which can cause considerable increases in water consumption, is the possible imbalance in grinding and filtering operations. An example is when the production rate of the grinding mills is higher than that of the filtration, with consequent filling of the homogenizing tanks and the need to stop these same mills. Continuing the filters to operate, with the mills stopped, the filtrate water overflows from the thickener to process water tanks and, from there, to the disposal and/or recovery basins, but with losses.

To illustrate these situations, the block diagrams of figures 9.1 and 9.2 were drawn, highlighting wet and dry pelletizing circuits*,* contemplating the three basic stages of processing: preparation, balling and burning.

As can be seen in Figure 9.1, in pelletizing with wet ore preparation, there are a number of interconnected unit operations, *grinding-classifying-thickening-homogenization-filtration*, where there is a *closed system of water recirculation*. In operational equilibrium, there is a minimum loss of water in this stage of preparation of the ore for balling. The moisture water (6% to 8%) that reaches the system with the pellet feed is incorporated into the flow of this circuit. In the last phase of the ore preparation, the filtration, the material is obtained with moisture as close to the optimum point for the formation of the green pellets, (8% to 10%), so that in the mixing and balling only a small final adjustment of this moisture is made. At this point of processing, filtration-mixing-balling, there is, therefore, a demand for water, equivalent to the difference between the humidity of the pellet feed and the green pellet. In a rough estimate, this water consumption would be equivalent to a value of the order of 2 to 4%, or something like <u>*20 to 40 l/t of pellet,*</u> (0.020 to 0.040 m3/t). In this way, any additional consumption that occurs in a pelletizing plant, *with wet circuit*, can be interpreted as <u>*possible loss*</u> and, therefore, must be continuously studied and minimized.

On the other hand, as shown in Figure 9.2, in the pelletizing circuit with dry ore preparation*,* all moisture water from the pellet feed is lost, due to the drying operation, before grinding. Therefore, to obtain the green pellets, there is a need to incorporate water to the ground and dry material in the mixture, making the final adjustment in the balling section. Considering the optimal moisture content in the range of 8% to 10%, there would be a water consumption in the range <u>*of 80 to 100 l/ t of pellet*</u> (0.080 to 0.100 m3/ t). As in the previous case, all additional water consumption should be analyzed as a possible loss, deserving continuous control and study for its minimization.

In these two cases, open water consumption systems were not considered, such as: sprinkling water on ore piles and pellets to control the emission of particulates, humidification of burned pellets for shipment to customers, other open systems without recirculation, etc.

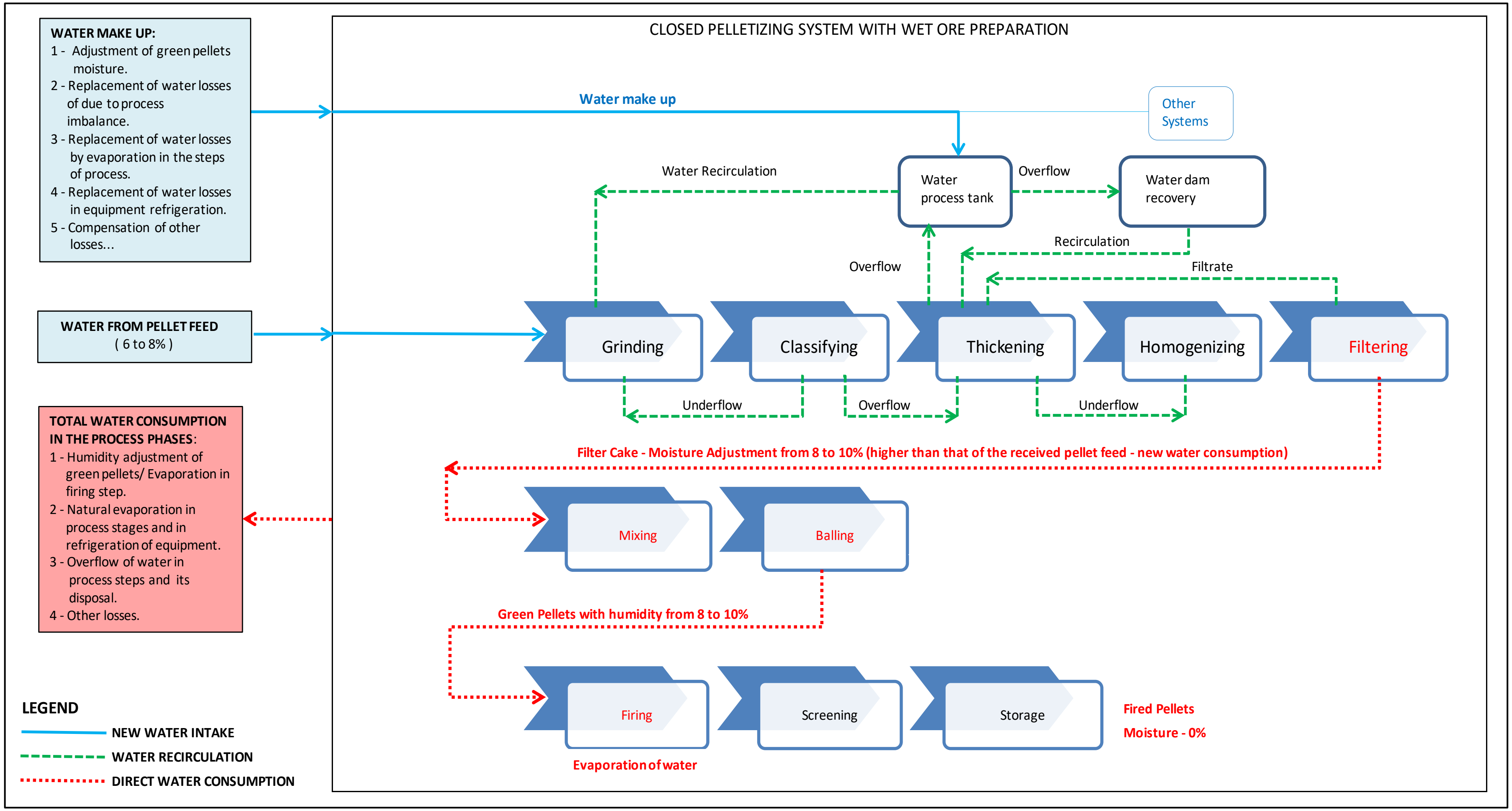


Figure 9.1 - Block diagram of wet basis pelletizing plant, showing, generically, the systems of supply, recirculation and water demand.

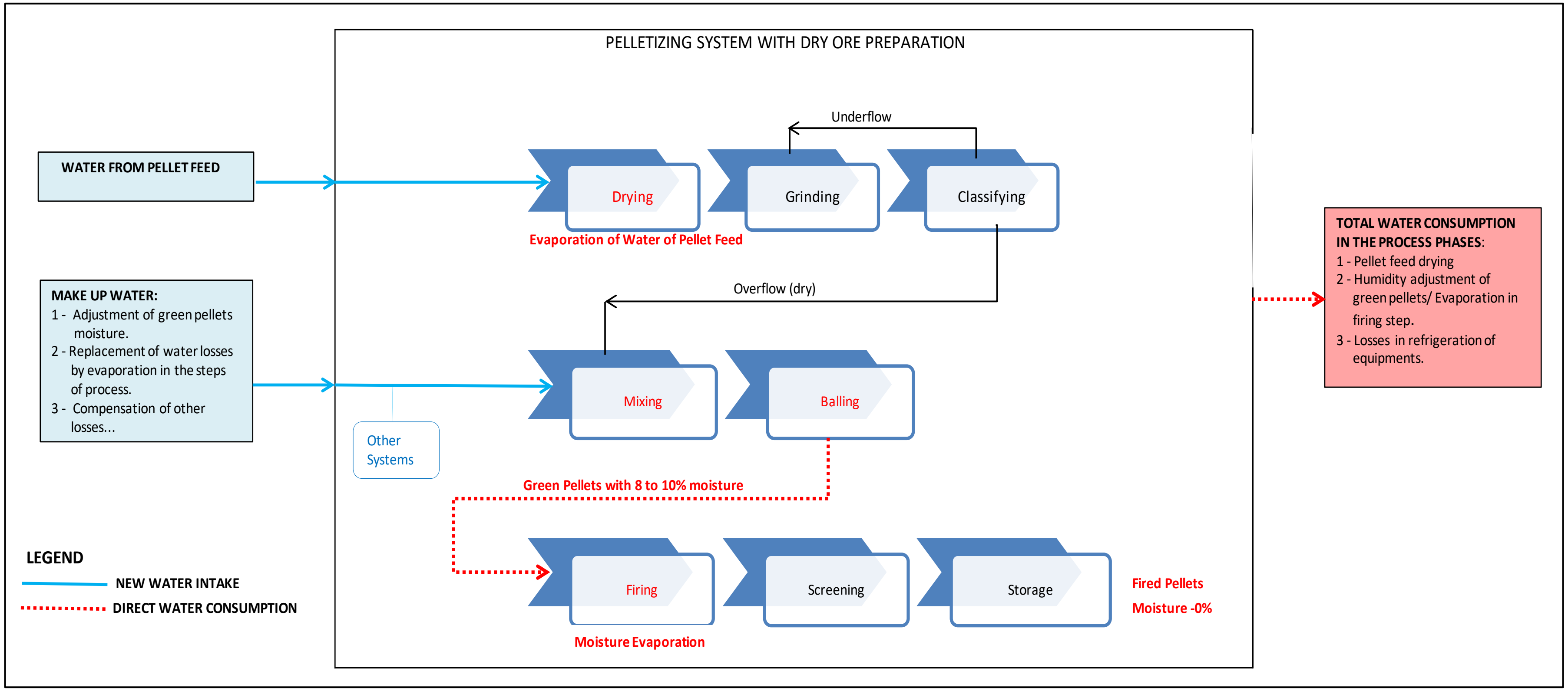


Figure 9.2 – Block diagram of pelletizing with dry ore preparation, showing generically the water supply and demand systems.

CHAPTER 10

SPECIFIC SURFACE OF MATERIALS FOR PELLETIZING (1)

10.1 – *Conceptual Aspects.*

For pelletizing, iron ore and additives need to be ground to such a point that it is possible to obtain green and burnt pellets with quality that meets the requirements of the process and the market. The grinding operation is one of the most expensive of pelletizing, requiring a lot of electricity and consumables. Thus, a strict control of this operation is established, so that there is neither deficient grinding of the materials, with a negative impact on the quality of the pellets, nor excessive, which, on the other hand, increases the operating cost. Considering the different materials of pelletizing mixtures, iron ore, fluxes, binders, and solid fuel, what degree of grinding should be given to each of them? This is an issue that can only be solved by running tests at laboratory and pilot scales. Operational best practices indicate that due to the large difference in true density of these materials, grinding should be conducted separately.

Considering that iron ore is the main and largest constituent of a mixture to be pelletized, the definition of optimal fineness focuses on this material. In this definition, among other parameters, the average particle size of the ore is determined. After the characterization of the optimal fineness of the iron ore, good practice considers that the other additives are ground at least to the same degree as the ore, that is, with the same *average particle size*.

As previously mentioned, each iron ore has its own characteristics, which influences its behavior in unit processing operations and pelletizing. Thus, each type of ore has its optimum fineness for pelletizing. From the literature, it is known that there are several methods to measure the fineness or even the state of size of mineral particles. *Different methods lead to different results*. In the case of pelletizing, two types of tests are commonly performed for this evaluation:

- Determination of the largest sizes by sieving in sieves of 100, 200 and 325 meshes, and of the complete particle size distribution, through different methodologies (sedimentation, cycloning, laser beams, etc.);
- Measurement of specific surface, through air permeameters (12,13,14).

Combined, the results of these two types of tests allow to establish the control parameters of the fineness of the materials, for the pelletizing process.

The specific surface (SE) was so designated because it represents the relationship between the outer surface area of the particles (A) and their mass (M).

It could also be referred to volume, which is less common. Thus, the equation that defines the specific surface of a ground material is given by:

SE = A/M, (usually expressed in cm^2/g in the CGS system). Eq. 10.1

The specific surface, being a parameter of easy and fast determination, has become one of the most important indexes of fineness control of materials for pelletizing. In particular, the measurement methods developed by Blane and Fisher (12, 13, 14) based on the permeability of a porous bed of particles to the passage of air, prevail in industrial units. Such methods do not consider the form factor of the mineral particles, nor the roughness and porosity, which in a way implies some inaccuracy in the measurement.

Specifically, Fisher, in his studies, regarded the particles as spherical. He constituted beds compacted in cylindrical tubes and subjected them to the passage of controlled air flows. Thus, it was established standardized test conditions in which correlations were verified between the average diameter of the spherical particles and the air flow pressure loss, indicative of the permeability of the particle bed. The measuring instrument (13, 14), developed by Fisher determines the average diameter (Dm) of the mineral particles.

Taking the diameter (Dm), the specific mass of the material under consideration (μ) and considering the spherical shape, according to Figure 10.1, it is possible to calculate the general equation of the specific surface, SE = f(Dm, μ).

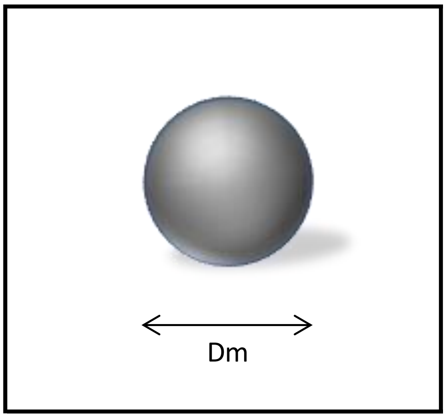


Figure 10.1 – Spherical particle with diameter Dm

From geometry, one has that:

Area of the spherical cap = A = $\pi . Dm^2$, and

Volume of the sphere = $V = \pi . Dm^3/6$

Hence, it can be written that the mass of the sphere is equal to:

$M = \mu . V = \mu . \pi . Dm^3/6$

From the equation Eq. 10.1, the specific surface is equal to:

$SE = A/M = \pi . Dm^2/(\mu . \pi . Dm^3/6)$

Simplifying this equation yields:

$SE = 6/(\mu . Dm)$ Eq. 10.2

Where:

μ - true specific mass in g/cm^3,

Dm – average diameter of the mineral particle in cm, and

SE – specific surface in cm^2/g.

Since the Fisher measuring instrument provides the average diameter in micron (10^{-4} cm), one can adjust the equation Eq. 10.2 to the following general form:

$SE = 60,000/(\mu . Dm)$ Eq. 10.3

Where as:

μ - true specific mass of the material in g/cm^3,

Dm – average diameter of the mineral particle in micron, and

SE – specific surface in cm^2/g.

Considering that in pelletizing there is a mixture of iron ore with fluxes, binders and sometimes solid fuel, specific surface curves could be drawn as a function of the average diameter of particles for these different materials. With these curves, the specific surfaces of these additives could be defined, so that the average diameter of their particles would be the same as that of iron ore.

Just to illustrate, consider the true specific masses of hematite iron ore, limestone, and anthracite, respectively: 5.0; 2.1 and 1.5 g/cm^3. Taking such values into the general equation Eq. 10.3, the following three equations result for these materials:

$SE = 12,000/Dm$, relative to iron ore, Eq. 10.4

SE = 28,571/Dm , relative to limestone, and Eq. 10.5

SE = 40,000/Dm, corresponding to anthracite. Eq. 10.6

Figure 10.2 shows the graphical representation of the Specific Surface as a function of the average diameter of the particles for these three materials, iron ore, limestone, and anthracite. Observing this figure, it can be easily seen that:

- The smaller the average diameter of the mineral particles, the higher the value of the specific surface, regardless of the material considered.
- For the same average diameter, the specific surface value is higher for materials of lower specific mass.

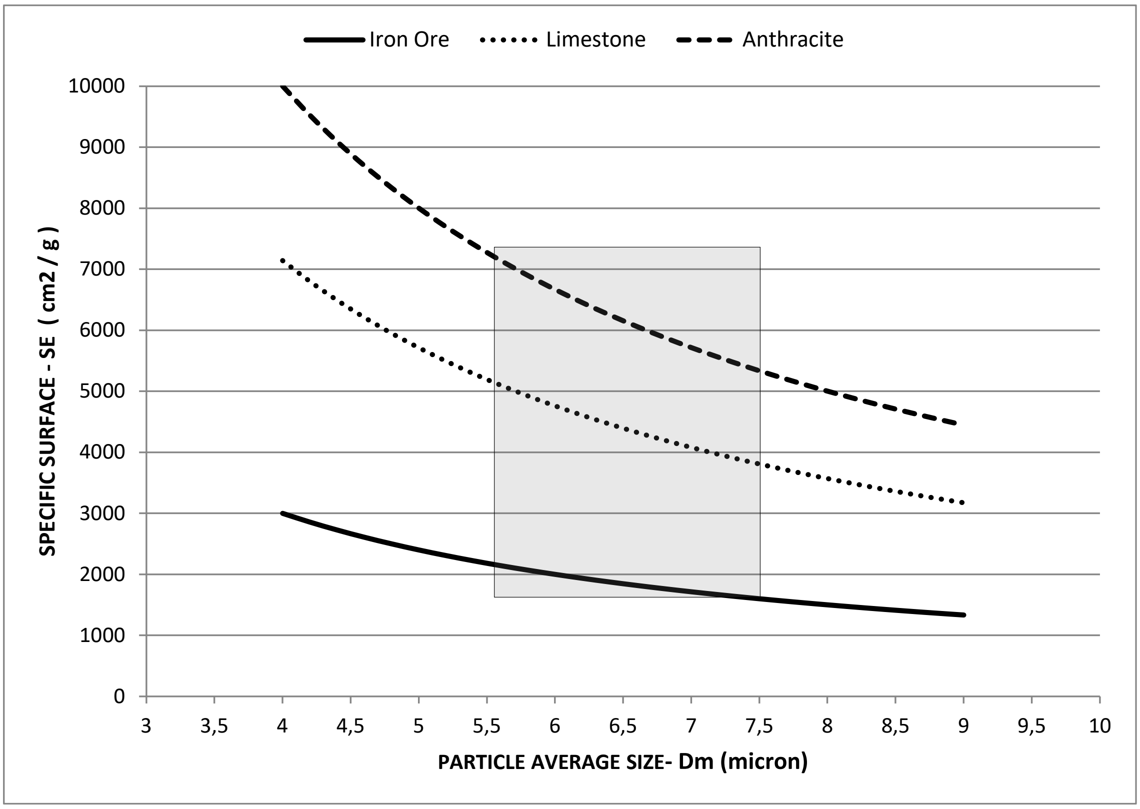


Figure 10.2 – Specific surface as a function of the average diameter of the particles

The optimal specific surface for pelletizing depends on the type of ore and the quality specifications of the burned pellets.

It is verified that, in the various industrial plants, this parameter varies in a very wide range, such as: 1,600 to 2,200 cm^2/g.

Thus, the average diameter of the particles of ore milled and ready for pelletizing has a corresponding variation, being in the range of 5.5 to 7.5 microns, values that are easily calculated by the equation Eq. 10.4.

Considering the best industrial practices, additives should be ground to at least the same average size as iron ore. Thus, through the graph of Figure 10.2, it can be verified that limestone and anthracite need to be prepared with specific surface, in the approximate ranges of 4,000 to 5,000 cm^2/g and 5,000 to 7,000 cm^2/g, respectively. The indicative field of these ranges of variation, corresponding to iron ore, limestone, and anthracite is indicated by the shaded area in Figure 10.2.

The strength of the green pellets, formed in the balling discs or drums, depends on the size of the mineral particles used in their composition. Thus, this resistance depends not only on the state of size of the iron ore particles, but also on the additives.

According to Rumpf (15, 16), the resistance of the green pellet is given by the following expression:

$$\sigma = k' . [(1-\varepsilon)/\varepsilon] . \Upsilon . (1/Dm) . \cos\phi_{(L-S)} \qquad \text{Eq. 10.7}$$

Where:

σ - rupture stress of the green pellet,

k' – constant that depends on the ore, liquid, etc.,

ε - porosity of the green pellet (fractional),

Y – surface tension of the liquid,

Dm – average diameter of mineral particles,

$\cos\phi_{(L-S)}$ – liquid-solid contact angle.

Through the studies developed by Rumpf, it can be verified that the resistance of the green pellet ***(σ)*** *increases* with the *decrease of the average size of the mineral particles* and the porosity. On the other hand, it increases with the surface tension of the liquid and with the reduction of the liquid-solid contact angle (ϕ), or the same as with the increase of the wettability or hydrophilicity of the mineral particles.

In this last aspect, it is important to highlight that balling only occurs when the water wets the mineral particles that must have the hydrophilic character ($\phi = 0^o$). In this condition, the capillary forces of cohesion develop, mainly responsible for the formation of green pellets. As the iron ore arrives at the balling section from other processing operations, such as concentration and dewatering, where chemical additives are used, it is important to verify that the hydrophilicity of the mineral particles has not been mischaracterized. Typically, amines, used in the concentration of iron ore for silica flotation, and additives used in filtration operations, residual and diluted in the moisture of the ore, can alter the hydrophilicity and therefore the ballability of the material.

Taking the expression of the Specific Surface of the equation Eq. 10.3, comes:

$SE = 60{,}000/(\mu . Dm)$, or the same as: $SE = k''/(\mu . Dm)$. From this, it follows:

$Dm = k''/(\mu . SE)$

Putting Dm in the Rumpf expression of the equation Eq. 10.7 results:

$\sigma = k'.[(1-\varepsilon)/\varepsilon].Y.(1/Dm).\cos\phi_{(L-S)}$ or $\sigma = k'.[(1-\varepsilon)/\varepsilon].Y.(\mu . SE/k'').\cos\phi_{(L-S)}$

Simplifying, it results the generic expression of the strength of the green pellet:

$$\sigma = k . SE . \mu . \Upsilon . \cos\phi_{(L-S)} . (1-\varepsilon)/\varepsilon \qquad \text{Eq. 10.8}$$

Where:

σ - rupture stress of the green pellet,

k – constant that depends on the ore, liquid, etc.,

SE – specific surface of the balling mixture,

μ - true specific mass of the particles in the mixture,

Y – surface tension of the liquid,

$\cos\phi_{(L-S)}$ – liquid-solid contact angle,

ε - porosity of the raw pellet (fractional).

Thus, it is verified that the Strength of the green pellet (σ) increases with the growth of the Specific Surface (SE) of the mixture of minerals in balling.

Some researchers and scholars of the subject of balling of iron ore prefer to refer the calculation of the specific surface to the unit of volume and not to the weight, as has been developed so far. The justification is that, proceeding in this way, one has an easier notion of the state of fineness between materials of very different specific masses. There are some industrial plants that also adopt this procedure. Within this concept, the definition of specific surface (SE') would become: "the *relationship between the area of the outer surface of particles (A) and their volume (V)*".

$SE' = A/V$ (usually expressed in cm^2/cm^3, in the CGS system) Eq. 10.9

Similarly in the previous calculations, in which the particles were considered as spherical (Figure 10.1), we have from the geometry:

Area of the spherical cap = $A = \pi . Dm^2$, and

Sphere volume = $V = \pi . Dm^3/6$

Substituting the expressions of A and V in the equation Eq. 10.9, results:

$SE' = \pi . Dm^2 / (a.Dm^3/6)$

Simplifying, it comes:

$SE' = 6/Dm$

As the measuring instrument of the average diameter provides the value in micron (10^{-4} cm), one must adjust this last equation, to have the result in the CGS system, resulting:

$$SE' = 60{,}000/Dm \qquad \text{Eq. 10.10}$$

Where:

SE' – specific surface, in cm^2/cm^3,

Dm – average diameter of mineral particles, in micron.

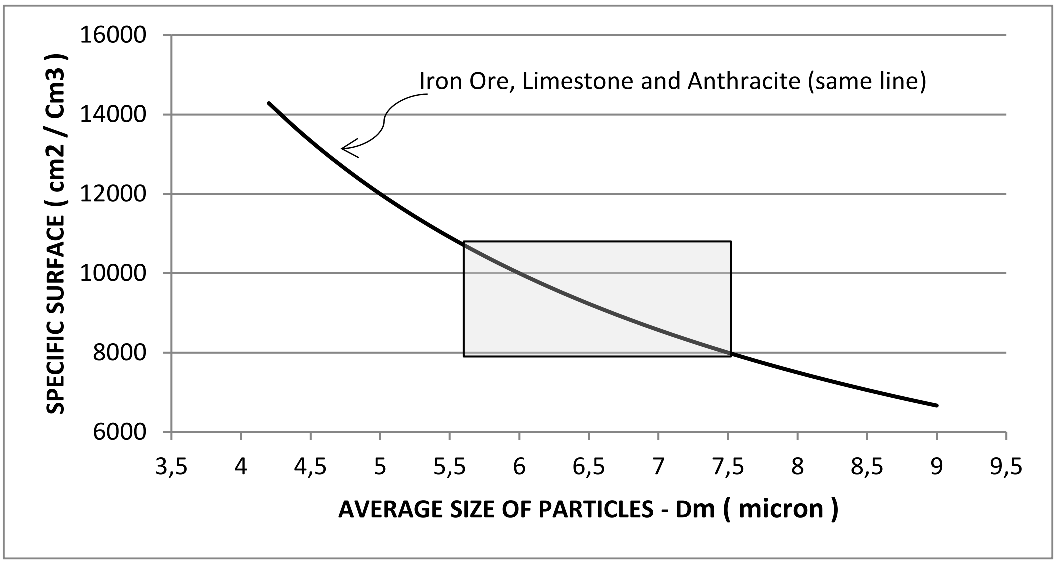


Figure 10.3 – Specific Surface in cm^2/cm^3 versus average diameter of mineral particles, showing the independence of the type of material.

In this conception, it is verified that the calculation of the value of the specific surface (SE') is independent of the value of the specific mass of the material. Because of this fact, when drawing curves SE' = f(Dm) for different materials, they are coincident in a single one. Figure 10.3, prepared for iron ore, limestone, and anthracite, shows this fact, in contrast to Figure 10.2.

This seems to make much more sense than the calculation referenced to mass, in ***cm^2/g****, because different materials, with particles of the same average diameter, have the same specific surface value, when expressed in* ***cm2/cm3****.*

The specific surface referenced to the volume (SE') can be calculated by knowing the value of the specific surface related to the mass (SE) and vice versa. For this, it is necessary to know the true specific mass (μ) of the considered material. Thus, lets take the equations Eq. 10.3 and Eq. 10.10:

SE = 60,000/(μ.Dm) and SE' = 60,000/Dm

From the first equation, it can be written that:

Dm = 60,000/(μ.SE), inputting in SE' and simplifying, it results:

$$SE' = SE.\ \mu \qquad \text{Eq. 10:11}$$

Where:

SE' – specific surface, in cm^2/cm^3;

SE – specific surface, in cm^2/g, and

μ - true specific mass of the particles in g/cm^3.

Taking the Eq. 10.11 and the true specific masses of iron ore, limestone, and anthracite as 5.0; 2.1; and 1.5 g/cm3, respectively, it can be written that:

SE' = (5.0).SE , relative to iron ore; Eq. 10.12

SE' = (2.1).SE , relative to limestone; and Eq. 10.13

SE' = (1.5).SE, corresponding to anthracite. Eq. 10.14

With these equations, the graph in Figure 10.4 below can be drawn, showing the relation SE' = f(SE), for the considered materials. It is easy to see that, for the same value of SE'

(cm^2/cm^3), or the same average diameter of particles, the values of SE (cm^2/g) are very different for iron ore, limestone, and anthracite.

For reference, the shaded area was indicated in this figure, covering the range of surface variation (SE) of iron ore in industrial plants, that is: 1,600 to 2,200 cm^2/g or the same as 8,000 to 11,000 cm2/cm3. Hence, value matches can be made with the limestone and anthracite curves.

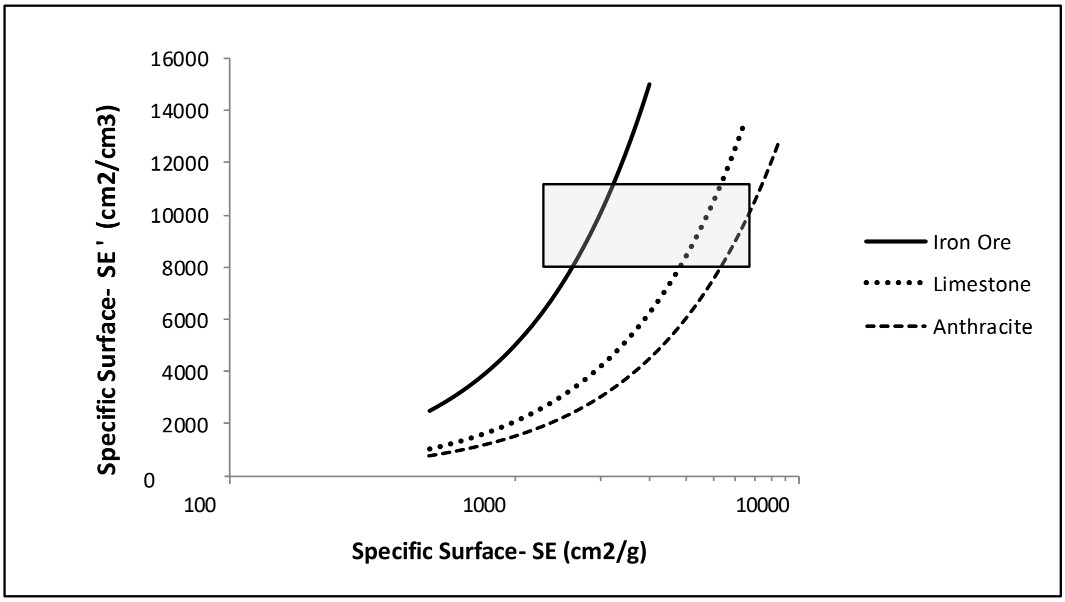


Figure 10.4 – Specific surfaces referenced to mass and volume for different materials, components of the balling mixture.

In pelletizing, fluxes, and other materials (solid fuel, for example) are dosed to the ore in different proportions, depending on the type of pellet that is considered. Good industrial practice suggests that these additives be ground into such a state that the average diameter of the particles is close to that defined for the iron ore particles. Thus, it would be the same as saying:

"*In pelletizing, the specific surface, expressed in cm^2/cm^3, shall have the same value for iron ore and additives.*"

The vast majority of research institutes, universities and industrial pelletizing plants adopt the concept of specific surface referenced to the mass of mineral particles (cm^2/g). It is not clear why this choice! Probably, the first scientists who studied the subject considered this reference, which spread in the technical environment. However, as seen, the volume-based concept allows an immediate notion of the relative state of size (Dm) of the different materials that make up a balling mixture.

10.2 – *Specific Surface and Ore Grindability*. (18, 19, 20, 21, 22)

For pelletizing, as seen in Chapter 2, iron ore must be prepared properly. Normally, to achieve the fineness of the material for the balling step, there is a need to subject it to a very fine grinding, usually conducted in a ball mill. This unitary processing operation is one of the most expensive in the pelletizing process, requiring an appreciable amount of electricity and also grinding media and mill liners. When the grinding is dry, there is still an additional consumption of thermal energy (heat).

As is known from the literature, in order to have a good stability in the operation of grinding, it is necessary that the feed ore presents small variability in terms of geo-morphological and granulometric characteristics.

However, these properties depend on the genesis of the ore and its physicochemical characteristics. Thus, different iron mining can generate pellet feeds for pelletizing with very variable grindability indices. Then, when the feed of the mills is composed of mixtures of ores of different origins, there is a need to know the grindability index of each component, in order to determine a standard feed mixture, with controlled variability.

The comminution or fragmentation of ores began to be studied in depth, from the 1840s, with the Austrian scientist and master *Peter Ritter von Rittinger*.

Later, around 1860, another scientist, also Austrian, *Friedrich Kick*, joined the theme. He developed a theory of comminution different from that of Rittinger, and there were great debates between them, in academic and industrial circles. In the mid-1930s, the American *Fred C.* Bond developed his preliminary studies on comminution, establishing equations and empirical laboratory tests for the calculation of the Bond Work Index. Due to the robust study developed, with large data content of industrial practice, Bond's theory spread in the design and optimization of equipment for grinding ores.

As the three theories developed by these scientists differed in fundamental aspects, many other scholars of the subject have dedicated much time on the evaluation and definition of the field of validity of each theory. In this sense, the Finnish *R.T. Hukki*, in 1961, published a technical article with the result of years of study and collection of industrial data, showing that the three theories are applicable and have their fields of validity, as shown in the graph of figure 10.5. In this figure, we can see the consumption of electrical energy in the comminution, as a function of the average particle size of the comminution and grinding product. It is also observed that *Rittinger's theory* has its validity in the field of the finest grinding, where the ground product has the average diameter of the particles less than 100 microns.

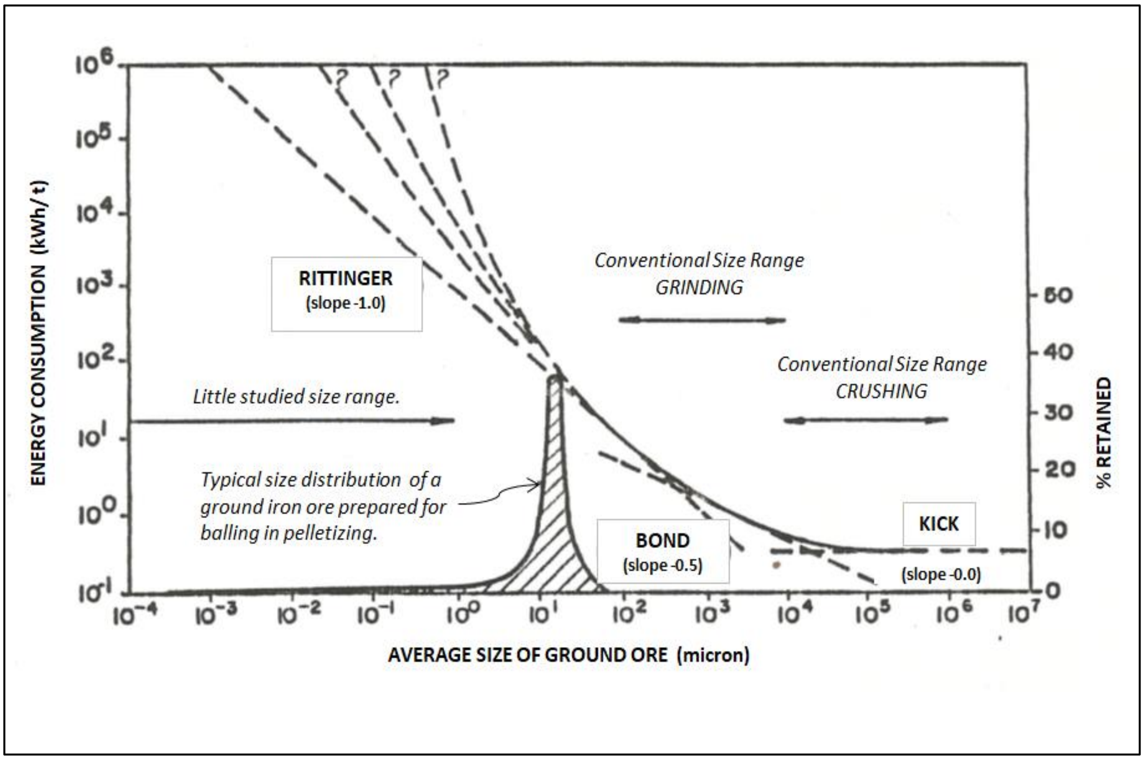


Figure 10.5 – Relationship between energy consumed in comminution and average particle size, according to Hukki - 1961. (Included a typical particle size distribution curve of ground iron ore for pelletizing.)

According to Hukki, Bond's theory is applicable to coarser grinding and Kick's to crushing operations, where particles are of larger sizes. It is not the purpose of this chapter to discuss and evaluate the different theories of comminution, which has been exhaustively done by many authors (22). Here, the intent is seeking to determine which of them is best suited to fine grinding for the pelletizing of iron ore. In this figure, it was plotted the typical curve of particle size distribution of a ground pellet feed with the fineness suitable for pelletizing (specific surface in the order of 1800 cm2/g). The positioning of this curve allows to verify that the grinding for pelletizing is more in accordance with the field dominated by Rittinger's theory.

In recent years, many scholars, including the author of this book, have dedicated themselves to the study of the subject, either in the evaluation of the behavior of iron ores in industrial grinding or in the development of grindability tests to characterize the different types of iron ore. (19, 20) As pelletizing is a technology governed by physicochemical phenomena of surface, it has been verified in such studies a better concordance of the Rittinger’s theory with the grinding of iron ore for pelletizing.

According to Rittinger’s Law:

" The energy consumed in ore grinding is directly proportional to the *new surface* generated ".

From this statement, it can be written that:

$$E = K'. (Sf - Si) = K'.\Delta S \qquad \text{Eq. 10.15}$$

Where:

E – consumed energy,

K' - constant that depends on the characteristics of the ore;

Si - initial surface area of the ore;

Sf - final surface area of the ore; and

$\Delta S = (Sf - Si)$ – new surface area generated by grinding.

Mourão and Stegmiller (19) after years of studies on the subject in the 1980s and considering the need to develop a *grindability test* for the characterization of the different iron ores, made use of the expression of Rittinger's theory, adapting it for this purpose.

Thus, the following adjustments were made to Rittinger's equation, Eq. 10.15:

$E = K'. (Sf - Si) = K'.\Delta S$, or

$1/K' = (Sf - Si)/E = \Delta S/E$. From this, it results:

$K = (Sf - Si)/E = \Delta S/E$

Finally, it comesthe final expression of the *grindability index*:

$$K = \Delta S/E \qquad \text{Eq. 10.16}$$

Where:

K – grindability index of iron ore ($cm^2/g.wh$);

Si – initial specific surface of the ore (cm^2/g);

Sf – final specific surface of the ore (cm^2/g);

$\Delta S = (Sf - Si)$ – generation of new specific surface area by grinding (cm^2/g),

E – amount of energy consumed to create the new surface area - ΔS, (wh).

Thus, through a standardized laboratory assay (19) it was possible to characterize the different types of iron ore, determining their characteristic grindability indexes.

This enabled the development of models to forecast the production rate of the industrial grinding mills and estimate the corresponding energy consumption in iron ore preparation for pelletizing.

Figure 10.6 shows, generically, how this index varies in a large amplitude, which is a function of the geo-morphological characteristics of the different ores.

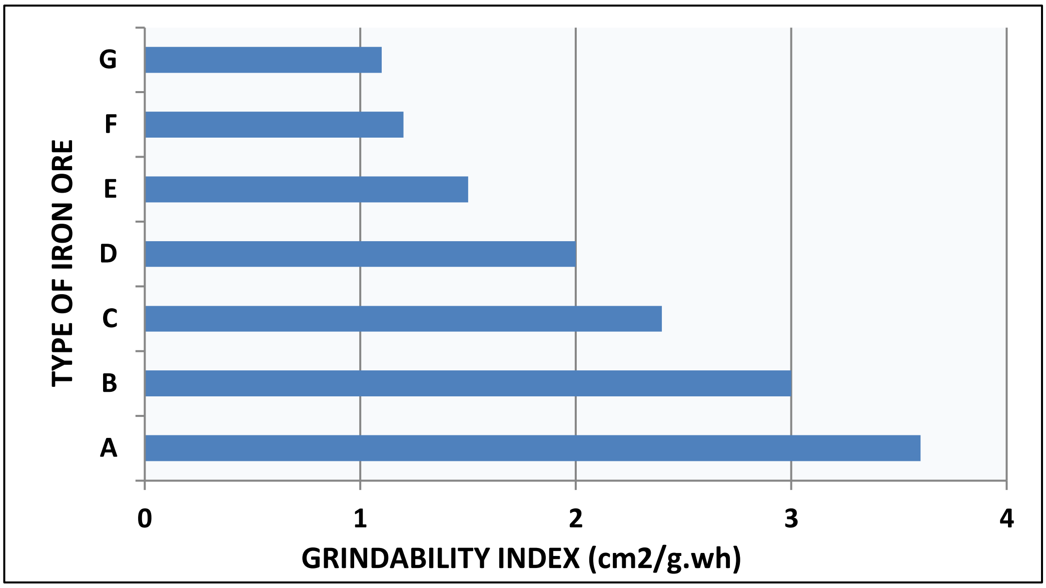


Figure 10.6 – Grindability index of some types of iron ore fines.

CHAPTER 11

ASPECTS RELATING TO THE DENSITY AND POROSITY OF PELLETS (1)

The issue of porosity is a matter of fundamental importance in the pelletizing of iron ore. Both the one related to the individual mineral particle and that related to a group of particles and/or pellets. There are several standardized methods for the determination of this property of materials, especially those developed by international organizations such as ISO, ASTM, etc. However, these methodologies lack greater detail in their development, which makes it difficult to understand the physical aspects involved.

The pellets, whether green or burned, are subjected to gas flows during burning in the pelletizing furnaces. As the raw material of the reduction processes, the burned pellets feed the steel reactors, coming into contact with streams of reducing gases. In these processes, the porosity of the individual or set of pellets plays a very important role in the performance and productivity of the furnaces.

11.1 – Density

The concept of density (d) is always linked to that of specific mass. Except for gases, the density of a material in relation to water is normally taken. It is defined as the ratio of the specific mass (μ) of a material to the specific mass of water at 4 oC (μa). At this temperature, the specific mass of water reaches the maximum value, equal to 1 g/cm3, (CGS system of units). Being the ratio of specific masses, the density is a dimensionless quantity. Because the density of the reference water is equal to the unit, the same number that expresses the specific mass of a material also expresses its density. According to this concept, it can be written that:

$d = \mu/\mu a$, or $d = \mu/1$

Hence,

$\mathbf{d = \mu}$ (dimensionless).

Three are the basic densities, defined in the technical literature and used in academic compendia and in industry in general:

- *Real density (dr) or True density*

The true density of a particle is that in which the individual porosity is zero. In fact, it would be the unit cell density of the mineral. The analytical methods for its determination use an approximation of this state, since they take a sample of the finely ground material, in which case the pores are eliminated to the maximum, as shown in Figure 11.1.

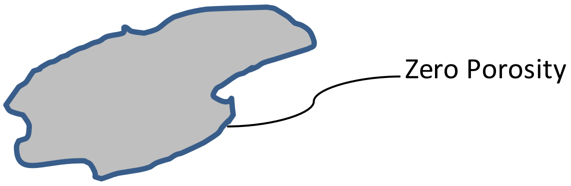


Figure 11.1 – Compact mineral particle with zero porosity.

Within the defined concept, the true density of the mineral particle would be given numerically by:

dr = ms/v , where: Eq. 11.1

dr – true density;
ms – mass of the solid (particle);
v – external volume of the particle.

When considering an iron ore, it has mineral impurities that have true densities quite different from that of the base iron mineral of the ore. Thus, the true density of an iron ore depends on the type and level of contaminants. When considering the mixtures to be pelletized, where there are different types and proportions of the additives, the true density should express the effects of these additives. In the case of burned pellets, there are different component phases in their structure. The predominant phase is always hematite, Fe_2O_3. In this case, too, the true density will be a function of the type of pellet and the nature of the other phases. Normally, in industry, correlations between the true density and the iron content of the mixture to be pelletized or of the pellets burned are determined, in order to facilitate the determination of other parameters characteristic of these mixtures and pellets. This is the case, for example, of the determination of the specific surface discussed in chapter 10, which depends on the true specific mass, of the same number as the true density.

- *Apparent Density (da)*

Consider that the particle in Figure 11.1 above contained internal pores, as shown in Figure 11.2 below:

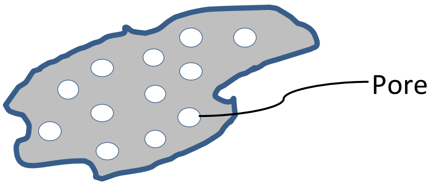


Figure 11.2 – Porous mineral particle.

The apparent density of this particle is defined by:

da = ms'/v, where: Eq. 11.2

da – apparent density;
ms' – mass of the solid (particle);
v – external volume of the particle.

As ms' < ms, due to the pores, the apparent density is always lesser than the true density (da < dr).

- *Bulk Density (dg)*

Bulk density is a property related to a solid material, in the granular state. For its determination, it is necessary to consider a set of granules or particles of the material, placed in a container, compacted or not, as the case may be. The bulk density is the ratio of the mass of the set of particles to the volume occupied in the container.

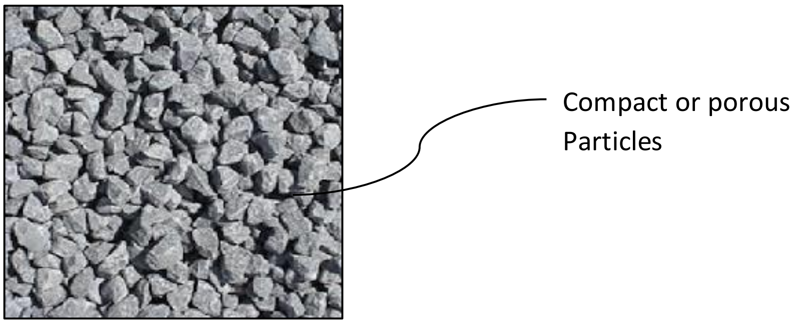


Figure 11.3 – Material in granular state

The bulk density is defined as being:

dg = m/V, where: Eq. 11.3

dg - bulk density;
m – mass of the granulated material;
V – total volume occupied by the granulated material.

Bulk density is widely used in calculations related to operations with minerals, ores, and pellets. There is a standardized methodology for its determination, which can be conducted with or without the compaction of the material. By compacting themselves, higher values are obtained. By the definitions, one can always write that:

dg < da < dr

11.2 – Porosity of Pellet

In the case of iron ore pellets, it is known that the material is more or less porous, depending on the constituents, the degree of grinding of the ore and additives, the intensity of burning, etc. When someone examines a pellet under a microscope finds that there are two types of pores, the closed ones and those said to be open, which communicate with the outside. The determination of open and closed porosity separately involves the use of a more complex methodology. Normally, in industrial operations, the *total porosity of the pellet* is determined, which considers the two types of pores and is an important parameter to control the pelletizing process.

Consider figure 11.4, which shows, schematically, a diametral section of a burnt pellet, highlighting the open and closed pores.

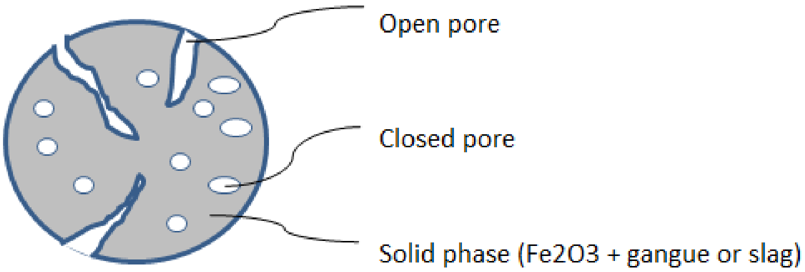


Figure 11.4 – Diametral section of a porous pellet.

The Total Pellet Porosity (P) is defined as the relative ratio between total pore volume and the pellet volume. Thus, it can be written that:

$P = (Vp/V).100\ \%$, where: Eq. 11.4

P – total porosity of the pellet in %;

Vp – total volume of pores (open and closed);

V – Volume of the pellet.

On the other hand, designating by Vs the real volume occupied by the solid phase of the pellet, it can be written that:

$Vp = V – Vs$

Substituting VP in the equation Eq. 11.4, results:

$P = 100. (V-Vs)/V$, or

$P = 100. (1-Vs/V)$ Eq. 11.5

Using the concepts of true density (dr) and apparent density (da), defined earlier, it is possible to write that:

$da = ms/V$, where ms is the mass of the solid phase of the pellet. From this, it follows that:

$V = ms/da$

Since Vs is the volume occupied by the solid phase, one can also write that:

$dr = ms/Vs$. From here, one can extract Vs, resulting in:

$Vs = ms/dr$

Taking the values of $V = f(ms, da)$ and $Vs = f(ms, dr)$ in the equation Eq. 11.5, results:

$P = 100. [1-(ms/dr)/(ms/da)]$.

Simplifying, the final expression of the total porosity of the pellet results:

$P = 100. (dr-da)/dr\ \%$, where: Eq. 11.6

P – total porosity of the pellet in %;

dr – true density of the pellet;

da – apparent density of the pellet.

Thus, the total porosity of a pellet can be easily calculated, knowing the true and apparent densities, which can be determined through methodologies standardized by international institutions, such as ISO, ASTM, etc.

The porosity of iron ore pellets is a very important property. A dry pellet, before burning in pelletizing furnaces, presents porosity in the order of 45 to 50%. As the burning processing is conducted at temperatures around 1300 oC, there is the formation of liquid slag, rounding of the hematite grains, contraction of the structure and elimination of pores. With this, there is a gradual reduction in porosity (or increase in apparent density), as the burning process progresses. Due to these facts, the porosity (or apparent density) can be used as an indication of the degree of maturation of the pellet structure or even the intensity of burning. Normally, well-consolidated pellets in the firing process present porosity in the range of 20 to 25%, depending on the constituents used in their production. If solid fuel dosing is used for green pellets, this range is extended to something like 25 to 35%, depending on the type and dosage of such fuel. This increase in porosity occurs because, when burning, this fuel leaves additional pores inside the pellets. In any case, the initial porosity of dry pellets, with or without solid fuel, decreases as the burning intensity advances. Figure 11.5 shows, generically, this evolution of the porosity of the pellet with the progress of the burning, considering the real density of the material equal to 5.0 and that the pellet had no addition of solid fuel, (anthracite, coke breeze, etc.).

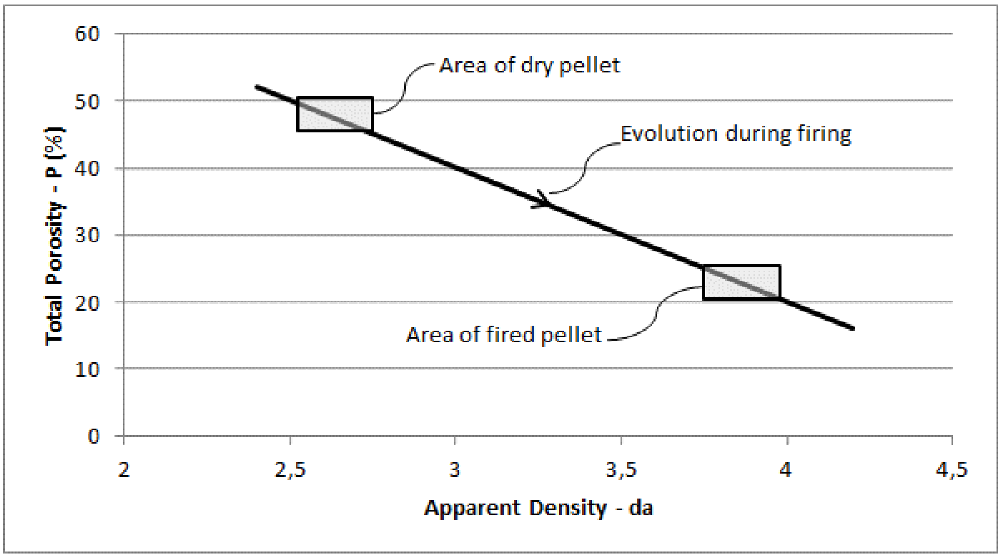


Figure 11.5 – Generic evolution of the porosity of pellet, during burning (without addition of solid fuel).

Porosity and apparent density are therefore indicators of the degree of maturation of the iron ore pellet structure, or even of the intensity of burning received (time and temperature). Higher burning intensity meaning lower porosity and higher apparent density.

Each type of pellet has a well-defined composition of raw materials, resulting in characteristic values of porosity and apparent density. Such evidence makes it possible to establish control parameters of the firing process in industrial pelletizing plants.

Care should be taken in this analysis when using the solid fuel dosage, because, if it is not homogeneous, it may distort the relationship between the intensity of burning and the porosity of the pellet, leading to antagonistic results. It would be the case that the pellet received, mistakenly, an excessive dosage of solid fuel and, consequently, presented a very high final porosity, even having suffered a marked intensity of burning.

11.3 – Total porosity of a pellet bed

The total porosity of a pellet bed is very important, both for the pelletizing process and for the pellet use in subsequent steel processes. In both cases, the pellets are subjected to gas flows and must present permeability compatible with the requirements of the processes. In pelletizing, the porosity of the bed facilitates the exchange of heat by convection with the hot gases. In the steel processes, it favors the reduction reactions of the pellets and the production of iron.

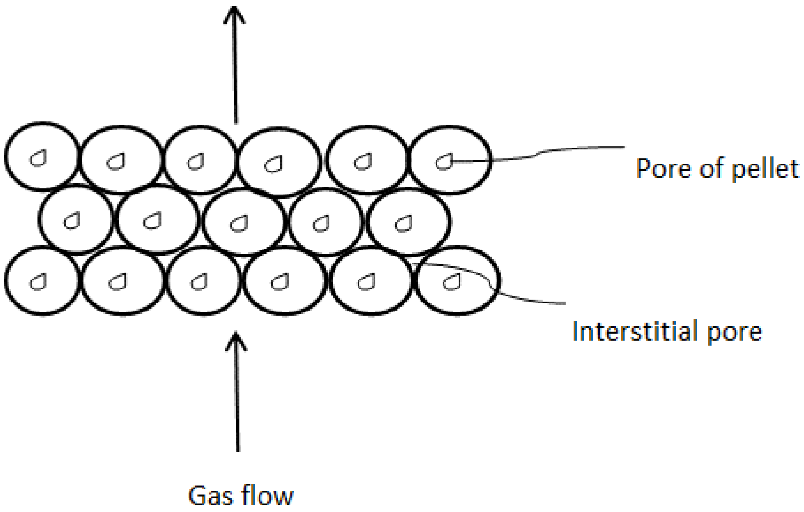


Figure 11.6 – Total porosity – interstitial pores + internal pores of the pellets.

Figure 11.6 shows a schematic drawing that shows the elements of the total porosity of a pellet bed, the interstitial pores and those of the pellets, which can be opened or closed, as previously evidenced. The pores of the pellets, especially the open ones, play a fundamental role, both in the processing of firing and during the reduction in the steel industry. During burning, moisture from green pellets is vaporized, carbonates are calcined, crystallization water is volatilized, solid fuel is burned, etc. These reactions generate significant amounts of gases that exit through the open pores. On the other hand, when used in steel reactors, the reducing gases penetrate through these pores of the pellets, accelerating the reduction reactions and the production of iron.

Analyzing Figure 11.6, and by analogy with the previous definitions of porosity and densities, it can be written that:

$Pt = 100.Vp/Vt\ \%$, where: Eq. 11.7

Pt – total porosity of the pellet bed (interstitial + interior of the pellets);
Vp – total pore volume (volume of interstitial pores + volume of pores of the pellets);
Vt – total volume occupied by the pellet bed.

On the other hand, the following relationship is true:

$Vp = Vt - Vs$, where *Vs is the total volume of the solid part*, component of the pellets.

Substituting this value of Vp into the equation Eq. 11.7, results:

$Pt = 100. (Vt-Vs)/Vt = 100. (1-Vs/Vt)$ Eq. 11.8

From the previous definitions of *true density (dr)* and *bulk density (dg)*, it can be written that:

$dr = ms/Vs$ and that $dg = ms/Vt$, where ms *is the total mass of the solid part of the pellets*. Developing, comes:

$Vs = ms/dr$ and $Vt = ms/dg$. Taking these values of Vs and Vt into the equation Eq. 11.8, results:

$Pt = 100. [1- (ms/dr)/(ms/dg)]$

Simplifying, the final equation of the total porosity of the pellet bed results, as a function of the bulk density of the pellets and the true density of the solid material, a component of those pellets:

$$\mathbf{Pt = 100. (dr-dg)/dr\ \%}$$ Eq. 11.9

Where:

Pt – total porosity of the pellet bed in % (interstitial + internal of the pellets);

dr – true density of the solid constituent of the pellets;

dg – bulk density of pellets.

For hematite pellets, produced with the addition of anthracite, it is common to have typical results of densities equal to: dr = 5.0 and dg = 2.0. Taking these values into the equation Eq. 11.9, it is verified that, in this case, *the Total Porosity of the Bed of these pellets would be equal to 60%.*

Pellets, with insufficient degree of burning, fragment in handling and transport operations, generating unwanted quantity of fines in the reduction reactors. These fines penetrate the interstices of the pellet bed, *increasing the bulk density of the material* and therefore *decreasing the porosity of the bed*.

Considering this bed of pellets in Figure 11.6, it would be important to know the interstitial porosity and porosity due to the internal pores of the pellets, (referred to the total volume of the pellet bed). Conceptually, it can be written that:

Pt = Pi + Pp %, being:

Pt – total porosity of the pellet bed;
Pi – interstitial porosity;
Pp – porosity due to the internal pores of the pellets.

It follows that:

Pp = Pt – Pi % Eq. 11.10

On the other hand, the following relationship is true:

Vt = Vi+Vp , where:

Vt - total volume of the pellet bed;
Vi - total volume of interstices;
VP – Total volume occupied by the pellets in the bed.

Developing, comes:

Vi = Vt-Vp , dividing everything by Vt yields:

Vi/Vt = 1 – Vp/Vt Eq. 11.11

From previous definitions, it can be written that:

Pi = 100.Vi/Vt, or Vi/Vt = Pi/100. It is true that:

Vp = ms/da and Vt = ms/dg , where:

Vp – Volume occupied by the pellets in the bed (includes the internal pores);

ms – mass of the pellets;

da – apparent density of the pellets;

dg – bulk density of pellets.

Replacing the variables Vi/Vt, Vp and Vt in the equation Eq. 11.11, results:

Pi/100 = 1 – [(ms/da)/(ms/dg)]

Simplifying the expression results the general equation for determining the *interstitial porosity* of a pellet bed:

Pi = 100. (da-dg)/da Eq. 11.12

Thus, using the equations Eq. 11.9, Eq. 11.10, and Eq. 11.12, it is possible to calculate the total porosity of the pellet bed (Pt), the porosity due to the interstices or voids between the pellets (Pi) and the porosity conferred to the bed by the internal pores of the pellets (Pp).

Just to illustrate, consider a typical case of iron ore pellets, in which the following values are determined for their properties:

- dr = 5,0 (true density);
- da = 3,6 (apparent density);
- dg = 2.0 (bulk density).

Taking to the three mentioned equations, the following values are obtained for the porosities, <u>relative to the corresponding pellet bed</u>:

- Total porosity (Pt) = 60%,
- Interstitial porosity (Pi) = 44%,
- Bed porosity due to pellet pores (Pp) = 16%.

It is important to note that the calculated porosity, due to the pores of the pellets, is relative to the total volume of the bed.

The porosity of the pellet, individually, is calculated by the use of the equation Eq. 11.6, previously defined as:

P = 100. (dr-da)/dr %

Thus, substituting in this equation the values of dr and da, we find the value of 28% for the porosity of the pellet, individually, (porosity referred to the volume of the pellet and not to the volume of pellet bed).

11.4 – Porosity and strength of green pellet (1, 15, 16, 17)

Porosity is one of the most important properties of iron ore pellets. The pores are true channels of movement of gases, involved in the burning and subsequent reduction of the pellets. These pores can be opened or closed. The most important open ones constitute the vast majority of those that exist.

The porosity of the green pellet, which depends on the densification and compaction of the mineral particles in the balling phase, is directly related to the rupture stress (pellet strength).

Studies on this topic have shown the dependence of the rupture stress (**σ**) of the green pellets with the porosity (15, 16) or, indirectly, with the degree of saturation (**S**) of the pores of the green pellet with the liquid involved in the balling stage (usually water with dilution of mineral salts, gel, etc.). Thus, different authors have shown that the strength of green pellets grows with the saturation of the pores (filling with the liquid), up to the maximum point, close to 90% (**S**) as shown, generically, in Figure 11.7. This point is part of the region characterized as a capillary zone, in which the pores of the green pellets act as true capillary vessels, developing the forces of cohesion between the mineral particles (**S** = 80 to 90%). In this figure, it is verified that, above 90% saturation, the capillaries are broken, and the strength of green pellet quickly drops to zero (17). Many treatises have been written on this subject (15, 16), not being the object of the present work.

The degree of saturation of the pores of green pellet (**S**) is defined as the percentage of the pores of the raw pellet that is filled by the liquid used in the balling (water and with or without dilution of mineral salts, other substances, gel, etc.). Thus, **S** is an indirect function of the porosity of the green pellet. It is important to note that, here, it is said of the open porosity, the one that can be filled by the liquid during balling. The closed pores of the mineral particles are disregarded or rendered null in this development.

Within this concept, it can be written that:

$$S = (Vl / Vp).100\ \%, \text{ where:} \qquad \text{Eq. 11.13}$$

S – degree of saturation of the pores of the raw pellet in %;

Vl – volume of liquid in the pores of the green pellet;

Vp – Volume of pores of the green pellet.

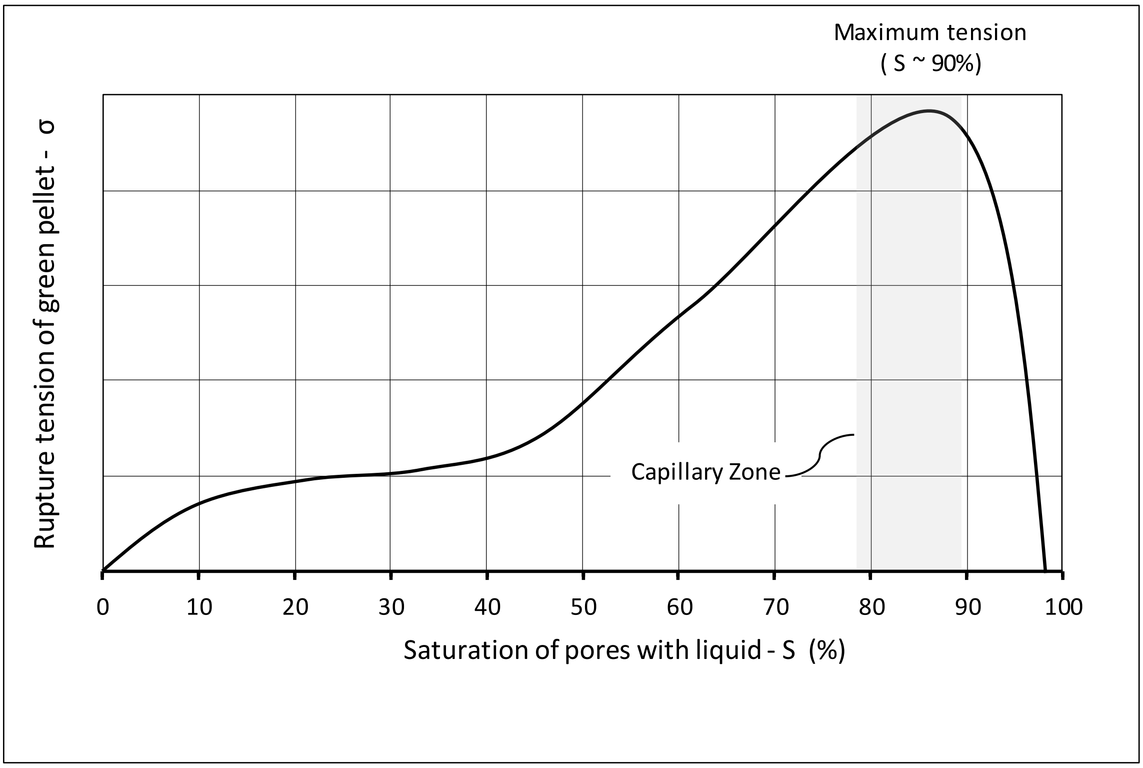


Figure 11.7 – Generic influence of the degree of saturation (S) of the pores with liquid on the rupture tension of the green pellet. (15, 16, 17)

The volume of liquid Vl is a function of the moisture of the green pellet (U%), known through determination in laboratory tests. Consider the liquid to be pure water, with specific mass equal to the unit. Thus, the following expression is true:

VI = f(U)

U = (Ma/M).100 %, where: Eq.11.14

Ma - mass of water, and

M - mass of raw pellet.

The mass occupied by water can be given by:

Ma = Va. μa

Where:

$Va = Vl$ – volume occupied by water in the green pellet, and

μa - specific mass of water, equal to 1 g/cm3.

As $\mu a = 1$, $Ma = Va$. Substituting Ma into the equation Eq.11.14 yields:

$U = (Va/M).100$

Hence, one has that $Va = Vl$, or:

$Va = Vl =(U.M/100)$ Eq. 11.15

Substituting $Va = Vl$ in the equation Eq.11.13 and simplifying yields:

$S = U.M/Vp$ Eq. 11.16

On the other hand, the porosity of the pellet is defined as being:

$P = 100.Vp/V = 100. (dr - da)/dr$ in % . Where: Eq. 11.17

P – porosity of the green pellet, in %,

Vp – pore volume of the green pellet;

V – volume of green pellet;

dr - true density of the mineral mixture of the green pellet,

da – (apparent) density of the green pellet.

From the equation Eq. 11.17 It is stated that:

$Vp = V.P/100$.

Substituting the value of Vp in the equation Eq. 11.16, results:

$S = (100.U.M)/(V.P)$ %

Since $da = M/V$, it follows that:

$S = 100.U.da/P$, in % . Eq. 11.18

From Eq. 11.17, one has the expression of $P = f(da)$, or:

$P = 100. (dr - da)/dr$

Explaining da, it can be written that:

$da = dr - P.dr/100$, or $da = dr.(1-P/100)$, then:

da = dr.(100 – P)/100 Eq. 11.19

Taking the value of da, from the expression Eq. 11.19, in the equation Eq.11.18, yields:

$$\mathbf{S = U.dr.\ (100\text{-}P)/P\ \%} \qquad \text{Eq. 11.20}$$

Where:

S – degree of filling of the pores of the green pellets by water, in %;

U – moisture content of the green pellet, in %;

dr – true density of the balling mixture;

P – porosity of the raw pellet, formed by the open capillary channels, in %.

In the case of hematite ores, with dosage of flux, solid fuel, and binder, one can consider the true density (dr) of this mixture, which generates the green pellets, as being something around 5. In this case, the equation Eq. 11.20, would take the following form:

$$\mathbf{S = 5.U.(100\text{-}P)/P\ \%} \qquad \text{Eq. 11.21}$$

Taking this equation Eq. 11.21, S = f(U,P), the graph of Figure 11.8 was drawn, as follows, highlighting the saturation field (S) in the range of 80 to 90%, when there is the occurrence of the capillarity regime and the maximum strength of the green pellet. This equation shows a very important aspect which is:

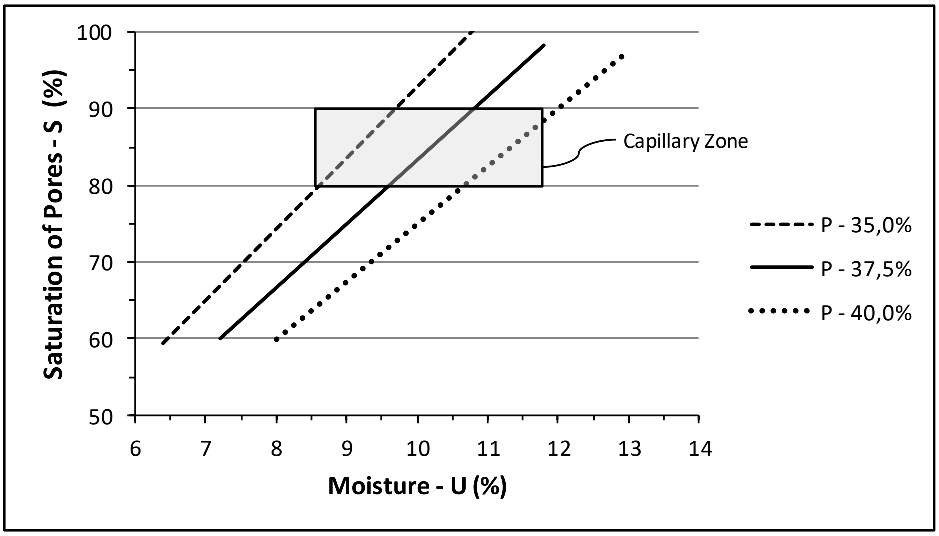


Figure 11.8 – Saturation degree of the pores of the green pellet, as a function of moisture content and porosity.

"The greater the densification of the mineral particles during balling, the lower the number of open pores (P) and the lower the moisture of the raw pellets (U), to achieve the same degree of saturation (S), or the same strength."

Lower humidity means lower consumption of thermal energy, in the next step of burning the pellets.

It should be mentioned that the porosity of the green pellet, in which the capillaries are filled with the pelleting liquid (water), and the particles are joined by various cohesion forces, is less than the porosity of the dry pellets (the evaporation of moisture from the green pellets creates new channels and pores).

CHAPTER 12

GASES INVOLVED IN DRYING AND BURNING OF PELLETS (1)

As seen earlier, iron ore pellets are composed of a mixture of the ore with fluxes, binders and, in the case of hematite, with a certain percentage of solid fuel. All materials are ground to a convenient degree of fineness, usually with particles below 0.074 mm, and moistened with water for the formation of the green pellets, so that the final moisture content is in the range of 8 to 10%.

The green pellets are formed as a moist agglomerate of such particles of different natures, with a complex system of capillary tubes, these tubes filled with water which communicate with the external environment. During the drying and burning of these pellets, there is a series of reactions that generates a large volume of gases inside them, which migrates, through the pores, to the external environment. In turn, oxygen from the hot drying and burning air penetrates the core of the hematite pellets (Fe_2O_3) for the burning of the solid fuel. In the case of magnetite pellets, it penetrates to oxidize Fe_3O_4 particles.

For these reasons, binders must be used in pelletizing that give the green pellets sufficient strength to withstand the internal pressures and tensions caused by these incoming gases and by those that evade the pellets in the drying and burning phases. A simple analysis of the phenomena that occur in the drying and burning of the pellets can show that the volume of these gases is very large, compared to the initial volume of the pellets.

Just for reference, take a *green hematite pellet* with a certain moisture content and additions of limestone and anthracite charcoal. During the drying and burning of the pellets, generally, the following reactions involving gases can occur:

12.1 – Evaporation of moisture.

The moisture from the green pellets is evaporated upon heating and the gases escape from the pellets, through the capillaries, or open pores. The chemical reaction governing this phenomenon is given by:

$$H_2O(l) + Heat \text{ -----------> } H_2O(v) \qquad \text{Eq. 12.1}$$

12.2 – Calcination of Limestone.

Limestone is calcined during burning, releasing a considerable portion of CO_2 in the gaseous state, which also evades from the pellets through the pores. The equation that governs the phenomenon is as follows:

$$CaCO_3 + \text{Heat} \text{ -----------> } CaO + CO_2(g) \qquad \text{Eq. 12.2}$$

12.3 – Burning of coal.

Generally, consider only the fixed carbon (Cfix) of coal. In this situation, this carbon is oxidized by oxygen from the air (21% O_2 and 79% N_2) that penetrates the pellet, also generating an appreciable amount of CO_2 that migrates to the outside, along with an equivalent amount of N_2 from the air. The chemical reaction that governs this reaction is as follows:

$$C_{fix}\text{ (coal)} + (O_2 + N_2)_{air} \text{ ------------> } CO_2(g) + N_2(g) \qquad \text{Eq. 12.3}$$

From this simple analysis, the following scenario results:

- Gases that penetrate inside the pellets: O_2 and N_2 (inert), in the proportion existing in the ambient air, which penetrate the pores of the pellets for the oxidation (burning) of the coal;

- Gases that evade from the pellets, through the pores, to the environment: H_2O from moisture evaporation, CO_2 from calcined limestone, CO_2 from coal burning and N_2, inert gas, from the air that has penetrated inside the pellets.

Figure 12.1 shows a generic visualization of these phenomena that occur during the heat treatment of *hematite pellets*.

There are scholars of the subject who have determined other models for the chemical reactions that occur during the burning of hematite pellets. They suggest that coal carbon is partially oxidized to CO by CO_2 from limestone decomposition. In turn, this generated CO partially reduces a fraction of Hematite to Fe_3O_4 and FeO. From there, these phases are fully reoxidized by oxygen in the air, regenerating hematite (secondary Fe_2O_3). In any case, in any model, there is the equivalent generation of CO_2 gas inside the pellets, which escapes to the outside, through the capillary pores.

Also, there is the migration of oxygen from the air (O2) from the outside to the inside of the pellets, for the oxidation of the phases, partially reduced (Fe3O4 and FeO).

Generically, the following sequence of reactions is suggested by these scholars:

- CaCO3 + Heat -----------> CaO + CO2 (this CO2 reacts with coal);
- Cfix + CO2 ----------------> 2CO (hematite reducing gas);
- Fe2O3 + CO ------------> (Fe3O4, FeO)+ Fe2O3 + CO2 (hematite reduction and CO2 release);
- (Fe3O4, FeO) + Air (O2, N2) -----------> Fe2O3 + N2 (hematite restoration and N2 evasion).

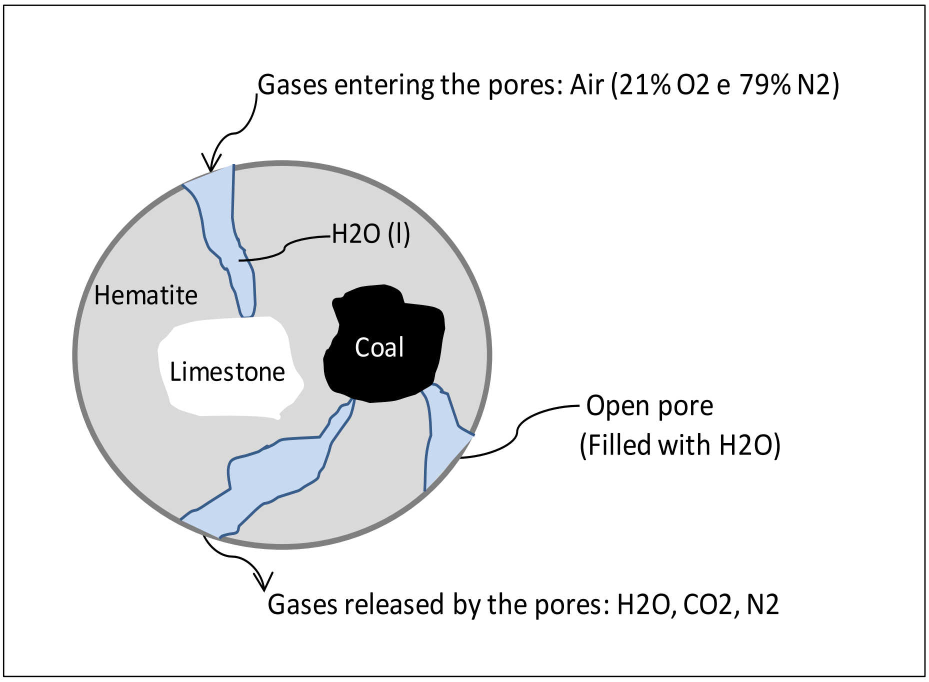


Figure 12.1 – Evolution of gases during drying and burning of hematite pellets.

To have a dimension of these volumes of gases, in relation to the initial volume of the green pellet, consider a pellet with a diameter of 12 mm, typical of the material produced in an industrial scale. Thus, the volume of this green pellet (V) would be given by:

$V = (4.\pi.6^3)/3$, or

V = 905 mm3, or

V = 0.91 cm3 (volume of an individual pellet, with diameter of 12 mm). Eq. 12.4

On the other hand, this green pellet is considered to have been produced having regard to the following specifications:

- Moisture content: 9%;
- Specification of the burned pellet: SiO_2 = 2.50% and CaO = 2.75%;
- CaO doser: limestone;
- Anthracite dosage: equivalent to 1.1% Cfix (dry basis);
- Bentonite binder dosage: 0.5%.

Taking these data to the mass balance (Excel spreadsheet), detailed earlier in chapter 6, the *composition of the balling mixture (%)* can be calculated. In this way, the following data result:

- Moisture content: 9.0%;
- Limestone dosage: 4.6%;
- Anthracite/coal addition: 1.3% or equivalent to 1.04% Cfix in the green pellet (wet basis);
- Binder dosage: 0.5%;
- Iron ore: 84.6%;
- Total: 100%.

Considering, just as an example, that the apparent density of a raw pellet of this nature is of the order of da = 3.8 g/cm3, (pores filled with water), and that the volume of the pellet under study, V = 0.91 cm3, it can be determined that its mass (M) is equivalent to:

M = da. V = 3.8 . 0,91, or Eq. 12.5

M = 3.46 g (raw pellet mass 12 mm in diameter).

Taking the percentage composition by weight of the raw pellet of 12 mm and the mass of this pellet, it is possible to calculate the *weights of water, limestone and fixed carbon,* component elements inside the pellet. Thus, it follows that:

- Water weight: Ma = (3.46) . (0.09) = 0.31 g; Eq. 12.6
- Limestone weight: Mc = (3.46) . (0.046) = 0.16 g; Eq. 12.7
- Fixed carbon weight: Cfix = (3.46) . (0.0104) = 0.04 g. Eq. 12.8

Considering what has been exposed so far and the physicochemical principles that govern the phenomena involved in the generation of gases during heating and burning of an individual green pellet of 12 mm in diameter, table 12.1 was drawn showing the corresponding set of calculated data.

By observing the results, it can be seen that, for this pellet of only 0.91 cm3 of volume, there is the following movement of gases, entering and exiting through its pore system:

- Volume of gases penetrating the pellet (air): 357 cm3,
- Volume of gases leaving the pellet: 779 cm3,
- Total volume of gases entering and leaving the pellet: 1,136 cm3.

In this way, it is verified how important it is to produce sufficiently strong green pellets, in order to resist the tension and stresses generated internally by these gases, in really, impressive volumes! Roughly speaking, it could be said that for each pellet of only 1 cm3, more than 1,000 Ncm3 or more than 1 liter of gases flow through its pore system.

The case of *magnetite pellets* is similar, but simpler, because there is no solid fuel dosage to green pellets. Thus, it is necessary to consider the evaporation of moisture from the green pellets, the oxidation of magnetite by the hot air of the furnace, and the decomposition of the fluxes and other possible hydrated minerals.

In the development of conceptual projects of pelletizing furnaces, this subject should be well studied in a pilot plant, especially when the iron ore is hydrated, limonite and/or goethite type, in which the crystalline water also releases from the pellet by its pore system. In pilot tests, the times and temperatures of the drying and pre-firing phases must be determined to preserve and guarantee the integrity of the pellets during the firing processing. In this particular, the speed of temperature increase in pre-firing is a factor of capital importance in the definition of the process parameters of this step.

In a more complex context, when considering the *self-reducing pellets*, where the dosages of coal, binders and fluxes are much higher, this problem of the flow of gases inside the pellets is more aggravated, requiring greater care *so that they do not disintegrate* during heat treatment.

Similarly, it is possible to estimate the complexity of the evolution of these gases in the sintering cake, given the large amount of fluxes and binders dosed to the mixture treated in this agglomeration process.

Table 12.1 - Generation and flow of gases in an individual green pellet with diameter of 12 mm (0.91 cm^3)

PROCESS	GERAÇÃO DE GASES / REAÇÕES	Input Volume		Output volume		TOTAL VOLUME
		cm^3	Gas	cm^3	Gas	cm^3
1 - Water evaporation	H20 (l) + heat -----> H20 (v) 18 g ------------------> 22,400 cm^3 0.31 g ----------------> X = (0.31) . (22,400)/18 = 386 cm^3			386	H2O (v)	386
2 - Calcination of limestone	CaCO3 + calor -----> CaO + CO2 100 g -------------------------> 22,400 cm^3 0.16 g ------------------------> Y = (0.16) . (22,400)/100 = 36 cm^3			36	CO2	36
3 - Burning of anthracite	C + O2 (air inlet) -----------> CO2 (outlet) 12 g ----------> 22,400 cm^3 -----------------> 22,400 cm^3 0.04 g --------> W ---------------------------> Z = W = (0.04) . (22,400)/12 = 75 cm^3	75	O2	75	CO2	150
	As the incoming air in the green pellet has N2, inert gas, and its composition is equal to 21% O2 + 79% N2, one can calculate the volume of N2entering and leaving the pellet: Volume of N2 = (volume of O2/0.21). (0.79) = (75/ 0.21) . (079) = 282 cm^3	282	N2	282	N2	564
TOTAL VOLUME of gases involved in burning of an individual green pellet: H2O(v) + CO2 + O2 + N2		357		779		**1136**

Obs: Volumes of gases calculated in the standard conditions, that is, at a pressure of 1 atm and temperature of 273 oK.

CHAPTER 13

PRODUCTIVITY FACTORS IN PELLETIZING (1)

The conventional pelletizing processes, traveling grate and grate-kiln, have a movable straight grate, as characterized and shown in chapter 2. For the evaluation of the productivity of the furnaces is used the area of these grates in m^2 as an important parameter in the calculations. Although the productivity values between the two types of process are not comparable, these numbers serve to evaluate the performance of furnaces of the same technology.

In general, the productivity of a pelletizing furnace is defined by:

$$Pr = P/(A.t) \text{ , where:} \qquad \text{Eq. 13.1}$$

Pr – furnace productivity, in t/m^2.day;

P – net production of the furnace, free of fines, in t;

A – useful or reaction area of the grate, in m^2;

t – time for the production of P, expressed in day or 24h.

The productivity can be impacted by several factors of the production process, such as:

- ✓ Stoppage of the grate due to electromechanical defects and other reasons;
- ✓ Type of pellet in production (direct reduction, blast furnace) and volume of additives;
- ✓ Type of iron ore (hematite, magnetite, limonite, etc.);
- ✓ Use of solid fuel in the pelletizing mixture;
- ✓ And so on.

- *<u>Furnace outages</u>*

The stoppages of the grate, and consequently of the pelletizing furnace, have a great impact on the productivity of the plant. Mainly, the unscheduled stops, caused by accidents, defects in equipment and other reasons. These accidental stops of the grate negatively affect productivity, since the continuous and stable production rhythm is broken, falling to zero instantaneously, as shown in figure

13.1. For the resumption of the previous production level, several operational actions are required, the responses to which are not immediate. Thus, the hourly production is growing gradually, during the period of reestablishment of the operational standard. With this, there is usually a significant loss of production, which is expressed by the hatched area of the graph in figure 13.1.

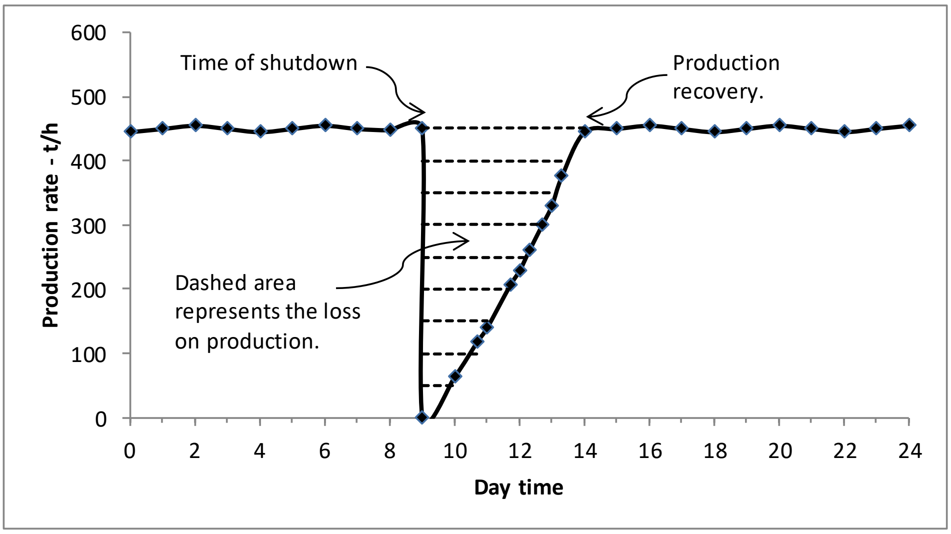


Figure 13.1 – Negative impact of grate shutdown on the productivity of the pelletizing furnace.

Normally, the owners of pelletizing technologies use in the development of the projects the operating time of 330 days/calendar year, safeguarding the remaining 35 days for corrective and scheduled maintenance, as well as other adjustments. With the evolution of pelletizing technologies, automation, control techniques and knowledge of unit operations, there are suppliers of traveling grate furnaces guaranteeing the annual operating time of 345 days, which represents a great advance in terms of annual production. Thus, the *variable time (t) to be considered in the calculation of productivity (Pr)* assumes a critical role in the evaluation, and may cause much discussion and controversy, if it is not well conceptualized and defined. As is well known, a pelletizing plant is composed of a series of operation units in addition to the kiln. Therefore, the following questions arise:

a) If there is a stoppage of the grate, caused by problems in other operation units than the kiln, how to treat this downtime?

b) Should productivity reflect the performance of the entire plant or just the pelletizing kiln?

c) Should productivity consider the calendar time, the scheduled time for operation or the effective time of operation of the plant?

d) Should productivity consider only the production and the effective time of operation of the furnace, disregarding the time of its stoppage?

There are some professionals who think the time to be considered in the calculation of productivity is the calendar time (tc). In this condition, all the inefficiency and stops of the pelletizing unit is reflected by the productivity index.

Others suggest that the time is that programmed for the operation (tn) which must be in accordance with the nominal capacity or even scheduled for the operation of the plant.

Finally, there is a current that considers the effective time of operation of the furnace (te). In this concept, the productivity of the pelletizing furnace is measured, disregarding downtime for any reason.

To get an idea of how the value of productivity is influenced by the definition of the variable time, table 13.1 was constructed with data from an industrial plant type traveling grate and different definitions of time for the calculation of productivity (effective time of operation of the grate and calendar time). From the data, it is verified that the calendar productivity, of lower value than the effective one, indirectly expresses the inefficiencies of the pelletizing plant.

Table 13.1 – Influence of operating time on the productivity of the pelletizing furnace.

Operation data of the furnace Traveling Grate type	Productivity - t/m^2.day	
	Effective	Calendar
Grate width - 4,0 m		
Grate length - 192 m	Pr = 7,300,000/[768.(365-38.33)]	Pr = 7,300,000/(768.365)
Great reaction area - 768 m^2	or	or
Production in calendar year - 7,3 Mt	Pr = 29.10	Pr = 26.04
Total downtime in the year - 920 h or 38,33 days		
Calendar time - 365 days		

The best way to compare the performance of different pelletizing furnaces is through the calculation of the Calendar Productivity, because it is independent of definitions and secondary considerations of the time variable!

- *Type of pellet and volume of additives*

As previously discussed, there are two types of iron ore pellets for application in the steel industry: pellets for direct reduction and pellets for blast furnace. When this subject was discussed in chapter 5, the graph in figure 5.1 was drawn, showing the chemical particularities of each of these types. For ease of understanding the link with productivity in pelletizing furnaces, that figure was replicated below, with the designation of figure 13.2.

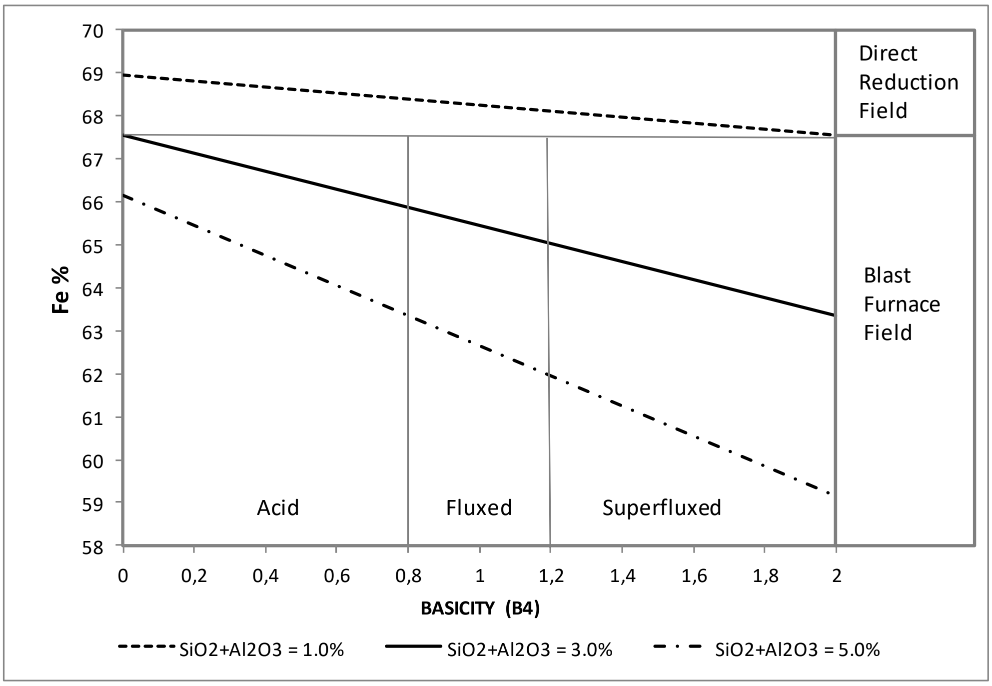


Figure 13.2 – Variation of iron content (Fe) with the basicity (B4) and with the percentage of acid slag (SiO2 + Al2O3) of the pellet. (1)

In Figure 13.2, it is verified that there are subtypes of pellets, characterized according to basicity, such as acid, fluxed and superfluxed pellets. In terms of iron contained, the pellets for direct reduction have it in contents greater than 67.5% and the blast furnace pellets lower than this value. It is evident that the lower the iron content of the pellet, the greater the volume of slag, largely caused by a higher dosage of fluxes. Generally, it can be said that the higher the basicity of the pellets and the volume of slag, the higher the flux dosages and, therefore, the lower the iron content. As inside the pelletizing furnaces the fluxes must be decomposed, with consequent loss of mass, the higher its dosage, the lower the productivity of the furnace (t/m2.day). Thus, the further one moves to the right in the graph of Figure 13.2, the lower the yields obtained in the pelletizing furnaces.

Using the mass balance, studied in chapter 6, a simulation could be made in order to quantify and evaluate the influence of binary basicity on the mass yield of pellets for different silica contents, % SiO_2.

This yield, as defined, is an *indirect measure of the productivity* achieved in the pelletizing furnace. For a better evaluation of the subject, it would be important to recall the definition of mass yield (Rm'), on a wet basis, seen in chapter 6, referring to the mass balance.

$$Rm' = 100. (P/P'),$$ where:

Rm' - mass yield of pellets in the pelletizing furnace (%);
P – net production rate of burnt pellets (t/h or t);
P' – feed rate of green pellets in the pelletizing furnace (t/h or t).

Therefore, consider the following assumptions for the simulation:

- Silica contents of pellets: 1; 3 and 5%;
- Binary basicity levels: 0.5; 1.0 and 1.5;
- Basic flux: limestone;
- Green pellet moisture: 9%;
- Anthracite dosage: 1.1% Cfix;
- Bentonite dosage: 0.5%;
- Generation of fines in the furnace: 2%;
- Al_2O_3 content of iron ore and limestone: 0.3%.

Taking these data to the mass balance, are obtained the results shown in table 13.2 and plotted graphically in Figure 13.3.

By analyzing these data, it is possible to indirectly verify the impact of the type of pellets and the volume of slag (chemical specifications) on the productivity of the pelletizing furnace (t/m2.day).

As an example, it is verified that the pellet with the highest mass yield (1% SiO_2 and basicity equal to 0.5) has a yield value of about 5% higher than that of the pellet poorer in iron, or with a higher volume of slag (5% SiO_2 and basicity of 1.5). This difference in yield, or productivity, is very relevant, significantly impacting the financial results of a pelletizing plant.

Table 13.2 – Mass yield in pelletizing as a function of silica content and basicity of burned pellet.

SiO2 content of pellet %	Mass Yield %		
	Pellet Basicity: CaO/SiO2		
	0,5	1,0	1,5
1,0	87,5	87,2	86,8
3,0	86,8	85,7	84,6
5,0	86,0	84,3	82,6

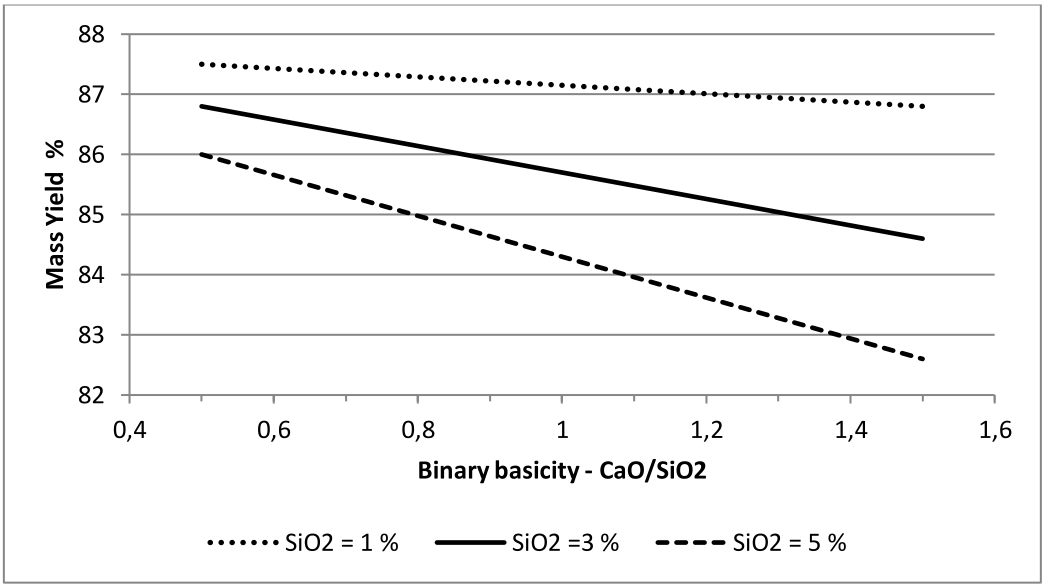


Figure 13.3 – Mass yield as a function of silica content and binary basicity of hematite pellets.

- *Type of iron ore*

The type of iron ore also has a great influence on the productivity of a pelletizing plant.

Magnetite ores (Fe_3O_4) are those with the highest productivity. During burning, the magnetite is oxidized to hematite and can have a mass increase of up to 3.5%, which directly impacts the productivity of the furnace. This oxidation is accompanied by a large release of heat inside the pellets, which accelerates the heating and reactions between the constituents of the pellet. This makes the firing cycle shorter, increasing furnace productivity (t/m2.day).

On the other hand, the hydrated ores, goethite and limonite types, (Fe2O3.nH2O), present a large amount of water in crystalline state. During the burning of the pellets, this crystalline structure is broken down to release water in the form of steam, at the expense of a large input of heat. This process delays the slagging and consolidation reactions of the pellets, making the firing cycle much longer and, therefore, the productivity lower.

In turn, hematite ores, with low hydration levels, present intermediate productivity to those of magnetite and hydrated ores.

The productivity of traveling grate furnaces varies greatly depending on these and other operating factors. Industrial plants, operating with these types of iron ore, with or without the addition of solid fuel to the raw pellet, in different locations, etc., have productivity indices in the following ranges:

- *Magnetite*: 27 to 40 t/m2.day,
- *Hematite*: 20 to 30 t/m2.day,
- *Hydrated*: 12 to 20 t/m2.day.

▪ *<u>Use of solid fuel in the pelletizing mixture</u>*

The second oil shock in 1979 triggered an accelerated rise in fuel prices and forced *<u>hematite pellet</u>* producers to rationalize and drastically reduce thermal energy consumption in a struggle for survival.

Many studies and research have been developed to optimize energy use, circumvent fuel shortage problems, and reduce the operating costs of pelletizing plants. In this scenario of difficulties, the past studies of adding solid fuel to the *hematite ore* in controlled quantities were rekindled, in order to preserve the quality of the pellets burned. Many fuels were considered, such as Coal, especially anthracite; Coke Breeze; Charcoal; Peat; Petroleum Coke - Petcoke; Pulverized Wood; etc. The idea was to create an *artificial magnetite*, during pellet burning in the furnace, by adding solid fuel to the green pellet in an amount equivalent to 1.3% fixed carbon (Cfix). After many studies, each plant has defined the optimal dosage for its operating conditions. It was found that this dosage was in the range of 1.0 to 1.2%. The upper limit of 1.3% Cfix in continuous and stable operation is still a challenge to be overcome.

Solid fuel proved to be a very important agent in the rationalization of thermal energy consumption, acting as a real substitute for petroleum fuels, commonly used in pelletizing plants (fuel oil and natural gas). Although the replacement was partial, for technical reasons, it represented an amount of the order of 25 to 30% of the total heat consumption in the pelletizing furnaces. With the evolution of the technique and the development of more suitable solid fuels (anthracite, for example), the substitution was growing and reaching levels around 45%.

The solid fuel provided the acceleration of the reactions inside the pellets in the furnace, reducing the firing cycle of the grate (total processing time) and, consequently, increasing the production rate and the productivity of the furnace. With the evolution and optimization of the use of this material, the owners of the technologies began to consider the performance data of the plants in the calculations of the new projects of pelletizing furnaces. The result was a generation of more efficient furnaces, with lower specific consumption and higher productivity. To get an idea of this evolution, consider the table 13.3 where the data of three furnaces designed by a holder of traveling grate technology, in different times and with increasing dosage of anthracite to green pellets, in the range of 0.0 to 1.1% Cfix. It can be seen, over time, a great growth in the productivity of the furnace, and, certainly, the anthracite had a large share of contribution in this increase.

Table 13.3 – Evolution of nominal productivity in projects of pelletizing furnaces type traveling grate, with increasing use of coal.

PLANT CHARACTERISTIC	PLANT # 1	PLANT # 2	PLANT # 3
Year of the project	1973	1995	2007
Anthracite dosage - % Cfix	0.0	0.9	1.1
Nominal capacity - t /year	3,000,000	4,000,000	7,000,000
Reaction area of the grate - m²	451	552	768
Operation time - # days	330	330	345
Nominal productivity - t/m².day	**20** (Reference)	**22** (+10%)	**26** (+30%)

Solid fuel addition studies were also conducted for magnetite ores, but without success. These ores naturally develop a large amount of thermal energy inside the pellets, via oxidation reaction, and there is no possibility of introducing more energy through solid fuel. As seen earlier in chapter 7, considerations on the use of thermal energy in pelletizing, this amount of energy generated by magnetite is of great magnitude, and there is no need for a complementary supply of heat, via solid fuel, for the burning of green pellets.

CHAPTER 14

OPERATIONAL ASPECTS OF THE GRATE OF A PELLETIZING FURNACE (1)

The movable grate of a conventional pelletizing furnace has a cross-section in the form of a trapezoid, with a small difference in sizes of the two bases, as indicated in figure 14.1 by B and b, respectively. When one observes an industrial unit, finds that this section is almost a rectangle.

In the case of the grate-kiln process, this grate is very simple, because it serves only to the conduction of the green pellets in the drying and pre-heating zones of the furnace, where the temperatures are lower. Thus, it is built with less sophisticated steels, has a small length and is similar to a continuous tractor track.

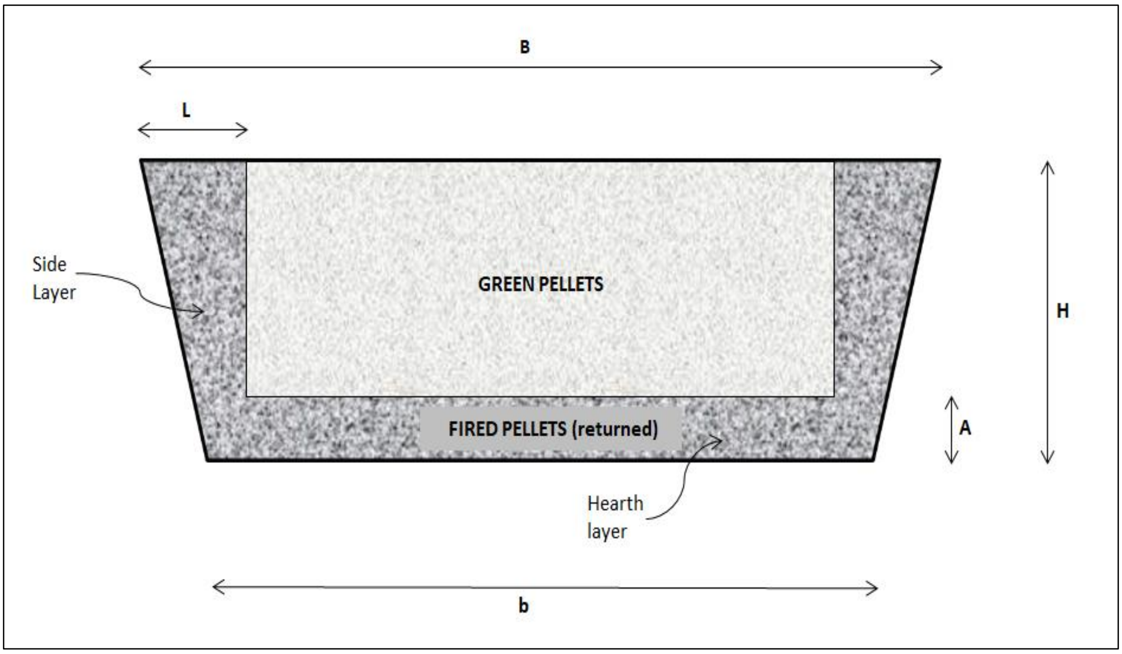


Figure 14.1 – Cross section of a grate car of the traveling grate process.

On the other hand, in the case of the traveling grate furnace, the grate undergoes the complete treatment of the pellets, exposing itself to higher temperatures and repetitive cycles. This grate is much larger in length, more robust in dimensions, the construction steels are special, and consists of a series of juxtaposed carts, which move, pressed by a counterweight. This conveyor grate is very similar to that found in iron ore sintering machines. Due to the higher temperatures, especially in the firing zone, the traveling grate requires side and bottom protection, which is done with burned pellets, recirculated or returned in the process (figure 14.1).

This recirculation material is a disadvantage of the traveling grate process because it consumes energy in the movement of the return material and also a portion of heat from the furnace. It also interferes negatively in the productivity of the grate, by occupying space that could be destined to green pellets. On the other hand, in addition to protecting the grate cars from high temperatures, it improves the permeability of the pellet bed to the passage of hot gases.

For the reasons already mentioned, the grate-kiln grate does not need this protection of lateral and bottom layers, and there is no recirculation of burned pellets in this process.

For the development of mathematical models, interconnecting the operational variables of a traveling grate, consider figure 14.1, where:

B – long base of the trapezoid of the cross section (m);

b – short base of the trapezoid of the cross section (m);

L – width of protection layer of side plate of grate (cm);

A – Height of the protective hearth layer (cm);

H – Total height of the layer of pellets in the grate (cm).

Thus, it can be written that the areas occupied by the green pellets (Sc) and burnt pellets (Sq) for grate protection (bottom and side layers) are given by the following equations.

a) <u>Area of the green pellets</u> (Sc)

$Sc = (B–2L/100). [(H-A)/100]$, in m^2 . Simplifying, it results:

$Sc = [(100B-2L)/100]. (H-A)/100$, or

$Sc = (100B-2L). (H-A)/10000$; in m2 . Eq. 14.1

b) <u>Area of the pellets for grate protection</u> (Sp)

The total area occupied by the bottom and side layers of the grate (Sp), consisting of pellets burned and recirculated in the traveling grate process, is equal to the difference between the total area of the trapezoidal section (St) of the grate car and the area occupied by the green pellets (Sc), as follows.

$Sp = St - Sc = \{[(B+b)/2]. H/100\} - (100B-2L). (H-A)/10000$; or

$Sp = (B+b). H/200 - (100B-2L). (H-A)/10000$, in m2. Eq. 14.2

Considering that the grate moves at speed **V** in m/min, it can be written that the volume flows of pellets from the protective layers of the grate (Qp) and of green pellets in the feed of the grate (Qc) are given by the following expressions.

c) Flow of green pellets in the feed of the grate (Qc)

This flow is given by: $Qc = Sc.V$

Replacing Sc with the value of the equation Eq. 14.1 and adjusting the dimension from V to m/h, yields:

$Qc = V. 60. (100B-2L). (H-A)/10000$; or

$Qc = V. (100B-2L). (H-A)/167$, in m3/h. Eq. 14.3

d) Flow of burnt protective pellets in the feed of the grate (Qp).

$Qp = Sp.V$ (flow rate referring to the recirculation of pellets for grate protection).

Replacing Sp with the value of the equation Eq. 14.2 and also, adjusting the dimension from V to m/h, yields:

$Qp = V.60. [(B+b). H/200 - (100B-2L). (H-A)/10000]$, or

$Qp = V.[(B+b). H/3.33 - (100B-2L). (H-A)/167]$, in m3/h. Eq. 14.4

e) Flow of green pellets in the feed of the Grate-Kiln (Qgk and Pgk)

In the case of the grate-kiln furnace, as there are no protective layers of the grate, the flow of green pellets (Qgk) can be defined as a function of the total area of the trapezoidal section. Thus, the following equation shall be taken for the calculation of the flow in volume (m3/h):

$Qgk = St.V$

Knowing the expression of St = (B+b). H/200, replacing it in the above equation and adjusting the units, yields:

Qgk = 60.V.(B+b).H/200, or

Qgk = V.H.(B+b)/3.33 in m^3/h.

Since dc is the density of the green pellets, one can calculate the mass flow of these pellets in the Grate-Kiln grate feed (Pgk in t/h) as:

Pgk = dc.Qgk, or

Pgk = dc. V.H.(B+b)/3.33, in t/h. Eq. 14.5

Returning to the Traveling Grate furnace, the burnt pellets for grate protection, which make up the bottom and side layers, pass through the oven without undergoing changes in weight. In turn, as previously seen in chapters 6 (mass balance) and 7 (productivity aspects), the green pellets are calcined and a significant loss of mass occurs, caused by the evaporation of moisture, decomposition of limestone, coal combustion, etc. This loss also depends on the type of pellets produced, the intensity of fine generation, the iron ore...

Knowing the following three parameters:

- ✓ bulk density of burnt pellets (dq);
- ✓ bulk density of green pellets (dc); and
- ✓ mass yield of green pellets, Rm', in %, (calculated by the mass balance in Chapter 6 and in wet basis),

the net hourly output of burnt pellets (P in t/h) derived from the burning of green pellets and the hourly amount of pellets recirculated in the bottom and side layers of the grate (Mr) in t/h can be calculated, both as functions of dimensions A and L (figure 14.1), defined in the operation of the grate. Thus, the calculations show the equations below.

f) Hourly amount of pellets recirculated in the bottom and side layers (Mr)

According to the above considerations, it can be written that:

Mr = Qp . Dq

Replacing the value of Qp with the expression of the equation Eq. 14.4, results:

Mr = V.[(B+b). H/3.33 – (100B-2L). (H-A)/167].dq , in t/h. Eq. 14.6

g) Net production of burned pellets (P)

Similarly, and within the concepts previously established:

$P = Qc . dc. Rm'/100$

Taking the Qc value of the equation Eq. 14.3, comes:

$P = V.dc.Rm'. (100B-2L). (H-A)/16700$, in t/h. Eq. 14.7

It is important to highlight that this is the net production obtained in the pelletizing grid, free of fines, after sieving, as conceptualized in chapter 6, referring to the mass balance. It must be understood that the total amount of pellets that is discharged by the grate is always a sum of: **P + Mr + pellet fines** (chips).

h) Productivity of the traveling grate (Pr)

The yield of the grate can also be calculated (Pr in t/m2.day) with the knowledge of the reaction area of the grate (Sg in m2). Within the concept of productivity, characterized in chapter 13, it can be written that:

$Pr = 24.P/Sg$, in t/m2.day (time adjusted from hour to day).

Considering the P-value of the equation Eq. 14.7, comes:

$Pr = 24. V.dc.Rm'. (100B-2L). (H-A)/(16700.Sg)$, or

$Pr = V.dc.Rm'. (100B-2L). (H-A)/(696.Sg)$, in t/m2.day. Eq. 14.8

Through these equations can be developed several simulations and evaluations of the production process, involving the variables considered in this study, especially those related to the recirculation pellets for the protective layers of the grate, as they relate to aspects of inefficiency of the pelletizing furnace, traveling grate type. The recirculation, in addition to increasing energy consumption, degrades the quality of the pellets, due to reburning.

Over the time, there has been a great evolution in the productivity of pelletizing furnaces and in the consumption of electrical and thermal energy. The projects evolved to considering longer and wider grates with also increase in total height of the pellet bed (figure 14.1). Another observed fact was the plant operators with hematite ores were able to minimize the recirculation of burned pellets, with the increase of coal addition to green pellets.

Due to the better uniformity of the burning, step by step, the dimensions of the bottom and side layers were being reduced. There were reports of plants operating without side layer, but there is always the risk of poor burning of the pellets at the interface with the side walls of the grate cars (wall effect, zero speed of the flow of hot gases). In any case, one should always seek the reduction of these layers of grate protection.

Within this context, a relationship (Rr) could be developed for the simulation of the amount of pellets recirculated (Mr) in relation to the net hourly production of the furnace (P), as follows.

i) Recirculation/pellet production ratio (Rr)

From the established concept, it can be written that:

$Rr = (Mr/P).100$, in %

Bringing the values of P and Pr from the equations Eq. 14.6 and Eq. 14.7, results:

$Rr = 100.\ \{V.[(B+b).\ H/3.33 - (100B-2L).\ (H-A)/167].dq\}/\ [V.dc.Rm'.(\ 100B-2L).\ (H-A)/16700]$

Simplifying, it results:

$Rr = [501502.(B+b).H - 10000.(100B-2L).(H-A)].dq/\ [dc.Rm'.(100B-2L).\ (H-A)]$ % Eq.14.9

Just as an example, consider figure 14.1, the equation Eq. 14.9, and the following data from a traveling grate industrial plant operating with hematite ore:

- ✓ long base of trapezoid of the grate cross-section: B = 3.68 m;
- ✓ short base of trapezoid of grate cross-section: b = 3.50 m;
- ✓ total height of the pellet bed in the grate: H = 40 cm;
- ✓ bulk density of burned pellets: dq = 2.0;
- ✓ Bulk density of green pellets: dc = 2,30;
- ✓ Mass yield of green pellets: Rm' = 84.8% (Data obtained through mass balance, referring to the production of a certain type of pellets for blast furnace).

Replacing these values specified above in the equation Eq. 14.9 and making the corresponding simplifications results in:

$Rr = [144031374 - 10000.(\ 368-2L).\ (40-A)]\ /[97.\ (368-2L).\ (40-A)]$, in %

In summary, we have the percentage relationship between the amounts of pellet recirculated and effectively produced in the travel grid, as a function of the height of the bottom layer (A) and the width of the lateral layer (L), or:

$$Rr = f(A,L) = [144031374–10000.(368-2L).(40-A)] / [97.(368-2L).(40-A)] \ (\%) \qquad \text{Eq. 14.10}$$

Through this equation, the graph of figure 14.2 was drawn, where a broader view of the influence of the dimensions of the bottom and side layers on the recirculation of burned pellets is obtained. As might be expected, the height of the hearth layer has a greater impact on recirculation than the width of the side layer. It is verified that for a hearth layer of 10 cm and side layer of 15 cm, in this characterized industrial furnace, the recirculation would be around 40%, that is, equivalent to 40% of the net production volume of the pelletizing furnace. Very high this recirculation! This point is insisted on, because the re-burning of the recirculated pellets degrades their physical and metallurgical quality. In this aspect, there is a disadvantage of the Traveling Grate technology in relation to the Grate-Kiln, because in the latter there are no layers of protection of the grid, that is, there is no recirculation or re-burning of pellets.

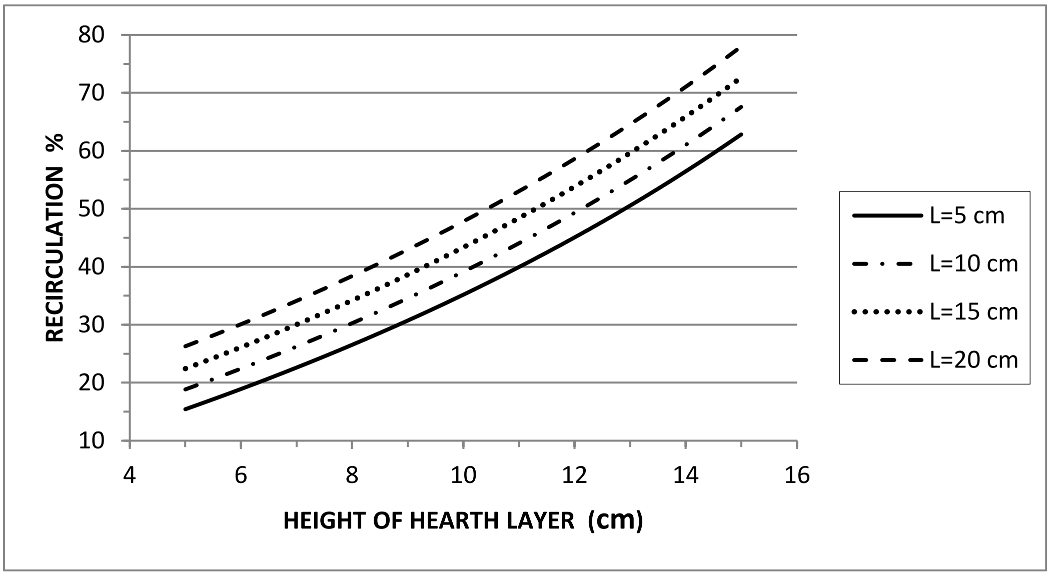


Figure 14.2 – Recirculation of burned pellets as a function of the height of the hearth layer (A), for different widths of the side layer (L).

j) <u>Aspects related to the residence time of the pellets in the kiln</u> (T)

Another important point in the operation of a pelletizing grate is the residence time of the pellets inside the furnace (T in min), also called the time of the thermal firing cycle. The control of this time is of fundamental importance in the operation because each type of pellet has a characteristic processing time in the furnace and should not suffer significant variations. It was demonstrated that both overfiring and underfiring of pellets cause quality degradation.

The cycle time is a variable that depends on the speed of the grate (V, in m/min) and the useful length of the furnace (C in m), and it can be written that:

T = C/V , in min.

From here, it comes:

V = C/T, in m/min. Eq. 14.11

From the equation Eq.14.7, it is seen that the net hourly output of the furnace is given by:

P = V.dc.Rm'. (100B-2L). (H-A)/16700, in t/h.

Substituting the V value of Eq. 14.11 in this equation, it results:

P = C.dc.Rm'. (100.B-2L). (H-A)/(T.16700), in t/h. Eq. 14.12

Considering the values of the variables of the reference industrial furnace (dc, Rm', B and H), previously defined, and knowing that:

- ✓ The length of this furnace is C= 129 m, and that
- ✓ The width of the protective side layer of the grate is L = 15 cm,

one can write the equation, Eq. 14.12, as follows.

P = (129). (2,3). (84,8). (368-30). (40a)/(T.16700) and simplifying, results:

P = 509. (40-A)/T, in t/h.

Thus, for this characterized industrial furnace, traveling grate type, with a nominal capacity of 3.0 Mt/year, this equation was established, involving 3 fundamental variables of operation of the traveling grate:

- ✓ Net Hourly Production of Pellets (P),
- ✓ Height of the Hearth Layer (A), and
- ✓ Residence Time in the furnace (T).

Thus, P = f(A,T) = 509. (40-A)/T, in t/h. Eq. 14.13

Taking the equations, Eq. 14.11 and Eq. 14.13, the residence time and grate speed of the considered industrial furnace can be calculated in nominal terms. It is important to emphasize again the data of such furnace:

- ✓ Nominal capacity: 3 Mt/year / 330 operation days;
- ✓ Grate length (C): 129 m;
- ✓ Nominal hourly production of burnt pellets, without fines (P): 379 t/h;
- ✓ Height of hearth layer (A): 10 cm.

From there, it comes:

Eq. 14.13: T = 510. (40-A)/P, or T = (510). (30)/379, then T = 40 min , and

Eq. 14.11: V = C/T, or V = 129/40, or V = 3.2 m/min

These, then, are the complementary nominal data of the industrial furnace studied, and for the type of blast furnace pellet considered in the mass balance, when the mass yield of the green, wet pellets was calculated, such as 84.8%.

On the other hand, taking the generic equations Eq. 14.11 and Eq. 14.13, the graphs shown in the following figures 14.3 and 14.4 were drawn, taking into account the data from the industrial furnace, taken as a reference in this development.

Figure 14.3 shows a direct relationship between the speed of the grate and the residence time of the pellets inside the reference industrial furnace, which has a useful length of 129 m.

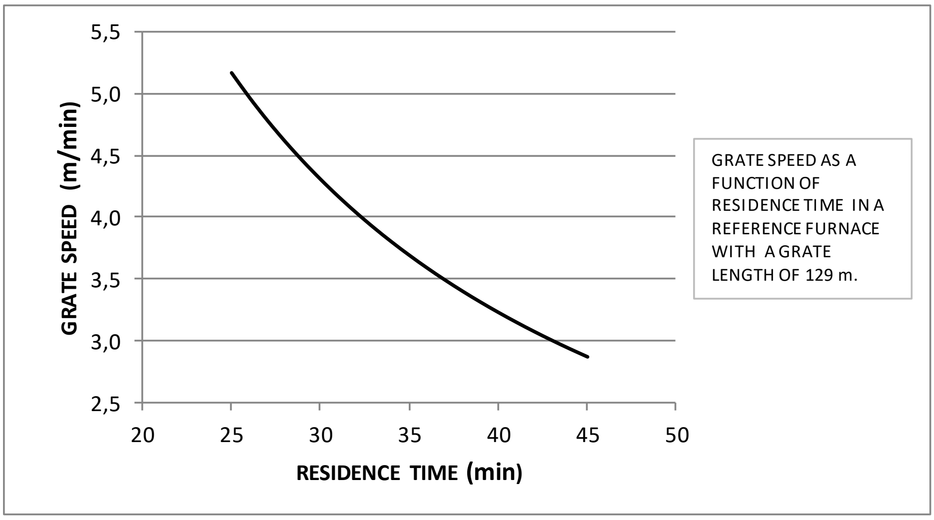


Figure 14.3 – Relationship between the speed of the grate and the residence time for the characterized industrial furnace, with a nominal capacity of 3.0 Mt/year.

In turn, figure 14.4, also drawn for the reference furnace, shows a relationship between three important operational variables:

- ✓ Net hourly production of burnt pellets, without fines, (P);
- ✓ Residence time of the pellets in the furnace (T) , and
- ✓ Height of the hearth layer (A).

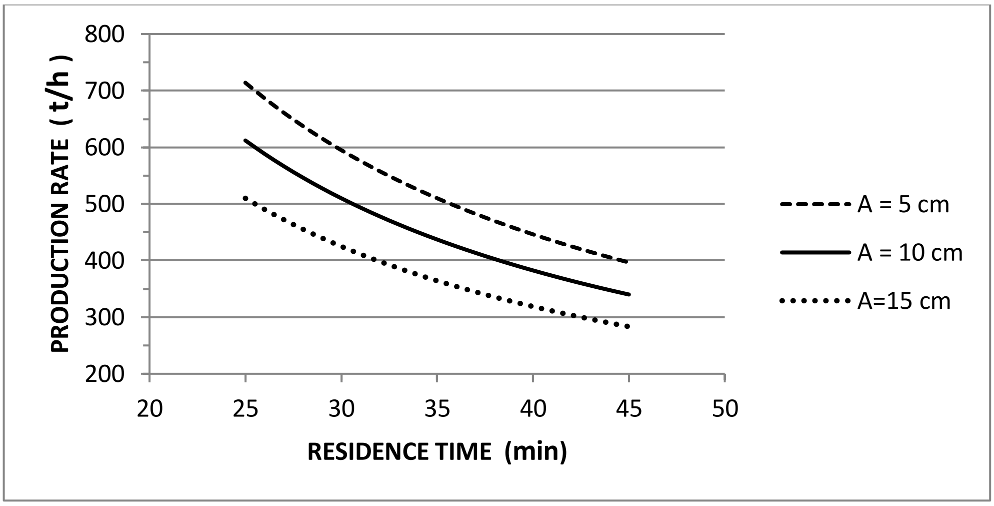


Figure 14.4 – Relationship between net hourly production and residence time in the reference industrial furnace, for different heights of lining layer (A).

In operational terms, considering a continuous and stable regime of operation of this furnace, with a defined hourly production rate, for example, equal to 400 t/h, what should be done, if there were, for any reason, variations of this production rate in the range of 350 to 450 t/h?

Should the residence time be maintained by changing the height of the hearth layer or the reverse procedure in which this layer would remain constant, causing a change in the residence time of the pellets in the furnace (modification of the grate speed)? Or should these two variables be adjusted simultaneously?

The answer depends on the magnitude of the variation around the standard production rate. For small variations, immediate action on the speed of the grid (small change in residence time) is recommended. For the large ones, it is necessary to act at the height of the hearth layer, even if in combination with the residence time.

The reduction of the total height of the pellet layer (H) to compensate a significant drop in production rate, is not recommended, because in addition to other problems of overheating, it exposes the sides of the grate cars to high temperatures, causing their accelerated degradation.

These are issues of domain of process control engineers, who are responsible for the development of conceptual studies and the definition of parameters and operational standards.

Analyzing the theme from the technological point of view, each type of pellet has a characteristic residence time (T) to obtain good quality. This time is defined, through exhaustive tests in the pilot plant, when the mass yield of the green pellets (Rm') and the bulk density of the same (dc) are also determined. Thus, when developing a traveling grate furnace project, this time and the optimal height of the hearth layer (A) accurately define the net hourly production of pellets (P). This calculation is possible, considering a great of established dimensions (C, B, L, H), in line with industrial practice and mechanical technology of heavy equipment. As seen earlier, the generic equation Eq. 14.12 may govern this calculation procedure.

Remembering the general equation of Eq. 14.12, comes:

$$P = C.dc.Rm'. (100.B-2L). (H-A)/(T.16700) \text{ in t/h.}$$

In this context of process and project development, the variables are:

P – net production rate, obtained in the traveling grate furnace (t/h);

A – height of the hearth layer of the grate (cm);

T – residence time of the pellets in the kiln (min).

And the known data from pilot plant testing, industrial plant operation and mechanical engineering would be:

dc – bulk density of green pellets,

Rm' – mass yield of green pellets in a wet basis (%),

C – useful length of the grate (m),

B – long trapezoid base of the grate cross-section (m),

L – width of the protective side layer of the grate (cm) and

H – total height of the layer of pellets in the grate (cm).

From there, the calculations and simulations to the project could be made.

In conclusion, it is said that*:*

"The grate is the soul of a conventional pelletizing furnace!"

It dictates the pace and quality of the production process, either in Grate-Kiln or Traveling Grate technology.

In this chapter, it was tried to interrelate some of the main variables of operation of the traveling grate, in order to enable general simulations, inherent to the development of process and engineering of iron ore pelletizing.

CHAPTER 15

ASPECTS RELATED TO WET OPERATIONS WITH IRON ORE (1)

The preparation of iron ore for pelletizing, as seen in Chapter 2, involves several unit processing operations. When this step is carried out in a wet way, there is the formation of water-ore pulp or slurry with varying densities (% solids), depending on the processing that is considered (grinding, thickening, filtering, etc.).

Engineers and process technicians, responsible for these operations, have to deal with many calculations, involving iron ores, additives and water, materials that have very different densities.

There are several important relationships involving the pulps of minerals and ores in water, which are discussed in the following topics.

(a) *% of solids in ore- water slurry (compact particles)*

Having a unit operation in wet route, involving ore-water slurry, it is very important to determine the % of solids (S), as a function of the densities of the ore and pulp.

Thus, take a pulp of water with any solid material, pulverized, as shown in figure 15.1 below. Consider, also, that the solid particles are compact, with zero porosity, with no absorption of water portion of the pulp. Consider that:

Vs – total volume occupied by solid particles;

Ms – total mass of solid particles;

ds = dr, true or real density of solid particles;

Vag– total volume occupied by water;

Mag – total mass of water;

dag = 1, density of water;

Vp – total volume of the aqueous pulp with solids;

Mp – total mass of the pulp;

dp – density of the pulp.

The percent by mass of solids in the pulp (SM) is defined as follows:

$$SM = 100.Ms/Mp \ \%$$

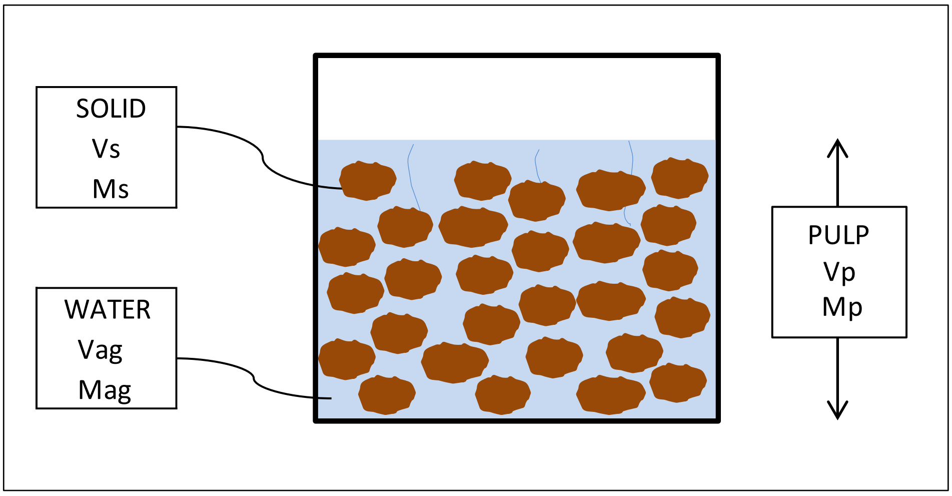


Figure 15.1 – Generic pulp of compact solid in water.

Thus, the following relationships can be written.

$$Mag + Ms = Mp, \text{ e} \qquad \text{Eq. 15.1}$$

$$Vag + Vs = Vp \qquad \text{Eq. 15.2}$$

From the concept of densities, one can transform the equation Eq. 15.2 in:

$Mag/dag + Ms/dr = Mp/dp$, or

$Mag + Ms/dr = Mp/dp$ Eq. 15.3

From equation 15.1, it comes:

$Mag = Mp – Ms$. Taking this value of Mag in Eq. 15.3, results:

$(Mp – Ms) + Ms/dr = Mp/dp$

Developing this last expression and simplifying, it comes:

$Ms/dr – Ms = Mp/dp – Mp$, or $Ms.(1/dr – 1) = Mp.(1/dp – 1)$, or

$Ms.(1 – dr)/dr = Mp.(1 – dp)/dp$, or $Ms/Mp = [(1 – dp).dr]/[(I – dr).dp]$

Hence, expressing this mass fraction in %, yields:

$SM = f(dp, dr) = 100.Ms/Mp = 100. [(1 - dp). dr]/[(1 - dr).dp]$

Then, the % *mass of solids in the pulp* is given by:

$$\mathbf{SM = f(dp, dr) = 100. (dp - 1).dr/[(dr - 1).dp] \ \%} \qquad \text{Eq. 15.4}$$

In many pelletizing plants, iron ore, fluxes and even anthracite are ground separately, in wet basis. Thus, different pulps of these materials are produced, with very different characteristics. Then, fluxes and anthracite are dosed to the ore, before filtering, through controlled dosages, of these materials, via pulp-by-pulp. As ore, fluxes and anthracite have different densities (dr), it is necessary to have a precise system of measurement and control of flows and densities, in order to have a mixture with low variability and within the specification.

To get a sense of how the density of the solid affects the density of the pulp and the corresponding % of solids, consider iron ore, limestone and anthracite, with the following values of true density (dr), 5.0, 2.1, and 1.5, respectively. Taking these values into the generic equation of SM, Eq. 15.4, the following three equations result, which govern the % by mass of solids of these materials, when in pulp with water:

For iron ore: $SMm = 125. (dp - 1)/dp$ %; Eq. 15.5

For limestone: $SMc = 191. (dp - 1)/dp$ %; Eq. 15.6

For anthracite coal: $SMa = 300. (dp - 1)/dp$ %. Eq. 15.7

Taking these last three equations, the curves SM = f(dp) can be plotted for iron ore, limestone, and anthracite, as shown in Figure 15.2.

In this figure, it is seen how the true density of the materials impacts the value of the % by mass of solids in the pulp. The lower this density, the greater the impact, as in the case of anthracite. Hence, when working in the industry with pulps of these minerals and ores of low density, for a small variation of this parameter in the pulp, there is a large variation in the percentage of solids of the pulp. As mentioned earlier, mixtures for pelletizing must be prepared with iron ore and flux dosages and, where appropriate, solid fuel, in rigid proportions as defined by the mass balance in Chapter 6. Greater care is needed in the additions to the ore of low-density materials, in pulp-to-pulp operations.

Logically, calculated the percentage by mass of solids, SM, the *mass percentage of water in the pulp (AM)* would be defined as the complement of 100%. That is:

$$\mathbf{AM = (100 - SM)\ \%} \qquad \text{Eq. 15.8}$$

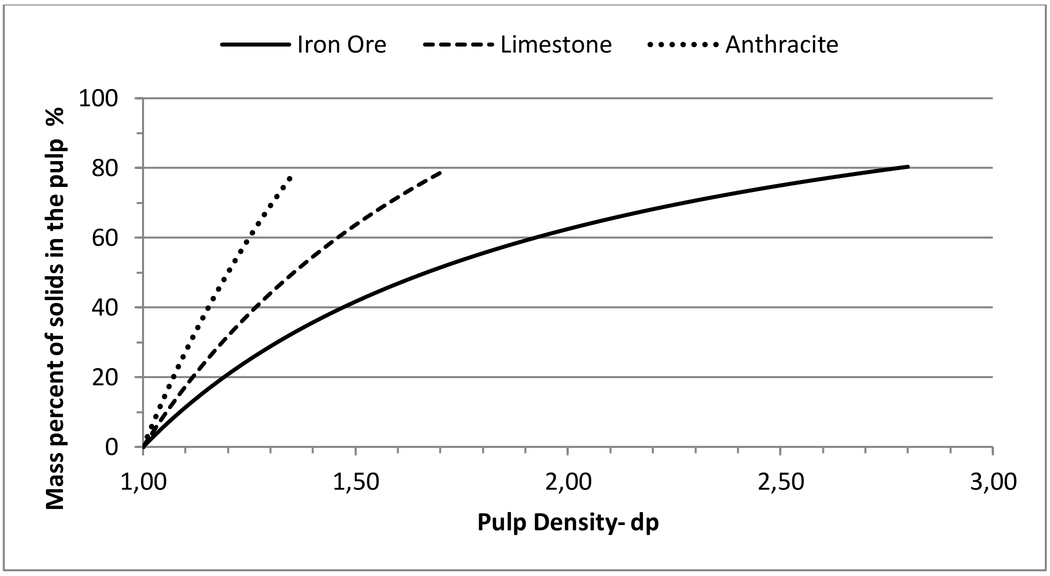


Figure 15.2 – Relationship between the % by mass of solids and the density of the pulp for iron ore, limestone, and anthracite.

The solids of the pulps are made up of *mineral particles*, which disperse within these pulps. When the particles are compact, with zero porosity, as in the present case, the volume of the *solid part of ore* is equal to the *volume occupied by the mineral particles*. On the other hand, when the ore is porous, which means that the pulp particles are also porous, the volume *of the solid part of the ore* is less than the *volume occupied by the mineral particles*. This is important when analyzing the *percentage by volume of* solids, which impacts the rheology of the pulp (fluidity, viscosity, etc.). It is important to always remember that the numbers that express *% in mass and % in volume* are very different.

The definition of *% of solids by volume of a pulp* (SV) is given by:

SV % = 100.Vs/Vp.

Since Vpm is the apparent volume of mineral particles (equal to the true volume of ore), it can be written that:

Vpm = Vs + Vpo, where Vpo is the volume of pores inside the particles.

As in the present case under study the *ore is compact, with zero porosity, $Vpo = 0$,* or:

$Vpm = Vs$

Taking this value of Vs in SV, it follows that:

$SV\ \% = 100.Vpm/Vp$, where: Eq. 15.9

SV % - % of solids by volume in the pulp;

Vpm – volume occupied by mineral particles (equal to the volume of solids), and

Vp – pulp volume.

On the other hand, considering the concept of percentage of solids by mass and the expression of the equation Eq. 15.4, it comes:

$SM = 100.Ms/Mp = 100.\ (dp - 1).dr/[(dr - 1).dp]\ \%$ Eq. 15.10

By the concepts of true density and pulp density, it is written that:

$dr = Ms/Vs$ and $dp = Mp/Vp$, or

$Ms = Vs.dr = Vpm.dr$ and $Mp = Vp.dp$

Taking Ms and Mp to Eq. 15.10, results:

$100.Vpm/Vp = 100.\ (dp-1)/(dr-1)$

According to equation Eq. 15.9, this expression represents the *% of compact solids by volume in the pulp*, or:

$SV = f(dr,dp) = 100.\ (dp - 1)/(dr - 1)\ \%$ Eq. 15.11

In the same way as was done for the % of solids by mass, and to verify how the true density of the solid affects the density of the pulp and the % of solids by volume, iron ore, limestone and anthracite are considered, with the following values of true density (dr), respectively, 5.0, 2.1, and 1.5. Taking these values into the generic equation of SV, Eq. 15.11, the following three equations result, which govern the *% of compact solids by volume of* these materials, when in pulp with water:

For iron ore: SVm = 25. (dp – 1) %, Eq. 15.12

For limestone: SVc = 91. (dp – 1) %, Eq. 15.13

For anthracite coal: SVa = 200. (SD – 1) %. Eq. 15.14

Taking these last three equations, the curves SV = f(dp) can be plotted for ore, limestone, and anthracite, as shown in Figure 15.3.

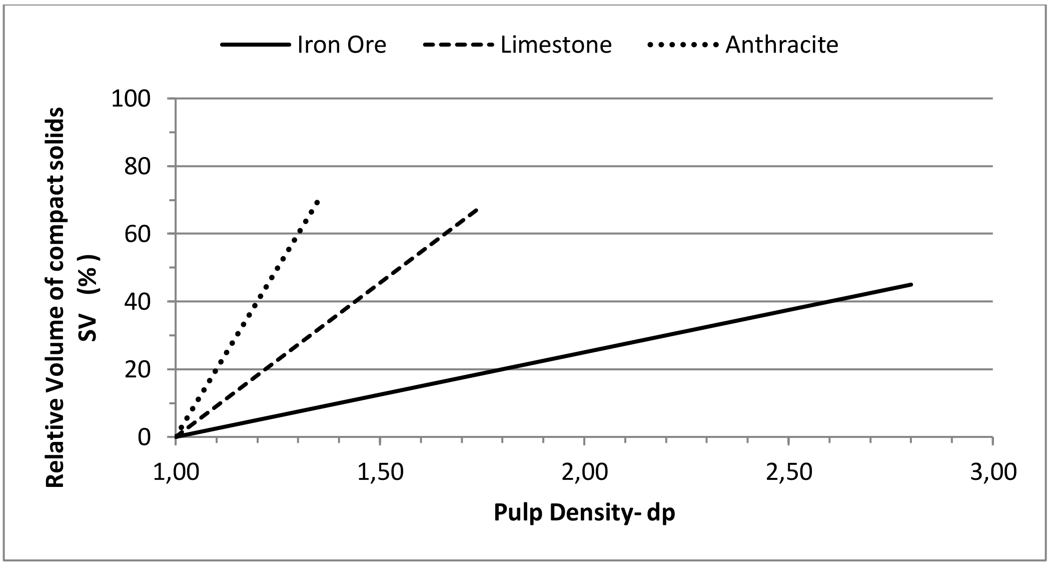


Figure 15.3 – Relationship between the % by volume of compact solids and the density in the pulp for iron ore, limestone, and anthracite.

In this figure, it is seen how the true density of the materials impacts the value of the *% by volume of compact solids of the pulp*. The lower this density, the greater the impact, as in the case of anthracite.

Logically, calculated the percentage of solids, SV, the percentage in volume of water of the pulp (AV) would be defined as the complement of 100%. That is:

$$\mathbf{AV = (100 - SV)\ \%} \qquad \text{Eq. 15.15}$$

Finally, considering the compact mineral particles, with zero porosity, and the equations of SM and SV, Eq. 15.4 and Eq.15.11, it is possible to develop the relationship between SM and SV, as follows.

From Eq. 15.4:

$SM = 100.\ (dp - 1).dr/[(dr - 1).dp]$,

and of Eq. 15.11:

$SV = 100.\ (dp - 1)/(dr - 1)$. From this, substituting dp and $(dp - 1)$, it comes:

$dp = [100 + SV.(\ dr - 1)]/100$, and

$(dp - 1) = SV.\ (dr - 1)/100$.

Substituting dp and (dp - 1) in the above SM equation yields:

$SM = 100.SV.\ (dr - 1).dr\ /\ \{(dr-1).\ [100 + SV.\ (dr - 1)]\}$, or

$$\mathbf{SM = f(SV,dr) = 100.SV.\ (dr - 1).dr\ /\ \{(dr-1).\ [100 + SV.\ (dr - 1)]\}} \qquad \text{Eq. 15.16}$$

Taking this equation, the curve can be plotted relating SM (% by mass of solids in the pulp) x SV (% by volume of solids in the pulp), for the ore, limestone, and anthracite, compact, which have true densities (dr), respectively, equal to 5.0, 2.1 and 1.5. Substituting these values into the equation Eq. 15:16, it comes as follows:

For iron ore: $SMm = 125.SV\ /\ (25 + SV)$; Eq. 15.17

For limestone: $SMc = 193.SV\ /\ (\ 92 + SV)$, Eq. 15.18

For coal: $SMa = 300.SV\ /\ (200 + SV)$. Eq. 15.19

In the graph of the figure 15.4, below, the representations of these three equations can be seen.

It is easily verified that the higher the density of the material, the greater the difference between the percentages of solids, in mass (SM) and in volume (SV), of the aqueous pulp.

It is important to emphasize again that in the present case, in which the ore considered is compact, with zero porosity, *the volume of the mineral particles of the pulp* is the same as *the volume of the solid part of the ore*. Thus, the % by volume of solid material *is equal to* % by volume of particles *in the ore pulp.*

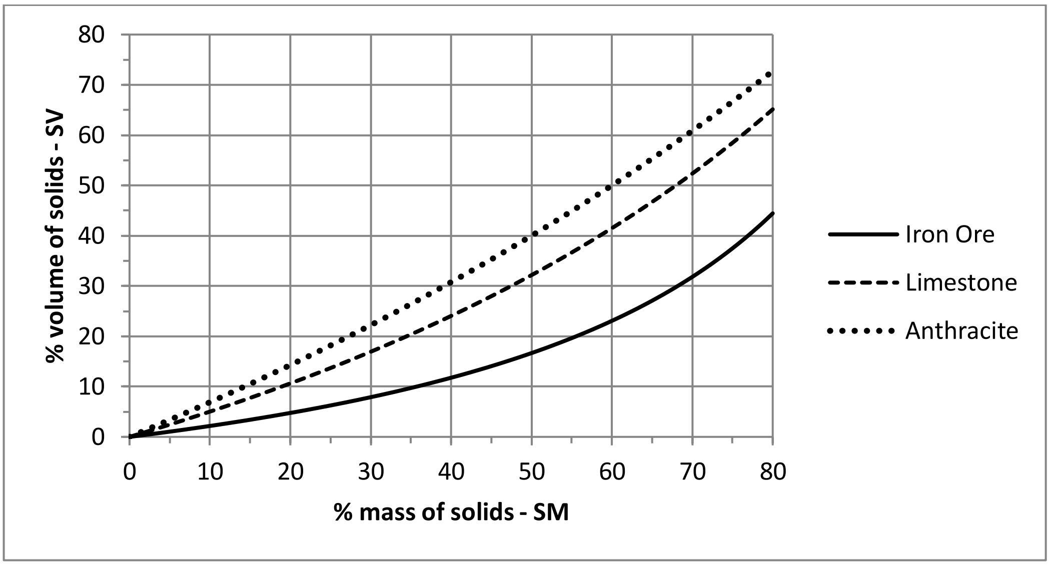


Figure 15.4 – Relationship between % by volume and % by mass of solids in the pulp, for iron ore, limestone, and anthracite.

b*) <u>% of solids in ore- water pulp (porous particles)</u>*

In the previous case, figure 15.1, the mineral particles were considered as compact, in which the individual porosity was zero. But in nature, these particles always have pores. There are iron ores with different levels of porosity, which depends on their genesis. Thus, when considering the natural ores in pulp with water, some portion of this water is absorbed by the mineral particles, changing the volumetric parameters, characteristics of this pulp. Figure 15.5 shows a generic representation of porous ore particles dipped in water, constituting a pulp.

Immediately, visually, it is verified that *<u>the volume occupied by the porous mineral particles</u>* is greater than *<u>the volume occupied by true mass of ore itself</u>*, the difference being equal to the volume of pores of the particles. This is important because the rheology of the pulp is more impacted by the volume of the particles than by the true volume of the ore.

As it is known, pores can be opened or closed. Only the open ones, those that communicate with the external environment of the particle, absorb some of the water from the pulp. The pores, usually, are open in their vast majority.

For the present development, the closed pores are disregarded, and it is assumed that *<u>the pores of the particles are all open and integrally filled with water</u>*.

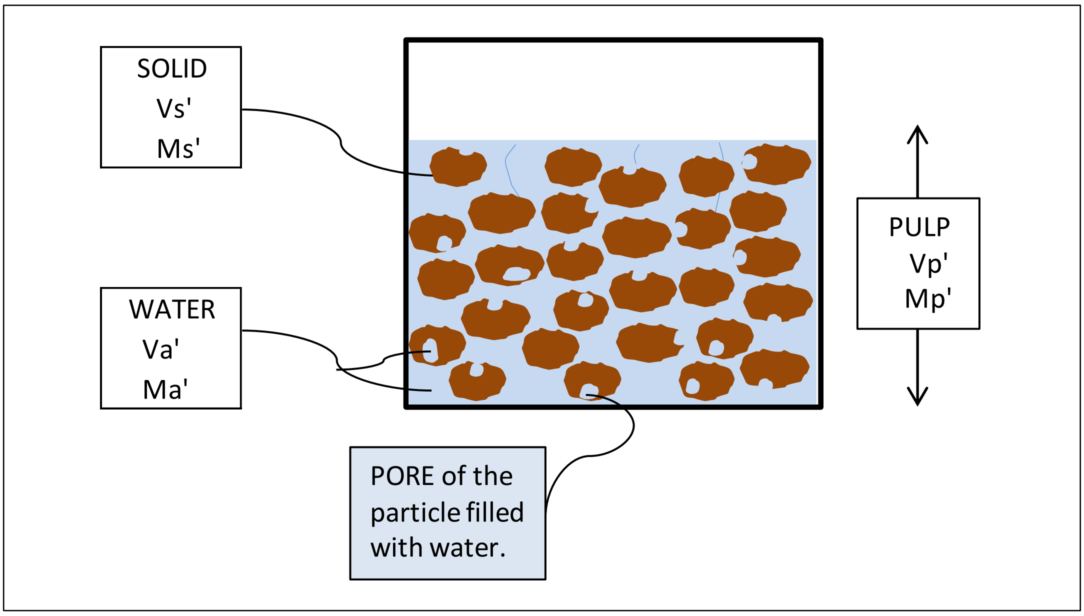


Figure 15.5 – Generic pulp of porous solid in water.

Similarly to the case of the pulp with compact particles, it can be written that:

Ma' + Ms' = Mp', and Eq. 15.20

Va' + Vs' = Vp' Eq.15.21

Where:

Ma' – total water mass of the pulp (includes mass of water in the pores);

Ms' – solid mass of the ore particles;

Mp' – Mass of ore-water pulp;

Va' – total volume of water (includes volume of water in the pores),

Vs' – volume of the solid mass of the mineral particles;

Vp' – volume of pulp;

Vpo' – volume of pores,

Vpa' – total volume of porous particles (volume of mineral mass + volume of pores).

Whereas:

dr' = true density of the ore,

da' – bulk density of porous mineral particles,

dag' – density of pulp water = 1

dp' – density of pulp,

One can do the development that follows.

Taking the equations Eq. 15.20 and Eq. 15.21 and making the necessary transformations, the following relations are obtained:

$Ma' + Vs'.dr' = Vp'.dp'$ Eq. 15.22

$Ma' + Vs' = Vp'$ Eq. 15.23

Subtracting, member by member, the equation Eq. 15.22 of Eq. 15.23, it comes:

$Vs' - Vs'.dr' = Vp' - Vp'.dp'$, or

$Vs'.(1 - dr') = Vp'.(1 - dp')$ Eq.15.24

Analyzing the figure 15.5, it is seen that the volume of the *particles is equal to the true volume of the mineral mass, added to the volume of pores*. Thus, and in view of the definitions of previous variables, it is valid to write that:

$Vpa' = Vs' + Vpo'$

Hence, $Vs' = Vpa' - Vpo'$ Eq.15.25

From the definition of porosity of mineral particles (Po'), it comes:

$Po'\ \% = 100.Vpo'/Vpa'$

Hence, $Vpo' = Po'.\ Vpa'/100$.

Taking Vpo' in the expression of Eq. 15.25, and simplifying it results:

$Vs' = Vpa'.\ (1 - Po'/100)$

Replacing Vs' in Eq. 15.24, it results:

$Vpa'.\ (1 - Po'/100).\ (1 - dr') = Vp'.\ (1 - dp')$, or

$Vpa'/Vp' = (1 - dp') / [(1 - Po'/100).(1 - dr')]$

This expression translates the *fractional volume, occupied by the porous particles in the* pulp. Expressed in percentage terms, the percentage *in volume of porous particles (SP') in the* aqueous pulp is obtained. That is:

$$\mathbf{SV' = 100. (dp' - 1) / [(1 - Po'/100).(dr' - 1)]\ \%.} \qquad \text{Eq. 15.26}$$

Where:

SV' – _percentage by volume_ occupied by the _porous particles in the_ pulp;

dp' – density of the pulp;

Po' – porosity of mineral particles in %;

dr' – true density of the mineral (solid).

Taking the expression of the equation Eq. 15.26, characteristic curves of a given mineral, can be plotted for different porosity levels, as shown in figure 15.6. Developing this equation for _iron ore_, (dr =5), and porosity levels of the particles equal to 0%, 15% and 30%, result the following expressions:

For porosity = 0%: $SV'_0 = 25.(dp' - 1)\ \%$;

For porosity = 15%: $SV'_{15} = 30. (dp' - 1)\ \%$;

For porosity = 30%: $SV'_{30} = 36. (dp' - 1)\ \%$.

It is worth mentioning that when the *porosity* of the particles is equal to *zero*, this means that the particles are compact and the expression that *governs* the percentage in volume is the same as that of the equation Eq. 15.12.

The variations of SV', shown in the graph of figure 15.6, influence the rheological parameters of the iron ore pulp and, therefore, its behavior in handling and transport operations.

For each ore or material, these characteristic curves can be plotted.

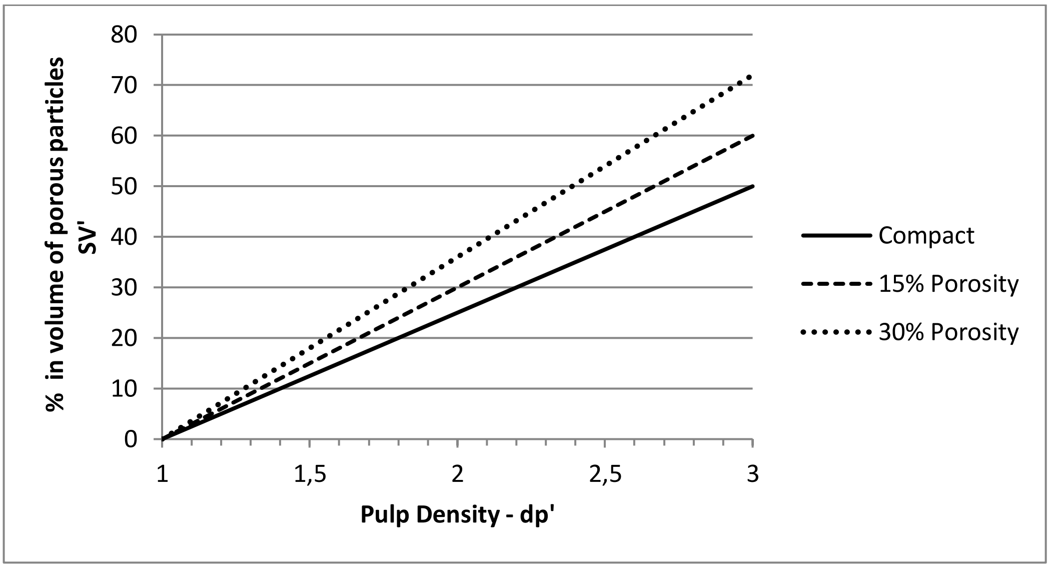


Figure 15.6 – Relationship between the relative volume of porous particles and the density of the pulp, for different levels of particle porosity.

It is important to note that *the percentage by mass of solids (SM') is independent of porosity,* being given by the same expression of compact particles (zero porosity. Analogous to Eq. 15.4, it is written that:

$$\mathbf{SM' = 100.\ (dp' - 1).dr' / [(dr' - 1).dp']\ \%} \qquad \text{Eq. 15.27}$$

In fact, the percentage by mass of solids is independent of the porosity of the mineral particles. This can be proven as follows. From the equation Eq. 15.26, concerning the percentage of solids by volume (SV'), it may be written that:

$SV' = 100.Vpa'/Vp' = 100.\ (dp' - 1) / [(1 – Po'/100).(\ dr' - 1)]$

But, from the concept of densities:

$Vpa' = Ms'/da'$ and $Vṕ = Mp'/dp'$.

Substituting in SV' expression above, it comes:

$ms'.dp'/mp'.da' = 100.\ (dp' - 1) / [(1 – Po'/100).(\ dr' - 1)]$

Or the same as:

$SM' = 100.Ms'/Mp' = 100.da'.\ (dp' - 1) / \{dp'.[\ (1 – Po'/100).\ (dr' - 1)]\}$

Knowing that $Po' = 100.\ (1 – da'/dr')$, results:

$(1 – Po'/100) = da'/dr'$.

Substituting in SM', it comes:

$SM' = 100. (dp' - 1) / [(dp'/dr').(dr' - 1)]$

Or, finally, as it was intended to demonstrate:

$$\mathbf{SM' = 100. (dp' - 1).dr' / [(dr' - 1).dp'] \ \%} \qquad \text{Eq. 15.28}$$

c) *Dosage of solids – pulp to pulp basis*

In many pelletizing plants, iron ore is wet ground and handled in the form of aqueous pulp. The best way to prepare the additives is through dry grinding, separately, dosing then the ore, before the mixers, through precision scales. However, for various reasons, there are industrial units in which some of these additives are wet ground and dosed to the ore, in the form of pulp in water. A typical case is that of anthracite, dosed in this way to the pulp of ore, as shown in Figure 15.7.

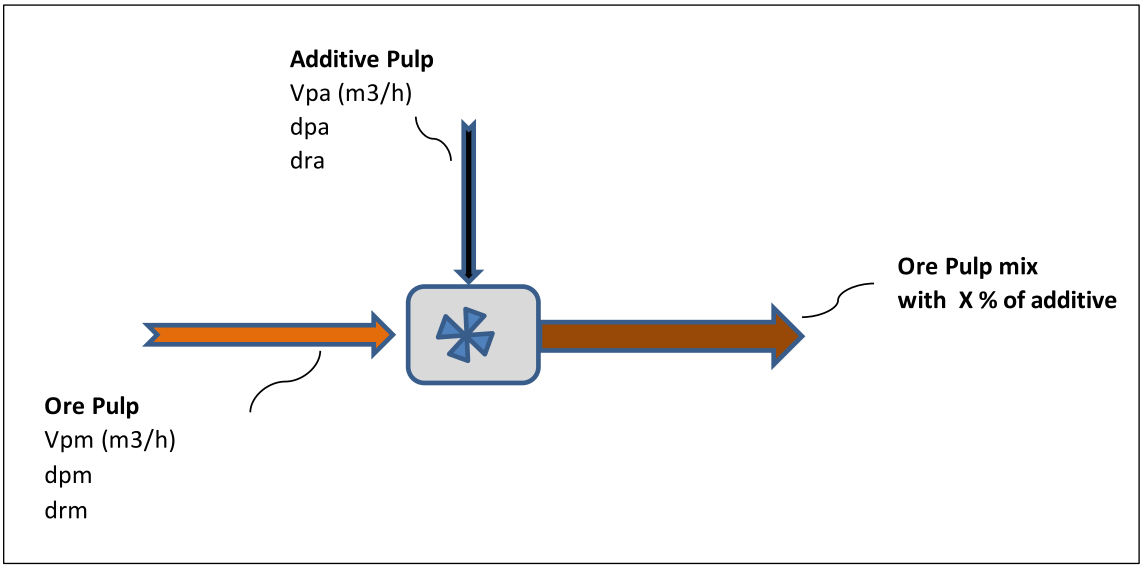


Figure 15.7 – Dosage of additive to ground iron ore via pulp-to-pulp.

In this case, it is necessary to have an accurate control of this processing, since an excess dosage of anthracite can cause serious operational and quality problems to the pellets. Thus, it is required that the dosing system has a quality and precise instrumentation.

For the development of the dosage model, two aqueous pulps are considered, one of iron ore and the other of any additive, to be dosed to the ore, in a previously defined percentage (Figure 15.7), with the following characteristics:

Iron ore pulp

- dpm – density of ore pulp;
- drm – true density of ore;
- Sm - % of solids in ore pulp;
- Msm – solid ore mass flow (t/h);
- Mpm – mass flow of ore pulp (t/h);
- Vpm – flow in volume of the ore pulp (m3/h).

Additive pulp to be dosed

- dpa – density of the additive pulp;
- DRA – actual density of the solid additive;
- Sa - % of solids of the additive pulp;
- Msa – solid additive mass flow (t/h);
- Mpa – mass flow of additive pulp (t/h);
- Vpa – flow in volume of the additive pulp (m3/h).

On the other hand, it is established that the additive must be dosed to the ore, in a mass percentage basis, equal to " X % ", that is:

$X = 100.Msa/Msm$ Eq. 15.29

From the previous studies (Eq. 15.4), it can be written that the percentages of solids of the aqueous pulps of ore and additive, characterized above, would be given by the following equations:

Ore pulp: $Sm = 100.(dpm - 1).drm/[(drm - 1).dpm]$ % , and

Additive pulp: $Sa = 100.(dpa - 1).dra/[(dra - 1).dpa]$ %

These equations could be unfolded as follows:

Ore Pulp:

$Msm/Mpm = (dpm - 1).drm/[(drm - 1).dpm]$

On the other hand, $Mpm = Vpm. dpm$

Substituting Mpm into the above equation of Sm and simplifying it yields:

$$Msm = Vpm.(dpm - 1).drm/(drm - 1) \qquad \text{Eq. 15.30}$$

Analogously, one can write a similar equation for the additive pulp.

Additive pulp:

$$Msa = Vpa. (dpa - 1).dra/(dra - 1) \qquad \text{q. 15.31}$$

From the established assumptions, it is known that the percentage in mass of dosage of the additive to the ore is given, according to the equation Eq. 15.29, by:

$X = 100.Msa/Msm$.

Dividing, member by member, the equations Eq. 15.31 and Eq. 15.30, one obtains the expression of Msa/Msm that can be taken in X, resulting:

$X = \{[100.Vpa.(dpa - 1).dra]/(dra - 1)\} / [Vpm.(dpm - 1).drm/(drm - 1)]$.

Simplifying, it comes:

$X = 100.Vpa.dra. (dpa - 1). (drm - 1) / [Vpm.drm.(dra - 1).(dpm - 1)]$

From this last equation, one can extract the function *" Vpa = f(Vpm, X, densities) "*, which translates the *dosage model of additives to iron ore*, via pulp-to-pulp:

$$\mathbf{Vpa = Vpm.X.drm.(dra - 1). (dpm - 1) / [100.dra.(drm - 1).(dpa - 1)]} \qquad \text{Eq. 15.32}$$

Where:

- Vpa – Flow in volume of the additive pulp (m3/h);
- Vpm – Flow in volume of the ore pulp (m3/h);
- X – percentage by mass of the additive to be dosed to the ore (%);
- drm – truedensity of ore;
- dra – true density of the solid additive;
- dpm – density of ore pulp;
- dpa – density of the additive pulp.

Once the mass dosage of the additive to the ore (X %) and the densities of the equation Eq. 15.32 are defined, it is possible to calculate the flow of the additive pulp (Vpa, m3/h), as a function of the flow of the ore pulp (Vpm, m3/h).

As an example, and based on figure 15.7, the following data are considered, referring to the dosage of ground anthracite coal to iron ore, also already ground, via pulp-to-pulp addition:

- X = 1.5% – percentage by mass of the anthracite additive to be dosed to the ore;
- drm = 5.0 – true density of iron ore;
- dra = 1.5 – true density of the additive, in this case anthracite;
- dpm = 2.3 – density of the iron ore pulp.

Keeping the anthracite pulp density (dpa) as a variable, and taking these data to the equation Eq. 15.32, the general equation of the dosage model for this specific case is obtained:

$$\mathbf{Vpa = f(Vpm, dpa) = Vpm / [123.(dpa - 1)]} \qquad \text{Eq. 15.33}$$

Taking this case and varying the density of the aqueous anthracite pulp (dpa), the following dosage equations can be obtained:

- For dpa = 1.05: Vpa = 0.163.Vpm; Eq. 15.34
- For dpa = 1.10: Vpa = 0.081.Vpm; Eq. 15.35
- For dpa = 1.15: Vpa = 0.054.Vpm. Eq. 15.36

In this way, it is possible to draw curves for dosage of 1.5% of anthracite to iron ore, according to the specifications stipulated and governed by the equations. Eq 15.34 to Eq. 15.36, as shown in Figure 15.8.

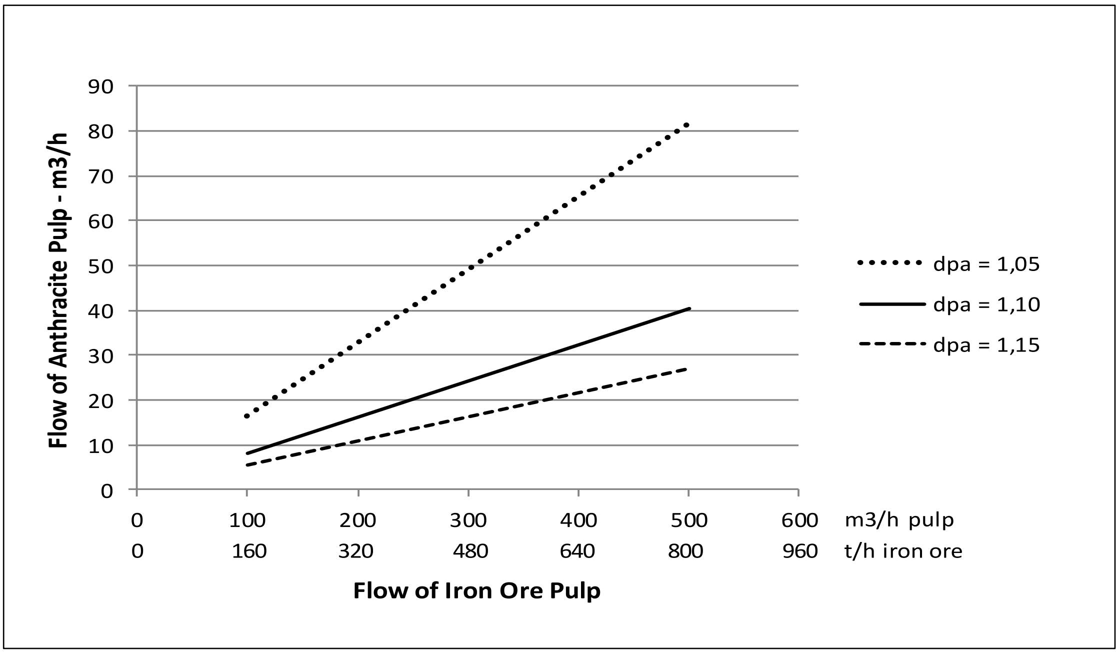


Figure 15.8 – Anthracite pulp flow for dosage of 1.5% by mass to iron ore in aqueous pulp, for different densities of anthracite pulp.

Through this figure, it is verified that the flow of the aqueous anthracite pulp strongly depends on the density of this pulp. Thicker pulps of this material, or of higher densities, allow operation at lower flows, which means less interference in the density of the pulp resulting from the mixture of iron ore pulp + anthracite pulp. In this case, the anthracite pulp should have the highest possible density, compatible with the viscosity and fluidity specifications of this material, in aqueous pulp. Industrial practice has shown that the limit density of anthracite pulps is around 1.20, which means a mass percentage of solids around 50%.

(d) *Wet grinding and sizing* (1)

In many pelletizing plants, the preparation of iron ore includes wet grinding, in a closed circuit with a classifier, which usually consists of a hydrocyclone battery, as shown schematically in figure 15.9.

Considering the following variables:

- Sa – flow of solids in the new mill feed – t/h;
- Aa – water flow that accompanies the feed of ore – m3/h or t/h;
- U – ore moisture %;
- dr – true density of solids;

- A = Ae + As – total flow of new water from the circuit – m3/h or t/h;
- Ae – new water flow at the mill inlet – m3/h or t/h;
- As - new water flow at the inlet of the mill discharge box – m3/h or t/h;

- Sd – flow of solids in the discharge of the mill – t/h;
- Ad – water flow at the mill discharge – m3/h or t/h;
- dd – density of the pulp at the mill discharge;

- Sh – flow of solids in the hydrocyclone feed – t/h;
- Ah – water flow at the entrance of the hydrocyclone – m3/h or t/h;
- Vh – flow in volume of pulp at the entrance of the hydrocyclone – m3/h;
- Mh – mass flow of pulp at the entrance of the hydrocyclone – t/h;
- dh – density of the pulp at the entrance of the hydrocyclone;

- Su – flow of solids in the circulating load of the circuit – t/h;
- Au – water flow in the circulating load of the closed circuit – m3/h or t/h;
- Vu – flow in volume of pulp in the underflow of the circuit – m3/h;
- Mu – mass flow of pulp in the underflow of the circuit – t/h;
- du – density of the circulating slurry of the closed circuit;
- su - % of solids in the pulp of the underflow;

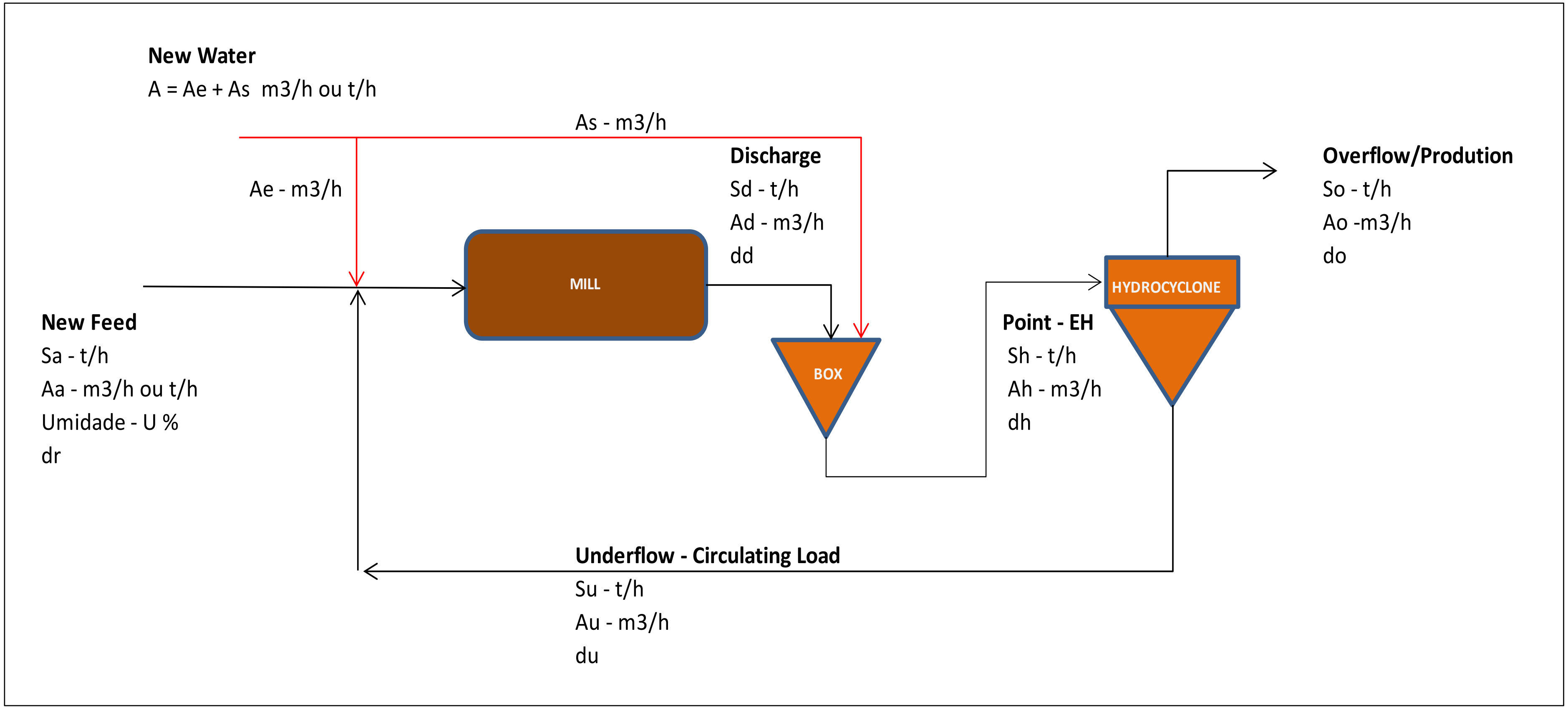


Figure 15.9 – Generic diagram of a wet grinding circuit, closed with classifier (hydrocyclone).

- So – flow of solids in the overflow/production of the circuit – t/h;
- Ao – water flow in the overflow/production of the circuit – m3/h or t/h;
- Vo – flow in volume of pulp in the overflow/production of the circuit – m3/h;
- Mo – mass flow of pulp in the overflow/production of the circuit – t/h;
- do – density of the pulp in the overflow of the circuit;
- so - % of solids in the overflow pulp.

Once having made these considerations about the closed grinding circuit of figure 15.9, one can write to the entry point of the hydrocyclone battery (EH point):

Solids Mass Balance

$Sh = Sa + Su = So + Su$, or

$$So = Sa, \text{ in t/h.} \qquad \text{Eq. 15.37}$$

In other words, the production of solids in the circuit (So) is independent of the circulating load (Su) and is equal to the new supply of solids to the mill (Sa).

Water Balance

$Ah = Aa + Ae + As + Au = Ao + Au$, or

$Ao = Aa + (Ae + As)$, then

$$Ao = Aa + A \text{ m3/h or t/h .} \qquad \text{Eq. 15.38}$$

Similarly, the flow of water in the overflow of the hydrocyclone battery (Ao), which forms the pulp with the solids produced, is independent of the circulating load of water (Au), being equal to the new water flow of the circuit (A), which adjusts the densities of the pulps of the circuit, added to the water that accompanies the solids in the feed of the mill (Aa), due to natural humidity (U%).

d.1 - *Circulating load of closed circuit grinding as a function of densities of hydrocyclone pulps*. (1)

The circulating load (CC) of a closed grinding circuit is very important because, in addition to contributing to the efficiency of the separation by size in hydrocyclones, it maximizes the productivity of the mills, avoiding the excess grinding of solid particles.

The circulating load (CC) is defined, in a circuit with the configuration of figure 15.9, as the percentage ratio between the mass flow of solids in the underflow (Su) and the mass flow of solids in the mill feed (Sa), or in the overflow/production of the hydrocyclone (So = Sa). Thus, it follows that:

$$CC = 100.Su/Sa\ \%, \text{ or } CC = 100.Su/So\ \%. \qquad \text{Eq. 15.39}$$

The control of the circulating load in large grinding circuits, where the solid and water flows are difficult to measure, can be carried out through indirectly determinations of densities of the pulps involved in the circuit. In the case of wet industrial grinding of iron ore, in a closed circuit with hydrocyclones, this circulating load assumes values in the range of 150 to 300 %. Considering a conventional mill, tubular, of dimensions 17' x 34' (D x L), with a feed rate of 200 t/h of solids, it is verified that the circulating load can reach 600 t/h of solids, which impacts the costs of electric energy for pumping of the hydrocyclone feed pulp.

Thus, it is possible to develop the calculation of this circulating load, through the measurement of the densities of the overflow/product pulps (do), of underflow (du) and of the entrance of the hydrocyclones (dh).

Considering the grinding circuit of figure 15.9, in equilibrium, it can be written that the flows of pulp volume of the overflow/product (Vo), the underflow (Vu) and the feeding of the hydrocyclones (Vh), are related to the respective mass flows, Mo, Mu and Mh , and to the densities of these same pulps, respectively, do, du and dh. Thus, we have:

$Vo = Mo/do$, and

$Vu = Mu/du$.

Dividing, member by member, these two above equations, yields:

$$Vu/Vo = Mu.do/(Mo.du) . \qquad \text{Eq. 15.40}$$

On the other hand,

$Vh = Mh/dh = Vu + Vo$ and $Mh = Mu + Mo$.

Explaining the variable Vu, it can be written that:

$Vu = [(Mu + Mo)/dh] – Vo$, and

taking Vu into the equation Eq. 15.40, results:

$[(Mu + Mo)/dh] – Vo = Mu.do.Vo/(Mo.du)$.

Simplifying, it comes:

$(Mu + Mo)/dh = Mu.do.Vo/(Mo.du) + Vo$,

or

$(Mu + Mo)/dh = (Mu.do.Vo + Vo.Mo.du)/(Mo.du)$,

then

$(Mu + Mo)/dh = Vo. (Mu.do + Mo.du)/(Mo.du)$

Once $Vo = Mo/do$, and replacing Vo in the above equation, it comes:

$(Mu + Mo)/dh = Mo. (Mu.do + Mo.du)/(Mo.du.do)$.

Simplifying, it results:

$(Mu + Mo)/dh = (Mu.do + Mo.du)/(du.do)$, and

$du.do. (Mu + Mo) = dh. (Mu.do + Mo.du)$, and yet

$du.do.Mu + du.do.Mo = dh.do.Mu + dh.du.Mo$

Explaining the variables Mu and Mo, it results:

$Mu.(du.do - dh.do) = Mo. (dh.du - du.do)$ Eq. 15.41

Mu and Mo are the mass flows of the underflow and overflow pulps, respectively. As seen in this earlier chapter, the products of the multiplications of the values of these flows by the values of the percentages of solids of the corresponding pulps, su and so, give the mass flows of solids, Su and So, in t/h, in the pulps of the underflow and overflow.

Thus, it can be written that:

$Su = su.Mu/100$, or $Mu = 100.Su/su$, and

$So = so.Mo/100$, or $Mo = 100.So/so$

Taking the expressions of Mu and Mo above in the equation Eq. 15.41, results:

$100.Su.so. (du.do - dh.do) = 100.So. (dh.du - du.do).su$

From here, one can write:

$Su/So = su. (du.dh - du.do) / [so.(do.du - do.dh)]$

or

$Su/So = su.du. (DH - Do) / [so.do.(du - dh)]$

From the equation Eq. 15.39, the circulating load is expressed by (CC %):

$CC = 100.Su/So$, or

$Su/So = CC/100$

Substituting in the above equation, it comes:

$$CC = 100.\ su.du.\ (DH - Do) / [so.do.(\ du - dh)] \qquad \text{Eq. 15.42}$$

On the other hand, the percentages of solids of the underflow and overflow pulps, respectively, su and so, can be expressed as functions of the densities of these pulps, du and do, and also of the true density of the solids involved in the circuit, dr. This has already been shown, through the equation Eq. 15.4, when the generic expression was written as:

$\%$ Solids in pulp $= f(dp, dr) = 100.\ (dp - 1).dr/[(dr - 1).dp]$ %, where:

dp – pulp density and

dr – true density of solids.

By analogy, the equivalent expressions are written for the underflow and overflow pulps of the grinding circuit:

- for the underflow pulp: $su = 100.\ (du - 1).dr/[(dr - 1).du]$, and
- for the overflow pulp: $so = 100.\ (do - 1).dr/[(dr - 1).do]$

Taking su and so into the expression of the circulating load CC, equation Eq. 15.42, and simplifying, yields:

$$CC = 100.\ (du - 1).dr.du.\ (dh - do).\ (dr - 1).do / [(do - 1).dr.do.(\ du - dh).\ (dr - 1).du]\ .$$

From this last equation, the generic expression for the calculation of the *circulating load* of the closed grinding circuit can be calculated as *a function only of the pulp densities involved in the circuit*:

$$\mathbf{CC = f(dh, do, du) = 100.\ (du - 1).\ (dh - do) / [(do - 1).(\ du - dh)]\ \%} \qquad \text{Eq. 15.43}$$

Where:

- CC – circulating load of the grinding circuit %;
- dh – density of the pulp at the entrance of the hydrocyclone;

- do – density of the pulp in the overflow of the hydrocyclone;
- du – density of the pulp in the underflow of the hydrocyclone.

In fact, this generic equation can be used to calculate the *circulating load* of any closed grinding circuit in aqueous medium, since the function only depends on the densities of the pulps, involved in the grinding circuit.

Typically, the grinding of iron ore for pelletizing is very fine, with a cut size in hydrocyclones of the order of 44 microns (0.044 mm). To improve the efficiency of hydrocyclones, the pulp in the feed is very diluted, which implies in obtaining an overflow pulp, also, with low density. It is common to verify, in industrial circuits, overflow pulp density around 1.20. On the other hand, hydrocyclones are calculated to have a high-density underflow pulp (circulating load), being common to work with density around 2.80. This high density allows operating with dense pulp inside the ball mill, with values greater than 2.70, which improves the efficiency of grinding (higher productivity and lower specific energy and grinding media consumptions).

In this context, to make some simulations for a closed circuit of iron ore grinding, it was taken the equation of the circulating load, expressed by the equation Eq. 15.43, and the following premises were considered:

- The underflow density was fixed at du = 2.80;
- Overflow density was considered at three levels: do = 1.18, 1.20 and 1.22;
- Three curves were drawn: Circulating Load versus Density of the feeding pulp of the hydrocyclones, CC = f(dh), considering the previous premises.

Under these conditions, the following equations arise, CC = f(dh), taking the stipulated values of du and do, into the equation of the circulating load, Eq 15.43:

- For do = 1.18: $CC = 1000.(dh - 1.18)/(2.8 - dh)$;
- For do = 1.20: $CC = 900.(dh - 1.2)/(2.8 - dh)$;
- For do = 1.22: $CC = 818.(dh - 1.22)/(2.8 - dh)$.

The Figure 15.10 shows the configurations of the representative curves of these three equations.

It is evident that these simulations only apply to the conditions stated in the premises, which considered typical values of closed circuits of fine iron ore grinding.

In this figure, there is indicated a shaded area, characteristic of operations with circulating loads in the range of 200 to 250%. For this situation, the range of variation of the feeding density of hydrocyclones (dh) would be from 1.45 to 1.60.

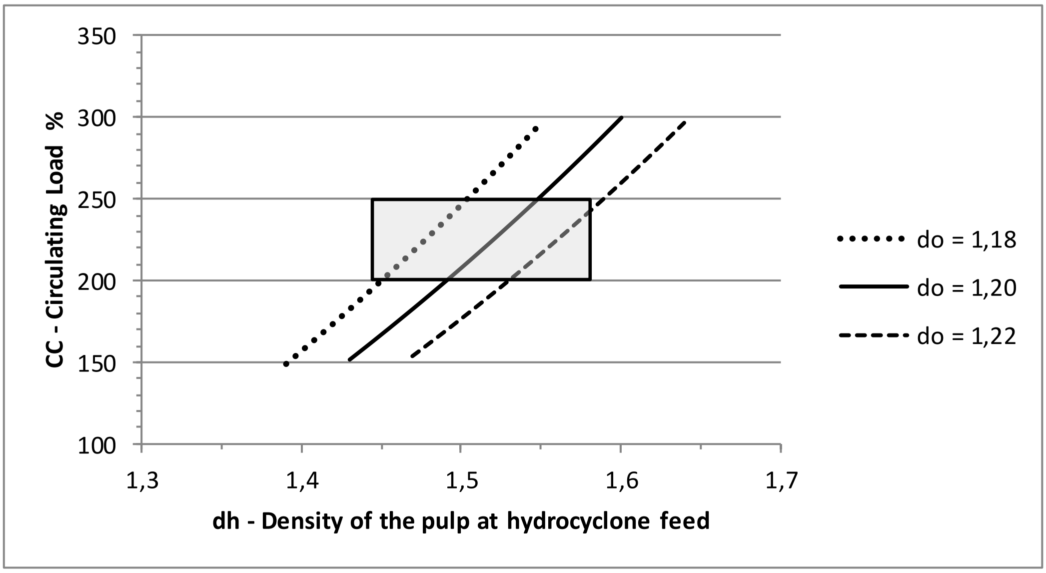


Figure 15.10 – Circulating load grinding circuit as a function of the density of the hydrocyclone feed pulp, for three levels of density of the overflow pulp.

The hydrocyclones of a fine grinding circuit of iron ore must be very well calculated in order to have good efficiency in cutting the sizes of the mineral particles and also adequate partitioning of the water so that the overflow density is very low and the underflow density very high, as previously reported.

With the generic equation of the circulating load, Eq. 15.43, other assumptions can be considered, and new curves generated.

d.2 - *Circulating load of closed grinding circuit as a function of granulometry*. (1)

In certain circuits of fine grinding of iron ore, closed with hydrocyclones, (figure 15.9), there is no sampling device of the feed pulp of the hydrocyclones (EH point). Thus, it is not possible to measure the density of the corresponding pulp, dh, making it impossible to calculate the circulating load, CC = f(dh, do, du). Also, there is no automated system for the online measurement of this density.

In these cases, there is an alternative to determine the circulating load, based on the *granulometry* of the solids *of the mill discharge, overflow and underflow,* a situation in which *the feed pulp of the hydrocyclones is not involved*, prevailing the direct sampling of *the mill discharge pulp*. This method is not very precise, due to aspects related to the fineness of the ground material and the shape factor of the mineral particles. It is difficult to determine the granulometry of the ground minerals precisely, and the calculations provide only an estimation of the circulating load.

It is important to mention that this methodology can also be applied to closed circuits of dry grinding of ores, in general. For the development of the calculation of the circulating load of a closed grinding circuit, based on granulometry, consider the diagram of the figure 15.11. There, the regions of the hydrocyclone and the mill discharge were taken into account. A reference *size "X"* was considered to determine the *passing %* of the solids *of the mill discharge*, *overflow* and *underflow*. This granulometric determination is done in the laboratory, using a *sieve with mesh opening equal to "X'*.

It is easy to see that in terms of the ground solids, the mass flow and the granulometry of the material in the discharge of the mill, remain unchanged at the entrance of the hydrocyclone.

The following variables were then considered, as shown in the diagram in figure 15.11:

- Sd – mass flow of solids in the discharge of the mill – t/h;
- Pd – percentage of mill discharge material, passing through the sieve with mesh opening equal to X - %;

- Sh = Sd – mass flow of solids in the hydrocyclone feed / mill discharge – t/h;
- Ph = Pd – percentage of material of the hydrocyclone inlet, passing through the sieve with mesh aperture equal to X - %;

- Su – mass flow of solids in the underflow or circulating load – t/h;
- Pu – percentage of underflow material, passing through the sieve with mesh aperture equal to X - %;

- So – mass flow of solids in the overflow/production of the circuit – t/h;
- Po – percentage of overflow material, passing through the sieve with mesh opening equal to X - %.

Having made these considerations, some equations can be written that govern the mass balance of the mill-classifier circuit, regarding the percentage of material passing through the aperture mesh X:

Mass balance of solids in the circuit - Sh = So + Su, and

Mass balance of the material passing in the mesh size X - Sh.Ph = So.Po + Su.Pu.

But, since the ground solids have the same characteristics at the discharge of the mill and at the entrance of the hydrocyclone, the following expressions are true:

Sh = Sd and Sh.Ph = Sd.Pd.

Replacing Sh and Sh.Ph in the above equations, the following relationships can be written:

$Sd = So + Su$, and

$Sd.Pd = So.Po + Su.Pu$.

Substituting the value of $Sd = So + Su$ into the above equation yields:

$(So + Su).\ Pd = So.Po + Su.Pu$, or

$So.Pd + Su.Pd = So.Po + Su.Pu$

Reordering and simplifying, it comes:

$Su/So = (Po - Pd)/(Pd - Pu)$ Eq. 15.44

From the equation Eq. 15.39, there is the expression of the circulating load (CC %):

$CC = 100.Su/So$, or

$Su/So = CC/100$.

Replacing in Eq. 15.44, it comes:

$CC = 100.\ (Po - Pd)/(Pd - Pu)$ %, or, finally, the expression of the *circulating load*, indicated below.

$$\mathbf{CC = f(\underline{Pd}, Po, Pu) = 100.\ (Po - Pd) / (Pd - Pu)\ \%} \qquad \text{Eq. 15.45}$$

Since $Pd = Ph$, for the configuration shown in Figure 15.11, one can also write that:

$$\mathbf{CC = f(\underline{Ph}, Po, Pu) = 100.\ (Po - Ph) / (Ph - Pu)\ \%} \qquad \text{Eq.' 15.45}$$

Where:

- CC – circulating load of the grinding circuit - %;
- Pd – percentage of material at mill discharge, passing through the sieve with mesh opening equal to X - %,
- Po – percentage of the material in the overflow, passing through the sieve with mesh aperture equal to X - %,
- Pu – percentage of material in the underflow, passing through the sieve with mesh aperture equal to X - %,
- Ph - percentage of the material in the hydrocyclone feed, passing through the sieve with mesh aperture equal to X - %.

Through this development, it is verified that this equation can be used for any type of classifier (aerocyclones, hydrocyclones, sieve, etc.), in wet and dry circuits.

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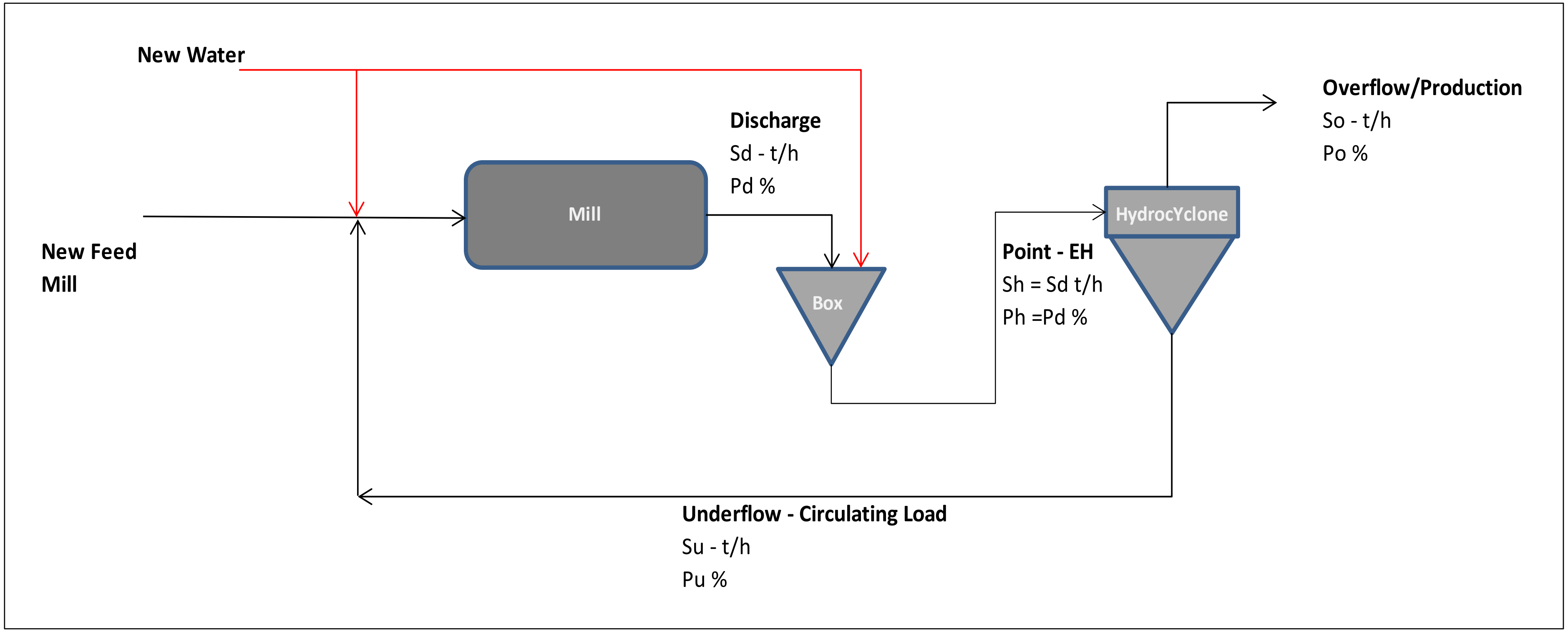


Figure 15.11 – Generic diagram of a wet grinding circuit, closed with classifier, highlighting the hydrocyclone region.

d.3 - *Efficiency of hydrocyclones* (1)

In unit operations involving size classifiers, dry or wet, it is important for process control to determine their efficiency.

Thus, sieves, hydrocyclones, aerocyclones, etc. should be monitored for this parameter, because inefficiencies in the operation of classifiers can generate additional costs, not foreseen in the planning and budget of the industrial plant.

When a *hydrocyclone battery* is installed *in a closed grinding circuit*, as shown in Figure 15.12, it aims to perform a " Y " *size cut* on the ground ore, sending particles larger than Y back to the mill (via underflow/circulating load). In turn, those smaller than Y are directed to the overflow/production of the circuit. In this way, the coarse particles are ground until they reach the desired size. The fines go straight to the production flow, avoiding overgrinding.

Conceptually, the ideal situation of a classifier would be one in which the " Y " size cut would be so perfect that 100% of the *fine particles (smaller than Y) would go into overflow* and, in turn, 100% of the *coarse particles* (larger than Y) would be directed to the underflow. In this situation, the efficiency of the classifier would be maximum and equal to 100%.

For the development of the generic expression for calculating the efficiency of a classifier, the following variables, as shown in Figure 15.12, are considered:

- Y – cut size designed for the classifier – opening of the mesh of a reference sieve, considered in the design of the classifier;
- E – overall efficiency of the classifier - %;
- Ef – efficiency of fine particles, or for separation of fine particles;
- Eg – efficiency of coarse particles, or for separation of coarse particles;

- Sd – mass flow of solids in the discharge of the mill – t/h;
- Pd – percentage of mill discharge material, passing through the sieve with mesh opening equal to Y - %;
- Rd – percentage of mill discharge material, retained in the sieve with mesh opening equal to Y - %;

- Sh = Sd – mass flow of solids in the hydrocyclone feed – t/h;
- Ph = Pd – percentage of material in the hydrocyclone inlet, passing through the sieve with mesh aperture equal to Y - %;
- Rh = Rd – percentage of hydrocyclone inlet material, retained in the sieve with mesh aperture equal to Y - %;

- Su – mass flow of solids in the underflow or circulating load – t/h,

- Pu – percentage of underflow material, passing through the sieve with mesh aperture equal to Y - %;
- Ru – percentage of underflow material, retained in the sieve with mesh aperture equal to Y - %;
- So – mass flow of solids in the overflow/production of the circuit – t/h;
- Po – percentage of overflow material, passing through the sieve with mesh aperture equal to Y - %;
- Ro – percentage of overflow material, retained in the sieve with mesh opening equal to Y - %.

According to the concepts enunciated, it could be said that: (1)

The efficiency of fine particles (Ef) would be equal to the fractional ratio between the mass flow of fine particles, smaller than size Y, in the overflow of the classifier (t/h) and the mass flow of these fine particles in the hydrocyclone feed (t/h);

The efficiency of coarse particles (Eg), in turn, would be equal to the fractional ratio between the mass flow of coarse particles, bigger than size Y, in the underflow of the classifier (t/h) and the mass flow of these coarse particles in the hydrocyclone feed (t/h);

The Overall Efficiency (E) could be understood as the product of the efficiencies of fine and coarse particles, expressed in percentage terms (%).

That said, the following mathematical expressions could be written, involving the variables defined above:

Ef = Po.So/Ph.Sh, Eq. 15.46

Eg = Ru.Su/Rh.Sh, and Eq. 15.47

E = 100.Ef.Eg % Eq. 15.48

On the other hand, the mass balances of fine and coarse particles of the classifier indicate that:

Balance for fines – particles smaller than Y.

Sh = So + Su, or

Su = sh –So, and

Ph.Sh = Po.So + Pu.Su

Bringing Su expression into this last equation yields:

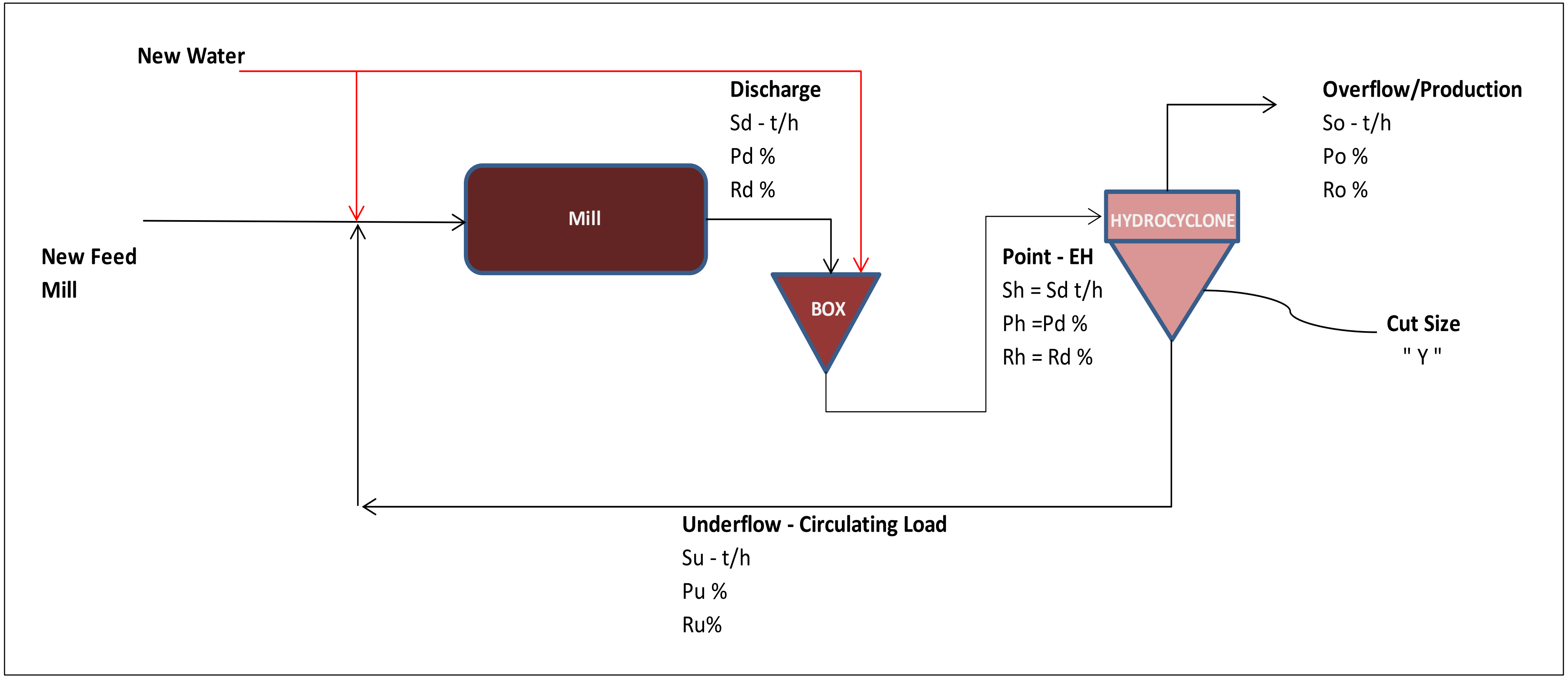


Figure 15.12 – Generic diagram of a wet grinding circuit, closed with hydrocyclone and cut-size " Y ".

$Ph.Sh = Po.So + Pu.(Sh - So)$.

Simplifying:

$(Ph - Pu).Sh = (Po - Pu).So$, or

$So/Sh = (Ph - Pu)/(Po - Pu)$.

Taking the expression of So/Sh into the equation Eq. 15.46, it comes

$Ef = Po.So/Ph.Sh = Po.(Ph - Pu)/[Ph.(Po - Pu)]$ and hence it results:

$$\mathbf{Ef = f(Ph, Po, Pu) = Po.(Ph - Pu) / [Ph.(Po - Pu)]} \text{ .} \qquad \text{Eq. 15.49}$$

Balance for coarse particles – particles bigger than Y.

Analogously as done for the fines, it may be written that:

$Sh = So + Su$, or

$So = sh - su$, and

$Rh.Sh = Ro.So + Ru.Su$

Bringing the expression of So into this last equation yields:

$Rh.Sh = Ro.(Sh - Su) + Ru.Su$.

Simplifying:

$Su/Sh = (Rh - Ro)/(Ru - Ro)$.

Taking the expression of Su/Sh into the equation Eq. 15.47, it comes:

$Eg = Ru.Su/Rh.Sh = Ru.(Rh - Ro) / [Rh.(Ru - Ro)]$. Hence it results:

$Eg = f(Rh, Ro, Ru) = Ru.(Rh - Ro) / [Rh.(Ru - Ro)]$.

However, at the same point in the circuit, the percentages of passing and retained material (P and R) in the same mesh of size Y are complementary by 100%. Hence, it can be written that:

- at the entrance of the hydrocyclone: $Rh = 100 - Ph$ %;
- in the overflow: $Ro = 100 - Po$ % and
- in the underflow: $Ru = 100 - Pu$ %.

Substituting Rh, Ro, and Ru into Eg equation, above, yields:

$Eg = (100-Pu).(Po - Ph) / [((100 - Ph).(Po - Pu)]$.

Then we obtain the expression of the efficiency of coarse particles, as a function of Ph, Po and Pu:

$$\mathbf{Eg = f(Ph, Po, Pu) = (100 - Pu).(Po - Ph) / [((100 - Ph).(Po - Pu)]} \qquad \text{Eq. 15.50}$$

Taking the expressions of Ef and Eg from the equations Eq. 15.49 and 15.50 to the equation of E, Eq. 15.48, one obtains the general equation of the efficiency of the classifier:

$E = 100.Ef.Eg = 100.Po.(100 - Pu).(Po - Ph).(Ph - Pu) / [Ph.(100 - Ph).(Po - Pu).(Po - Pu)]$.

That is, the function **<u>E</u>** = f(**<u>Ph</u>**, Po and Pu) arises, expressed by:

$$\mathbf{E = 100.Po.(100 - Pu).(Po - Ph).(Ph-Pu) / [Ph.(100 - Ph).(Po-Pu)^2]\ \%} \qquad \text{Eq. 15.51}$$

Where:

- E – overall efficiency of the classifier - %;
- Ph = Pd – percentage of material from the hydrocyclone inlet, passing through the sieve with mesh aperture equal to Y - %;
- Pu – percentage of underflow material, passing through the sieve with mesh aperture equal to Y - %;
- Po – percentage of overflow material, passing through the sieve with mesh opening equal to Y - %.

Thus, the efficiency of a classifier can be calculated through simple determinations of the granulometries of the feed, overflow, and underflow materials. For this purpose, there is a need to collect samples at these three points of the circuit. Sometimes sampling in the input stream of the classifier is difficult, or even of dubious representativeness. In this case, if the classifier is part of a circuit like in figure 15.12, in which there is no modification of the particle size of the material, between the mill discharge and the classifier, the material collected in the mill discharge represents that of the classifier input. And in this condition, it can be written that:

Pd = Ph.

Substituting in the equation Eq. 15.51, one obtains the alternative equation of the efficiency of the classifier, as a function of **Pd, E** = f(**Pd**, Po, Pu):

$$\mathbf{E = 100.Po.(100\text{-}Pu).(Po\text{-}Pd).(Pd\text{-}Pu)/\ [Pd.(100\text{-}Pd).(Po\text{-}Pu)^2]\ \%} . \qquad \text{Eq.' 15.51}$$

Where:

- E – overall efficiency of the classifier - %,
- Pd = Ph – percentage of mill discharge material, passing through the sieve with mesh opening equal to Y - %,
- Pu – percentage of underflow material, passing through the sieve with mesh aperture equal to Y - %,
- Po – percentage of overflow material, passing through the sieve with mesh opening equal to Y - %.

It is important to note that these equations, Eq. 15.51, and Eq.' 15.51, may be used for any type of classifier, operating wet or dry, including sieves, where, in the latter, an adjustment of the parameters of the equations must be made according to the concepts of oversize Ξ underflow and undersize Ξ overflow. Also, it should be noted that this efficiency *does not depend on* the type of material submitted to classification.

The *efficiency* of cyclone, hydrocyclone and *aeroclassifiers* is relatively *low*, due to the physical principles that govern the separation of fine and coarse particles (centripetal force, movement of solids in turbulent flows of liquid and gaseous fluids, variation of viscosity of flows, shape factor of mineral particles, etc.). There is always a drag of coarse particles to overflow and fine particles to underflow, which lowers the efficiency of this type of equipment. In industrial practice, efficient cyclones have this index in the range of 50 to 60%. Despite this fact, classifiers of this type are widely used in the industry, given the following factors:

- Low capital and operating costs;
- Ease of maintenance;
- High processing capacity, in equipment of small dimensions;
- Ease of adjustment of the cutting size (modification of opex, vortex, etc.);
- And so on.

Taking the equations Eq. 15.45 and Eq.' 15.51, simulations could be made for a *closed circuit of iron ore grinding*, simultaneously, analyzing the parameters related to the circulating load of the circuit and the efficiency of the classifier. To this end, the following premises are adopted:

- Grinding circuit with the configuration in Figure 15.11;
- Y = 44 microns (325 meshes sieve opening of the Tyler series) – projected cut size for the classifier;
- Ph = Pd = 50% (constant) – percentage of material of the hydrocyclone feed (Ph), or of the mill discharge (Pd), passing through the sieve with mesh aperture equal to 44 microns,
- Pu = 25, 30, 35 and 40%, percentage of underflow material, passing through the sieve with mesh opening equal to 44 microns.

Considering the premises and equations cited, the following expressions can be developed for calculations of the *circulating load* of the closed grinding circuit, CC, and the *efficiency* of the classifier, E.

Circulating Load (Eq. 15.45)

General equation - CC = 100. (Po – Ph) / (Ph – Pu) % .

As Ph = 50%, constant, CC = 100. (Po – 50) / (50 – Pu) .

For the three levels of Pu, 25, 30 and 35%, the following equations result:

- Pu = 25%: CC_{25} = (4.0). (Po – 50); Eq. 15.52
- Pu = 30%: CC_{30} = (5.0). (Po – 50); Eq. 15.53
- Pu = 35%: CC_{35} = (6.7). (Po – 50); Eq. 15.54
- Pu = 40%: CC_{40} = (10.0). (Po – 50). Eq. 15.55

Classifier Efficiency (Eq.' 15.51)

General equation:

$E = 100.Po. (100 - Pu). (Po - Ph). (Ph – Pu)/ [Ph.(100 – Ph).(Po - Pu)^2]$ %

As Ph = 50%, constant, $E = 0.04.Po. (100 – Pu). (Po – 50). (50 – Pu)/(Po – Pu)^2$

For the three levels of Pu, 25, 30 and 35%, the following equations result:

- Pu = 25%: $E_{25} = 75.Po.\,(Po - 50)/(Po - 25)^2$; Eq. 15.56
- Pu = 30%: $E30 = 56.Po.\,(Po - 50)/(Po - 30)^2$; Eq. 15.57
- Pu = 35%: $E35 = 39.Po.\,(Po - 50)/(Po - 35)^2$; Eq. 15.58
- Pu = 40%: $E40 = 24.Po.\,(Po - 50)/(Po - 40)^2$. Eq.13.59

Using the equations Eq. 15. 52 to 15.59, the curves of figures 15.13 and 15.14 were drawn, below. The analysis of these data allows the following comments to be made, keeping in mind the *established premises*:

- The industrial fine grinding of iron ore for pelletizing is usually designed to obtain an overflow (product), with a passing percentage in the mesh of 44 microns in the range of 85 to 95%. Although none of these functions studied depends on this mesh opening, or on the cutting size of the classifier, reference was made to this size only to establish a parallel with industrial practice.

- In Figure 15.13, it is verified that the fineness of the overflow (product) always increases with the increase of the circulating load of the grinding circuit. The shaded area of the figure indicates the field corresponding to overflow with passing percentage in 44 microns in the range of 85 to 95% and circulating load of 150 to 300%. It is seen that the curve for underflow with passing percentage in 44 microns equal to 40% does not meet the delimited area.

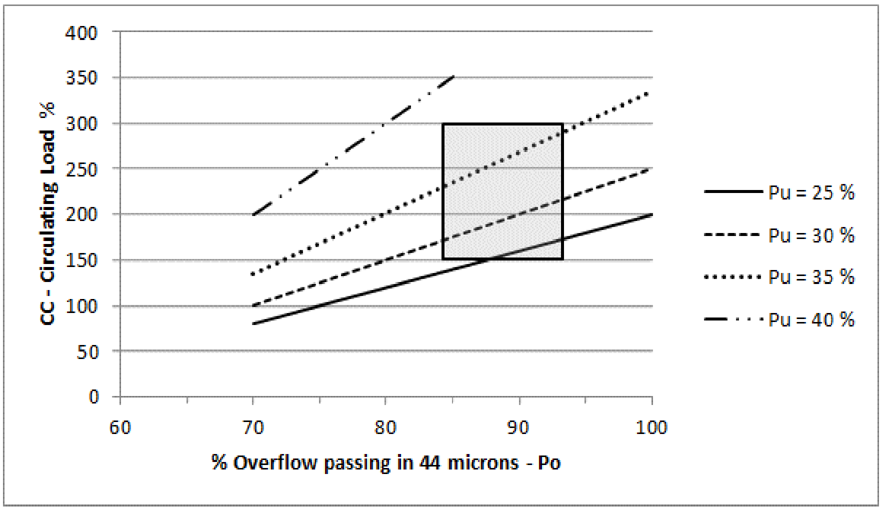


Figure 15.13 – Circulating load of the closed circuit of fine grinding of iron ore, as a function of the percentage of the overflow passing in 44 microns, (Ph = constant = 50%, and Pu varying in the range of 25 to 40%).

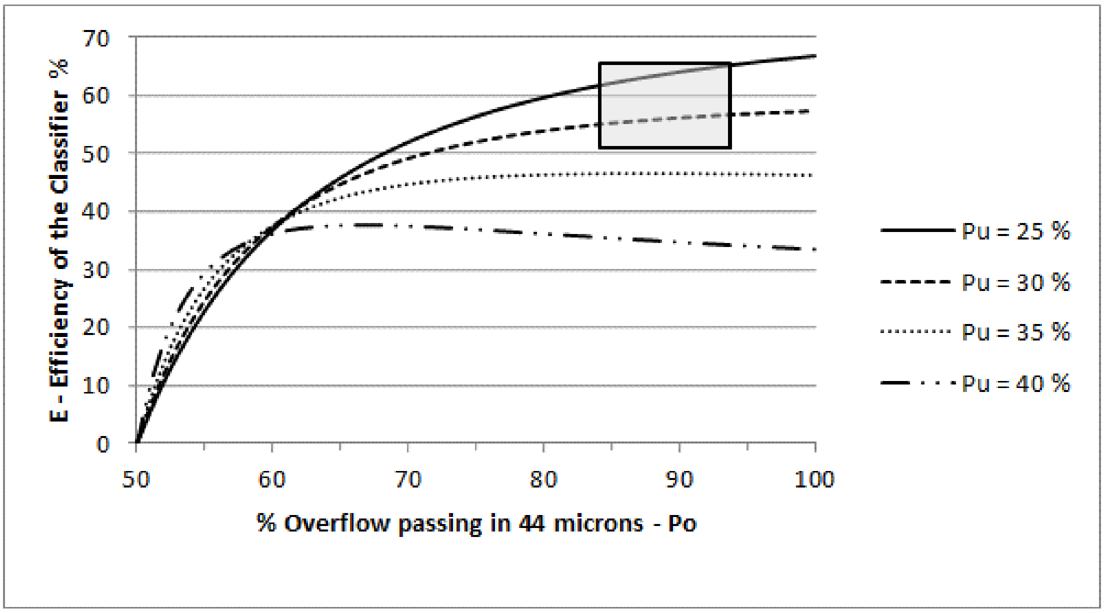


Figure 15.14 – Efficiency of hydrocyclones in closed circuit of fine grinding of iron ore as a function of the percentage of the overflow passing in 44 microns, (Ph = constant = 50%, and Pu varying in the range of 25 to 40%).

- Figure 15.14 shows the classifier efficiency curves as a function of the percentage of overflow passing through 44 microns. The shaded area in the figure indicates the field corresponding to overflow with passing percentage in 44 microns in the range of 85 to 95% and efficiency of the classifier in the range of 50 to 65%. Here, it is verified that the curves for underflow, with passing percentage in 44 microns equal to 35 and 40%, do not meet the delimited area.

- Combined analyses of different assumptions and simulations can contribute to the improvement of industrial iron ore grinding processes.

CHAPTER 16

ASPECTS RELATED TO THE SWELLING AND DISINTEGRATION OF FIRED PELLETS (1, 11, 23 , 24, 25)

The iron ore pellets, originated from hematite and magnetite ores, present as the main crystalline phase the Fe_2O_3, at the end of firing in the pelletizing furnace. To obtain the iron, as reported in Chapter 3, the pellets are subjected to a reduction processing in specific reactors, in environments at high temperatures, containing solid and gaseous reductants: C, CO and H_2.

The reduction of hematite occurs in stages, when the *oxygen atoms are gradually eliminated* from the mineral phases, and the *iron is generated* into the productive system. Taking the reduction by CO gas, just as an example, the reduction model would be as follows:

$$Fe_2O_3 \rightarrow Fe_3O_4 \rightarrow FeO \rightarrow Fe$$

The process steps are governed by the following chemical reactions:

$$3Fe_2O_3 + CO \rightarrow 2Fe_3O_4 + CO_2$$

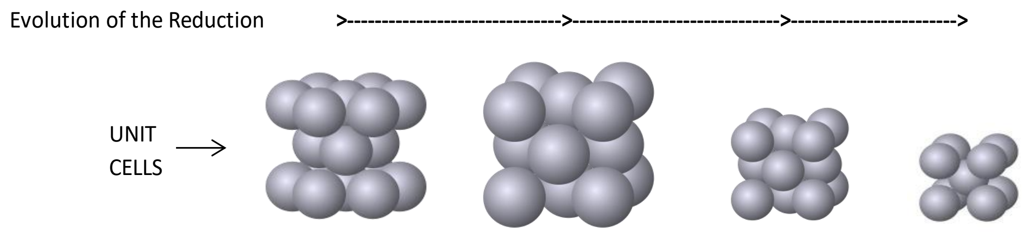
$$Fe_3O_4 + CO \rightarrow 3FeO + CO_2$$

$$FeO + CO \rightarrow Fe + CO_2$$

On the other hand, the solid phases involved in these reduction reactions are crystalline and have well-known and well-studied structures. (23, 24, 25) Generally, without keeping proportions and/or dimensions with the structural reality, figure 16.1 shows examples of unit cells of the crystals of these phases, Fe_2O_3, Fe_3O_4, FeO and Fe. There, some characteristic data of these crystalline phases are also shown.

The schematic drawings of the cells are illustrative only, to show the possible positioning and arrangement of the iron and oxygen atoms at vertices, faces, and interiors of the cells. In this particular, it should be mentioned that the iron atom is much larger than that of oxygen, and that structural calculations show that these dimensions are, respectively, 156×10^{-12} m and 48×10^{-12} m, or 156 pm and 48 pm (picometer). With such different dimensions, these atoms are ordered, for each phase, obeying the atomic proportion of the respective chemical formulas.

In figure 16.1, the different classes of crystals can be seen, moving from the compact hexagonal configuration of hematite, evolving to the face-centered cubic of magnetite and wustite, and ending in the body-centered cubic of iron in the solid state. Thus, as the reduction progresses, there is this sequence of crystalline changes, with large movements and repositioning of the iron and oxygen atoms. In the case of oxygen, gradually, the atoms are released by the reaction with the reducing gas CO, with a concentration of iron atoms. In addition to these changes, it is also found that unit cells have very different volumes. Starting from the unit cell of the hematite there is a large volume expansion (+95%) in the passage to the magnetitic phase and, hence, large contractions in the passages to wustite (-74%) and iron (-92%).



PHASE Formula	Hematita Fe_2O_3	Magnetita Fe_3O_4	Wustita FeO	Iron Fe
System	Trigonal	Isometric	Isometric	Isometric
Class	Hexagonal Compact	Cubic Face- centered	Cubic Face- centered	Cubic Body-centered
Volume of unit cell (A^3)	303 (Reference)	592 (+ 95 %)	80 (- 74 %)	24 (- 92 %)
Specific Mass (g/cm^3)	5,26	5,18	5,70	7,87

Figure 16.1 (23, 24) – Structural evolution of solid phases, during the reduction of hematite to iron, highlighting the differences in the volumes of unit cells, in cubic Angstrom - A^3, (1 A = 10^{-10} m).

According to many authors and scholars on the subject of hematite reduction (11, 25) these structural changes explain the phenomena of *swelling and disintegration at low temperature* that occur when iron ores and agglomerates are reduced in steel reactors. The evasion of oxygen atoms from the crystalline system and the continuous atomic repositioning in the unit cells of different sizes during the reduction cause many internal stresses in the remaining solid phases, creating fractures in the crystals and porosity in the grains. The set of stresses, fractures, gaps, and microscopic pores lead the solid structure to an *apparent volume* of greater dimension than the initial state of compact hematite (*swelling).* Likewise, portions of the structure can rupture, generating fragments and characterizing the phenomenon of *disintegration*. This occurs, not only, but mainly, in the first step of the reduction process, passage from *hematite to magnetite*, when the great expansion of volume of the unit cells occurs (+95%, in the passage from compact hexagonal to cubic face centered).

It is important to emphasize that, in this process of reduction, there is an *increase of this apparent volume*, *(*see Chapter 11), despite the fact that, continuously, there is a reduction in the mass of the different phases and, consequently, *a continuous reduction of the true volume of the solid phases*, until the final iron metal (Fe) is reached.

Table 16.1 – Structural evolution of solid phases during the reduction of hematite to metallic iron, emphasizing the different dimensional parameters.

Evolution of Reduction	>------------------------>---------------------------->-------------------------->			
PHASE Formula	HEMATITE Fe2O3	MAGNETITE Fe3O4	WUSTITE FeO	IRON Fe
Reduction Degree (%)	0	10	33	100
Evolution of solid mass during reduction of 1 mol of Fe2O3 (g)	160	155	144	112
True Density	5,26	5,18	5,70	7,87
True Volume (cm^3)	30 (Reference)	30 0%	25 (- 17%)	14 (- 53%)
Volume of unit cell (A^3)	303 (Reference)	592 (+ 95%)	80 (- 74%)	24 (- 92%)
Number of unit cells	9,9 E+22 (Reference)	5,1 E+22 (- 48%)	31,2 E+22 (+ 215%)	58,3 E+22 (+ 489%)

This structural porosity, surged from the *expanded apparent volume*, is evidenced in industrial practice, in direct reduction plants, where the entire hematite-iron reduction process takes place in the solid state. There, the metallic iron product obtained is quite porous, presenting low *apparent density*, contrasting with the *true density of iron*, which is equal to 7.87, (zero porosity). The product of direct reduction is so porous that in the past it was called *sponge iron*.

Using the concepts of solid-state in physical chemistry and of hematite reduction, table 16.1 was drawn, showing some data that facilitate the understanding of the facts reported above. It was highlighted the evolution of the *true volume* of solid material, generated in the reduction process, starting from "1 mol" of Fe2O3, or 160 g of hematite. Analyzing also the data referring to the evolution *of the number of unit cells*, one can imagine how intense are the movement of iron and oxygen atoms and the structural rearrangements.

CHAPTER 17

REDUCIBILITY OF IRON ORES AND AGGLOMERATES (1)

The reduction of iron ores and their agglomerates is one of the most important issues in the steel industry. As seen earlier, in Chapters 3 and 16, the reduction of iron oxides occurs in stages, by reaction with some reductants, usually C, CO and H2. The reduction reactions occur inside the blast furnaces and direct reduction reactors, producing *primary iron,* from the iron oxides. These reduction reactions depend on several factors. The structural characteristics of ores and agglomerates, dependent on the genesis and other geological aspects, greatly impact the speed of reduction reactions and, consequently, the productivity of steel reactors.

Conceptually, considering a process of this nature in progress, the degree of reduction of the iron oxide involved in the reduction reaction, at any instant t, is defined as the fractional relationship between the mass of oxygen eliminated, up to that instant, and the initial mass of oxygen, contained in the iron oxide. The mathematical expression of this concept is given by:

$R = (Oi - O)/Oi$

Expressed as a percentage, it comes:

$$R = 100.\ (Oi - O)/Oi\ \%\ . \qquad \text{Eq. 17.1}$$

Where:

R – degree of reduction of iron oxide, at the instant t of the reduction process;

Oi – mass of oxygen of iron oxide, at the initial instant of the reduction process;

O – mass of oxygen of iron oxide, at instant t;

(Oi – O) - represents the loss of mass of iron oxide, or the same as the loss of mass of oxygen of the iron oxide subjected to the reduction process, up to the instant t.

The speed of the reduction reactions of iron oxides, which, in other words, characterizes the speed at which *oxygen is eliminated* or the *iron produced*, is one of the most important variables involved in the reduction processes.

The study of the kinetics of the reduction reactions of iron oxides shows that the reaction rate between the iron oxide and the reductant (C, CO, H_2) is of the first order, that is, it is proportional to the mass of oxygen, remaining in the reactant iron oxide. Thus, the rate of removal of oxygen from the oxide is given by the following differential equation:

$$dO/dt = - k.O \,, \qquad \text{Eq. 17.2}$$

being:

dO/dt – rate of oxygen removal at instant t;

K – kinetic constant (depends on iron ore and other factors);

O – mass of oxygen remaining in iron oxide at instant t.

Since the degree of reduction is a function of the mass of oxygen, R = f(O), as shown in the equation Eq. 17.1, it may be written that:

$R = 100. (Oi - O)/Oi$, or

$$O = (100 – R).Oi/100 \,. \qquad \text{Eq. 17.3}$$

Differentiating this equation in relation to the variables O and R, it comes:

$$dO = - (Oi/100).dR \qquad \text{Eq. 17.4}$$

Replacing the expressions of O and dO of Eq. 17.3 and 17.4 in equation Eq. 17.2, it results the differential equation transformed into R:

$$- (Oi/100).dR = - k. [(100 – R).Oi/100].dt \,.$$

Simplifying, it results in the expression of the speed of reduction, or *reducibility*:

$$dR/dt = k.(100-R) \qquad \text{Eq. 17.5}$$

Where:

dR/dt – ***reducibility***, or rate of reduction of iron oxide at time t;

k - kinetic constant,

R – reduction degree of iron oxide at instant t.

The reducibility of iron oxides decreases with reaction time. That is, as the degree of reduction (R) increases, the speed of the reaction decreases, as shown by the equation Eq. 17.5. Due to this fact, in order to make the comparison of different types of iron ore, *laboratory tests* were standardized by the ISO Committees, aiming at measuring the reduction degree (R) with fixed test time.

In the case of reducibility, (dR/dt), it is variable throughout the reduction test. Thus, there was a standardization process to determine it in two moments of the laboratory test:

- $(dR/dt)_{40}$: at the instant t_{40}, when the degree of reduction (R) reaches the value equal to 40%, in the initial phase of the wustite transformation ;

  and

- $(dR/dt)_{80}$: at the moment t_{80}, when the degree of reduction (R) reaches 80%, the final phase of the wustite-iron reduction.

It was also standardized that these two reducibility indexes, characteristic of each type of iron ore, would be calculated through the following parameters that are obtained in the laboratory reduction tests, as detailed below.

- $(dR/dt)_{40} = f(t_{30}, t_{60})$, where: t_{30} and t_{60} are the times required to reach the degrees of reduction (R) equal to 30% and 60%, respectively, in the standardized laboratory tests,

  and

- $(dR/dt)_{80} = f(t_{75}, t_{85})$, where: t_{75} and t_{85} are the times required to reach the degrees of reduction (R) equal to 75% and 85%, respectively, in the same standardized laboratory tests.

For the development of the expressions for calculating these indixes, characteristic of the ores, the reducibility equation, Eq. 17.5, is considered:

$dR/dt = k.(100-R)$, or

$dR/(100-R) = k.dt$

Integrating both members of the equation into the limits [0,R] and [0,t] yields:

$\int_0^R [1/(100-R)].dR = k.\int_0^t dt$, thus

$[Ln\ (100 – R]_{0,\ R} = - k.[\ t\ ]_{0,\ t}$, or

$Ln\ (100 – R) – Ln\ (100) = - k.t$, or Eq. 17.6

$Ln\ [(100 – R)/100] = - k.t$, and finally

$100 – R = 100.e^{-k.t}$.

It then results:

$$R = 100.\ (1 – e^{-k.t}) \qquad \text{Eq. 17.7}$$

Where:

R % = reduction degree of iron oxide at time t;

k = kinetic constant that depends on the nature of the iron oxide and the conditions of the standardized laboratory test, in min^{-1};

t – time, usually measured in min.

Each material has its characteristic reduction curve. Some scholars of the subject use the "k" factor, measured in the standardized laboratory test, to characterize and distinguish the different types of iron ore and agglomerates. This is the case, for example, of a certain industrial group of the direct reduction field that determines the value of k as the slope of the line representative of the function:

$$Ln\ [100/(100-R)] = k.t \qquad \text{Eq. 17.8}$$

This function was developed from the equation Eq. 17.6.

Taking the equation Eq. 17.7, a curve of R x t was studied, considering a hypothetical iron ore that, at the end of the standard laboratory reduction test, with duration of t =

120 min, has reached a degree of reduction R = 95%. In this case, the value of k would be equal to 0.025 min^{-1}, easily calculated by the equation Eq. 17.7, as follows:

$R = 100.\ (1 - e^{-k.t})$

As R = 95% and t = 120 min, it comes:

$95 = 100.\ (1 - e^{-120.k})$

Simplifying and explaining k, it results:

$e^{-120.k} = 0.05$.

Calculating the Napierian logarithm (Ln) on both sides, we arrive at the value of k, characteristic of this hypothetical iron ore:

$-120.k = \ln 0.05$, then

$$k = 0.025\ min^{-1}\ .$$

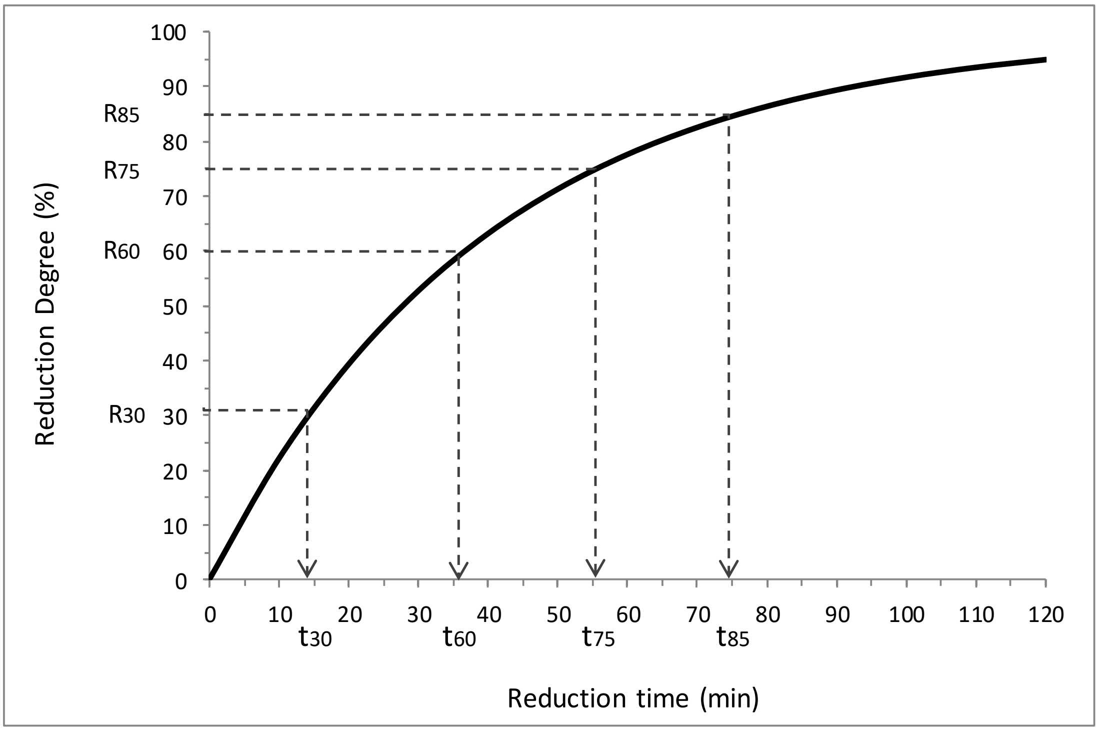


Figure 17.1 – Evolution of the reduction degree (R) as a function of time (t), for a hypothetical iron ore sample, with a value of k = 0.025 min^{-1}.

Thus, Figure 17.1 was drawn from the equation Eq. 17.7 and the value of k=0.025 min^{-1}, showing the evolution of the degree of reduction (R) over time (t), and the arrows indicative of the paths, for the determination of the characteristic values t_{30}, t_{60}, t_{75} and t_{85}, used for the calculations of the standardized indices of reducibility at 40% and 80% reduction.

In the case of the reducibility of this hypothetical iron ore, with k = 0.025 min^{-1}, *two characteristic equations* of the reducibility index can be studied. In this way, the following functions can be characterized for a given ore:

dR/dt = f(R) and

dR/dt = f(t).

For the development of the expressions of these functions let the general equation Eq. 17.5 be considered first:

dR/dt = f(R) = k.(100-R)

For the case of the hypothetical iron ore considered, with k = 0.025 min^{-1}, it results:

$$dR/dt = f(R) = 0.025.\,(100-R)\ \ \%/min\,. \qquad \text{Eq.17.9}$$

For the development of the general expression of reducibility as a function of the reduction time of iron ore, one must take the general expression of the equation Eq. 17.7:

$R = 100.\,(1 - e^{-k.t})$.

Transforming, it comes:

$R = 100 - 100.e^{-k.t}$

Differentiating both members of the equation and simplifying it, yields:

$dR = -100.\,(\,e^{-k.t}\,).\,(\,-k.dt)$, or

$dR = [100.k.e^{-k.t}].dt$

Replacing the value of k = 0.025 min^{-1} results in the expression of the reducibility of the hypothetical ore, as a function of time:

$$dR/dt = f(t) = 2{,}5.e^{-\,0{,}025.t}\ \%/min \qquad \text{Eq.17.10}$$

With the equations Eq. 17.9 and 17.10, the graph of Figure 17.2 was cdrawn, where the reducibility curves of the hypothetical iron ore can be seen, as a function of time and the degree of reduction. The most used curve is that of reducibility as a function of time, since it gives a better sense of the time elapsed, up to a certain point of the reduction process. In any case, both curves show that the reduction velocity, dR/dt, is decreasing as the reduction progresses, providing the same results for the same considered instant.

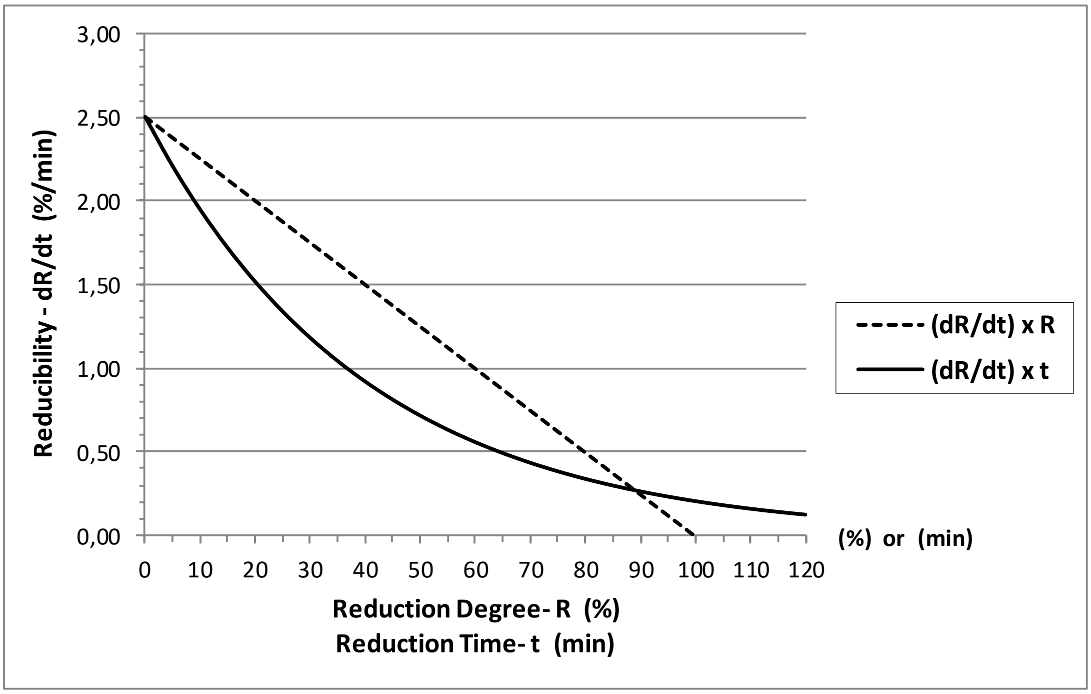


Figure 17.2 – Evolution of iron ore reducibility (dR/dt), in laboratory test, as a function of the reduction degree (R) and time (t), being k= 0.025 min^{-1}.

As mentioned earlier, scholars of the subject of iron ore and agglomerates have standardized the reduction tests in the laboratory and the way of calculating the two reducibility indices at the points where the degree of reduction reaches the marks of 40 and 80%. For the development of the calculation expressions of these characteristic indices of the ores, $(dR/dt)_{40}$ and $(dR/dt)_{80}$, the general equation of *reducibility,* Eq. 17.5, should be considered:

dR/dt = k.(100-R), or

dR/(100-R) = k.dt

Integrating both members of the equation, in the limits of R [30, 60] and t [t_{30}, t_{60}], yields:

$\int_{30}^{60}$ [1/(100-R)].dR = k.$\int_{t30}^{t60}$ dt

From there, it comes:

[Ln (100 – R)]$_{30}^{60}$ = - k.[t]$_{t30}^{t60}$, or

Ln(40) – Ln(70) = - k.(t_{60} – t_{30}) .

It follows, then, that:

k = 0.5596/(t_{60} – t_{30})

Taking the value of k, in the expression of the equation Eq. 17.5, dR/dt = k.(100-R), and making R = 40%, yields:

$(dR/dt)_{40}$ = 60 . 0,56/(t_{60} – t_{30}),

or, finally,

$(dR/dt)_{40}$ = 33.6 /(t_{60} – t_{30}) %/min . Eq. 17.11

Similarly, one can arrive at the expression of $(dR/dt)_{80}$, from the general equation of *reducibility*, Eq. 17.5:

dR/dt = k.(100-R), or

dR/(100-R) = k.dt

Integrating both members of the equation into the limits of R [75, 85] and t [t_{75}, t_{85}] yields:

$\int_{75}^{85}$ [1/(100-R)].dR = k.$\int_{t75}^{t85}$ dt .

It results:

[Ln (100 – R)]$_{75}^{85}$ = - k.[t]$_{t75}^{t85}$, or

Ln(15) – Ln(25) = - k.(t_{85} – t_{75}) .

It follows, then, that:

k = 0,51/(t_{85} – t_{75}) .

Taking the value of k, in the expression of the equation Eq. 17.5, dR/dt = k.(100-R), and making R = 80%, yields:

$(dR/dt)_{80} = 20 \,.\, 0{,}51/(t_{85} - t_{75})$,

or, finally,

$$\mathbf{(dR/dt)_{80} = 10.2 /(t_{85} - t_{75})\ \%/min} \qquad \text{Eq. 17.12}$$

The equations Eq. 17.11 and 17.12 are widely used for the calculations of these two reducibility indices in *laboratory reduction tests* and standardized by international organizations such as ISO and JIS. The reason for the standardization is so that the indices of different iron ores, sinter and pellets can be compared as to their speed of reduction. As is well known, the reduction kinetic strongly depends on the temperature and also on other parameters, such as the composition of the reducing gases, the size of the ore/agglomerate particles, etc. If there was no standardization of the test, the indexes of the different materials would not be comparable.

The index $(dR/dt)_{40,}$ which measures the speed of reduction at the beginning of the wustite-iron transformation, is most commonly used for the evaluation of ores and agglomerates for use in blast furnaces.

On the other hand, the $(dR/dt)_{80,}$ which evaluates the reducibility in the final stages of wustite-iron reduction, that is, when the metallic iron is formed in solid state, is more used for the evaluation of iron oxides for direct reduction.

CHAPTER 18

PELLETS FOR DIRECT REDUCTION AND METALLIZATION (1)

In Chapters 3 and 5, aspects related to the direct reduction technology and the chemical specifications of the iron ore pellets used to feed the reactors of this steel processing were shown. In general, direct reduction pellets should contain very low acid gangue content, and therefore the highest possible iron content.

As direct reduction (DR) had its industrial escalation in the early 1970's, at that time, there were still reserves and availability of high-grade lump ore, and adequate quality for use in such process. Along the time, these reserves were depleted, and the lump ore left room for the expansion of the use of pellets in the burden of DR reactors. With many advantages, the pellets gradually replaced the lump ore, having been responsible for the great increase in the productivity of the reactors. Nowadays, as a rule, the reactors of this technology are fed with 100% of DR pellets.

The *direct reduction*, unlike the blast furnace, performs the *reduction* of iron oxides in the *solid state*, generating a highly metallized product (DRI/HBI), which feeds the electric arc furnaces (EAF), in steel production. Thus, the DRI/HBI competes with the scrap of iron and steel, a product with a metallization degree practically equal to 100%, in the EAF feed. This motivates permanent studies to increase the reducibility of the direct reduction pellets, in order to push the productivity of the reactors and improve the metallization degree of the product, DRI/HBI.

Among the metallurgical properties of direct reduction pellets, the most important is *reducibility*, which translates into their ability to generate *metallic iron*. As seen in chapter 17, reducibility decreases with the advance of reduction process. Thus, for direct reduction, reducibility in the final stages of reduction $(dR/dt)_{80}$ deserves greater attention.

The industrial reactors of direct reduction work with temperatures lower than those prevailing in the blast furnaces, reaching maximum values in the order of 1000 oC. To improve the kinetics of the reduction reactions, the reducing gases are usually obtained by reforming natural gas (CH4), being rich in hydrogen (H2). The reducing atmosphere, rich in H2 and CO, has a high reducing potential and accelerates the reduction process in solid state.

Generally, one can write the hematite reduction reaction in this process as:

$$Fe2O3 + (H2, CO) \rightarrow Fe^{o}_{(Metallic/solid)} + (H2O, CO2)$$

The production of the iron metal depends on the complete elimination of oxygen, combined with hematite. However, this reaction does not occur integrally, in the direct reduction, obtaining a product composed of:

$$Fe^{o}_{(metallic)} + FeO + Fe2O3 \text{ (traces)}$$

It is evident that the objective of the DR process is to produce " $Fe^{o}_{(metallic)}$ " phase, however, due to thermodynamic and chemical kinetic issues of the direct reduction process, the reduction reaction in the solid state is never completed, and there are these oxide residues. For the quality control of the DRI/HBI, the index "*Metallization Degree*" was created, which combined with the reduction degree, provides an idea of the reducibility of the pellets and the efficiency of the reduction reactor. Thus, the degree of metallization (M) of a reduced iron oxide is the fractional relationship between the percentage of metallic iron (Fe^{o}) present in the DRI/HBI and the percentage of total iron (Fet), present in this product. The expression of (M) could be written as:

$M = Fe^{o} / Fe_{t}$.

Writing in percentage, it results:

$$M = 100.Fe^{o} / Fet \quad . \qquad \text{Eq. 18.1}$$

Where:

- M – metallization degree of the reduced material in %;
- Fe^{o} – metallic iron content of the reduced iron oxide in %;
- Fet – total iron content of the reduced material in %.

The levels of (Fe^{o}) and (Fet) are determined in the laboratory, by the chemical analysis of the reduced material, while the residual FeO (Fe^{2+}) and Fe2O3 (Fe^{3+}) contents are also determined.

Metallization is the parameter of greater competitiveness of DRI/HBI, in relation to scrap. The steel productivity, electricity consumption and metal yield of the EAF are directly dependent on the metallization of the DRI/HBI.

And how to calculate the degree of reduction of the product of direct reduction?

In Chapter 17, the degree of reduction (R) of a reduced iron oxide was well characterized and calculated as a function of the initial oxygen weight (Oi) of that material.

Having the complete chemical analysis of the iron state in the DRI/HBI, it is possible, through mass balance and stoichiometry, to calculate the degree of reduction of the product of DR reactor. To this end, the data and hypotheses indicated in table 18.1 are considered.

Table 18.1 – Production of DRI/HBI from iron ore pellet with 100% Fe2O3.

FEED OF DR REACTOR →	DR REACTOR →	OUTPUT OF DR REACTOR
PELLET	DIRECT REDUCTION	DRI/HBI
Mass = X	Reduction by gases:	Mass = Y
% total Fe = f	CO, H2	% total Fe = f1
Oxide: 100% Fe2O3		% metallic Fe = fo
		% FeO = f_2 (residual)
		% Fe2O3 = f_3 (residual/traces)

From stoichiometry, it is known that FeO has 22% oxygen and 78% iron. In turn, Fe2O3 has 30% oxygen and 70% iron.

In Chapter 17, it was seen that the degree of reduction (R) of the pellets, obtained in the direct reduction process, is defined as the fractional relationship between the loss of oxygen mass of the pellets (Oi – Of) and the initial mass of oxygen of these same pellets (Oi).

Like this:

R = (Oi – Of)/Oi

Expressed as a percentage, it comes:

$$R = 100.\ (Oi - Of)/Oi\ \% \qquad \text{Eq. 18.2}$$

Where:

R –reduction degree of the pellets, after passage through the reactor, in %;

Oi – initial mass of oxygen of the pellets;

Of – final mass of oxygen in DRI/HBI (residual);

(Oi – Of) - represents the loss of oxygen mass by the reduction process.

Considering the data in table 18.1, the stoichiometry and the mass balance, the development of the calculation of the of reduction degree of the reduced pellets can be done, from the chemical analysis of the reduced material, DRI/HBI.

(a) Iron mass balance of the process

The iron mass of the pellet = the iron mass of the DRI/HBI, or

$X.f = Y.f1$ Eq. 18.3

b) Initial oxygen mass (Oi) of the pellets

Oi = mass of oxygen contained in Fe_2O_3 of the pellet.

In hematite, the mass ratio of oxygen to iron is given by 48/112.

Thus, $Oi/(X.f)/100 = 48/112$, or $100.Oi/(X.f) = 48/112$.

Simplifying, it results:

$X.f = 233.Oi$

Substituting X.f into the equation Eq.18.3 it comes:

$233.Oi = Y.f1$, or

$$Oi = (0.43).\ (Y.f1)/100 \qquad \text{Eq. 18.4}$$

Thus, the initial mass of oxygen (Oi) of the pellets is obtained, as a function of the weight of the reduced material (Y) and its iron content (f1).

c) Residual oxygen mass (Of) in the reduced material - DRI/HBI

The residual/final oxygen mass (Of) = oxygen mass of FeO + oxygen mass of Fe_2O_3, both contained in the reduced material, i.e., in the DRI/HBI.

Then, it can be written that:

$O_f = Y.(f_2/100).(0.22) + Y.(f_3/100).(0.30)$

Simplifying, it follows that the final oxygen mass (Of) is equal to:

$$Of = Y.(0.22).f2 + (0.3).f3)/100 \qquad \text{Eq. 18.5}$$

d) Loss of oxygen mass in the process of direct reduction (Oi – Of)

Taking the expressions of the equations Eq. 18.4 and 18.5, one can calculate the loss of oxygen mass by the expression:

$(Oi - Of) = [Y.f1.(0.43)/100] - [Y.[(0.22).f2 + (0.3).f3]/100]$, or

$$(Oi - Of) = Y.[(0.43).f1 - (0.22).f2 - (0.3).f3]/100 \qquad \text{Eq. 18.6}$$

e) *Reduction degree (R) of pellets in the Direct Reduction process*

Taking the expressions of Eq. 18.6 and 18.4 and taking into the equation Eq. 18.2, the general and final expression is obtained for the calculation of the reduction degree of pellets, ores or agglomerates, in function of the analysis of the reduced product.

Thus, it results:

$R = 1\ 00. (Oi - Of)/Oi = 100. \{ Y.[(0.43).f1 - (0.22).f2 - (0.30).f3]/100\} / [(0.43).(Y.f1)/100]$

Simplifying, it comes:

$R = 100. [(0.43).f1 - (0.22).f2 - (0.3).f3] / [(0.43).f1]$, or finally:

$$\mathbf{R = 233. [(0.43).f1 - (0.22).f2 - (0.3).f3] / f1\ \%}\ . \qquad \text{Eq.18.7}$$

Where:

R – Reduction degree of the pellets, after passage through the DR reactor in %;

f1 – total iron content of the reduced material, DRI/HBI in %;

f2 – FeO content of the reduced material in %;

f3 – Fe2O3 content of the reduced material in %, normally negligible.

Thus, the general expression of the calculation of the reduction degree of any reduced iron ore and/or agglomerate, submitted to any reduction process, was obtained, by the chemical analysis of the final product:

$$R = f(f1, f2, f3).$$

It is evident that if the initial matrix of iron ore and/or agglomerate is composed of minerals other than hematite, such as magnetite, limonite, etc., the expression for the calculation of R must be adjusted with respect to the initial oxygen mass (simple stoichiometric adjustment).

As this development was aimed at the case of iron ore pellets, even starting from 100% magnetic ore, the final ferrous matrix of the pellet structure is basically composed of 100% Fe2O3 (the FeO content is negligible).

If it is desired to develop the expression of the reduction degree (R), considering the content of metallic iron (Fe^{o}), Fe^{++} and Fe^{+++}, it is enough to make some stoichiometric adjustments in the expression of the equation Eq. 18.7. It is important to mention that the iron balance of the final reduced material is given by:

$$\% \ \mathbf{Fe1} = \% \ \mathbf{Fe^{o}}_{\mathbf{(metallic)}} + \% \ \mathbf{Fe^{++}} + \% \ \mathbf{Fe^{+++}} \ .$$

The requirements of operators of direct reduction plants are very high, regarding the reducibility of iron ore pellets, or, indirectly, in relation to the susceptibility to generate metallic iron in the reactors. With the evolution of this technology and the quality of the pellets, along the time, it was possible to significantly increase the productivity of the furnaces and improve the reduction and metallization degrees of the DRI/HBI. Metallization grade > 94-95% is becoming a reference standard in world-class direct reduction plants.

CHAPTER 19

SOME ASPECTS RELATED TO THE USE OF IRON ORE PELLETS IN THE BURDENS OF REDUCTION REACTORS

The steel industry has evolved significantly since the early 1970s. In the area of primary iron production, direct reduction began its industrial escalation, and the Japanese engineers dissected the blast furnace, questioning the current concepts and revolutionizing operational practices. With the evolution of these reduction technologies, the preparation of the metal burden for feeding of the reactors has taken on a relevant role, with significant advances in the areas of iron ore processing and agglomeration (sintering and pelletizing). In this regard, pelletizing accompanied the technological development, consolidating the pellets as a metallic burden of the reduction reactors.

With the development of the concept of elaborated burden for feeding the blast furnaces with consequent increase in productivity, *the lump ore* was losing space to the sinter and pellets. In this aspect, *the pellets* were configured as the main component of the metallic burden of the American and Canadian blast furnaces (*American school*), and the sinter became the basis for the Japanese blast furnaces *(Japanese school)*, a practice that spread throughout the world, including Brazil.

As for the direct reduction, the pellets have become almost the absolute component of the reactor burden. The scarcity of lump ore and the strict specifications of chemical, physical and metallurgical quality for the metallic burden were responsible for the dominance of the pellets in direct reduction.

There are many studies and technical papers written and published on the use of iron ore pellet in the steel industry. The advantages and disadvantages of its use were widely emphasized. It is not the objective of this book to recover them, since the corresponding technical literature is available in the electronic media.

It is important to mention that the pellets make a great contribution to the preservation of the environment, especially as burdens of the blast furnaces, because this material increases productivity and reduces emissions to the atmosphere.

To illustrate this subject of the use of pellets in the steel industry, four technical papers were separated, prepared, and published by the author of this book, solo and together with other co-authors, which are presented in Annexes I, II, III and IV.

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ANNEX I

PAPER A

INCREASING TECHNICAL APPEAL OF PELLETS USE IN BLAST FURNACE

INCREASING TECHNICAL APPEAL OF PELLETS USE IN BLAST FURNACE

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ABSTRACT

Pellets are unquestionably the most engineered Blast Furnace ferrous burden available. This allows the pellet maker to design pellets to meet different requirements with respect to medium and high temperature properties. With the scarceness in availability as well as in quality of lump ores, the interest in pellets when burdening the furnace is increasing.

The pellet-indurating furnace has self-regenerative gas flow, which leads to higher energy efficiency, thus resulting in lower consumption of fossil fuels. The consequence is the reduction in emissions of greenhouse gases. Furthermore, as the mounting environmental pressure increases the difficulty of permitting new sintering plants, the demand for pellets will be favored.

This paper presents the trends for the evolution of production and consumption of pellets.

1. INTRODUCTION

The mainstay of the integrated steel industry is the production of primary iron units in the Blast Furnace. The majority of the integrated plants throughout the world operate with a significant fraction of sinter in the burden, usually higher than 70%. However, the sinter plant, along with the coke ovens, has been under pressure to attain higher levels of sustainability. Nonetheless, the sinter plant is the destination of several residues generated in the integrated plant, what is of fundamental importance in residue recycling policy. This is due mostly to the fact that they are generally located in the same site the ironmaking, steelmaking and mechanical transformation facilities are. Pellet plants, on the other hand are usually located at or near iron ore mines, and even though the addition of steelmaking residues is possible, its logistics costs hinder any real possibility of large-scale use.

Although both processes are used to agglomerate fine ores using liquid formation, its fundamentals differ significantly. The fraction of liquid in the sinter making operation is above 20%, whilst for pellet making, this figure tends to be lower than 10%. Except for the fuel used for ignition, the sinter plant runs on coke breeze or other solid fuels, while the pellet plant runs on a myriad of fuels. The traveling grate plants used by CVRD to pelletize hematite ores use a combination of fuel oil, natural gas, and coal.

Furthermore, not all sinter plants have gas recirculation systems, unlike all pelletizing furnaces, regardless of if they are of the traveling grate or grate-kiln types. This has a great impact on the energy consumption and therefore, carbon emission. However, the impact of the agglomeration routes used extends farther down the steel production chain. Sinter and pellets behave differently in the Blast Furnace in terms of reducibility, fluid dynamics, softening and melting properties and amount of slag formers, which impacts the furnace’s overall slag rate. Therefore, the impact of both routes is discussed in comparison to their performance in both the agglomeration plant as well as the Blast Furnace.

2. BLAST FURNACE CURRENT OPERATING PRACTICES

The strategy to define metallic burden for the blast furnace operation is a complex process and admits various solutions. Several factors must be taken into consideration based on the location of the plant, operational goals, and technological aspects. Moreover, different philosophical thinking leads to distinct forms to face the problem in the existing operational practices in the world. The extent of pellets participation in the metallic burden of the blast furnace is a major differentiation factor of operational practices around the world. This is due to various aspects, notably its acquisition cost and the fact that in most cases it is produced outside plant gates. In this section of the paper a preliminary analysis of different operational practices based on their metallic burden is presented.

2.1. BURDEN COST MINIMIZATION

One well-defined strategy for defining the metallic burden is the minimization of the costs of raw material procurement. This strategy is based on the fact that the pig iron production cost is mainly affected by raw material costs. Transportation and handling costs are the predominant aspect that ironmakers consider, thus the main sourcing options are close to their operations. As the worldwide availability of lump ore is getting restricted, this philosophy results in a stronger technological development of the sintering process, due to a higher flexibility for the selection of raw materials. Under this strategy, the dependence of high-cost sinter feed (imported) is minimized, and therefore its participation is reduced to the minimum necessary to achieve the desired quality and productivity of the locally produced sinter as well as of Blast Furnace slag and hot metal. The participation of pellets in the Blast Furnace burden is reduced following the same logic, as pellets have the highest acquisition cost.

As the main driver here is the sinter production optimization, the technological development aimed at obtaining for sinters trends similar to those obtained with pellets in the Blast Furnace operation. For example: reduction of the SiO_2 content to decrease slag rate and to improve high temperatures properties, reduction of the MgO contents, mixture segregation technologies to improve productivity, segregation of Al_2O_3 in nucleating particles, increase of magnetite participation, among others.

2.2. ENHANCED PRODUCTIVITY

If reasonable options of iron ore source close to the ironmaking site are unavailable and/or with low quality, another operational philosophy would prevail, based on the use of pellets as a burden complement. In this case the distance to the producing centers of pellet is not as critical as in the previous case and the comparison of the performance of sinters and pellets in the Blast Furnace, also considering their different costs of acquisition and production can result in an increased participation of pellets in the burden. An additional aspect, which is gaining momentum, is stricter environment restrictions for sinter production. In this case, the strategy of choice to increase the hot metal production, when needed, is a higher participation of pellets. Also, the trend of concentrating the production of hot metal in large, efficient, Blast Furnaces is another factor that contributes to an increased participation of pellets in the burden. Despite of higher acquisition costs, the gain in productivity, lower capital expenditures (for the ironmaker when purchasing external pellets) and more efficient use of the Blast Furnace can, in some circumstances, compensate it and the final steel can be more competitive. Of course, the extent of pellets use in this scenario depends on the market activity, the costs of raw materials and the demand for steel.

A variant of this philosophy is the existence of an abundant low quality ore source nearby. The participation of high-quality pellets in the metallic burden allows maximizing the use of poor local material in the sintering machines. Furthermore, it can be an option vis-à-vis the investment to expand the sintering capacity. These high-quality pellets can be purchased in the transoceanic market and/or produced at the site, with an appropriate mix of high-quality imported ore with poor local concentrates that can lead to a more advantageous final solution.

2.3. ALL-PELLET OPERATION

This operational philosophy is based on the intensive use of pellets in the Blast Furnace, often as the sole type of metallic burden, and seems to be reasonable in situations where the appeal for sinter production is minimum, such as tough environment restrictions coupled with outdated industrial sintering facilities and/or intensive capital costs for sintering projects. In this scenario, most sintering plants are eventually shut down but for those that are operated inefficiently to recycle internal reverts. Generally, these Blast Furnaces have high productivity and low fuel rate (together with a flexible burden distribution system), generating an advantageous global economic condition associated with the massive use of pellets in the burden.

3. THE CONCEPT OF VALUE IN USE

To gain competitiveness, the steelmaker tends to focus on its core business, which is generally related with final steel products and their applications. This trend is characterized by the relationship between steel plants and customers, thus requiring the steel producer to enter into the customer's value chain, adding services, solutions, and technology. Therefore, the iron ore supplier must be ready to play the same role downstream, by being able to offer ferrous burden solutions to the steel producers, based on thorough knowledge of the characteristics of its ores and intense cooperation with the raw materials preparation and ironmaking sections of the plant. Figure 1 shows schematically the resulting approach, commonly referred to as geo-metallurgy. This knowledge allows the supplier to fully understand the underlying phenomena that correlate iron ore genesis (related to iron ore deposit formation and enrichment processes) with its performance in mining, beneficiation, and ironmaking processes.

Value in use can be defined as the value proposition that a defined metallic burden can represent to a specific customer. Figure 2 describes the value in use concept. To define the proper metallic burden solution, it is necessary to first understand completely the customer's reality and goals. Then, the operational data must be organized, treated, and analyzed.

The generation of knowledge and creation of a specific strategy will vary according to the goals of the partnership. Nonetheless, they are achieved through simulations, both numerical and physical, which enable us to propose a metallic burden that would fulfill the customer's needs. Continuous efforts are done at CVRD to improve the estimation of the value in use of the metallic burden considering the customer's reality. CVRD has been investing expressively in people, pilot plants, characterization laboratories and numerical simulation to understand the geo-metallurgical relationships and their application towards generating value to the customer. Microstructural, chemical, and physical characterization, sinter pot, pellet pot grate as well as softening and melting tests, along with different simulation tools are available for this task. Usually, a linear optimization tool coupled with a neural network for quality and performance estimation is used to model the sintering plant, whose results are applied to a Computer Fluid Dynamics model of the Blast Furnace.

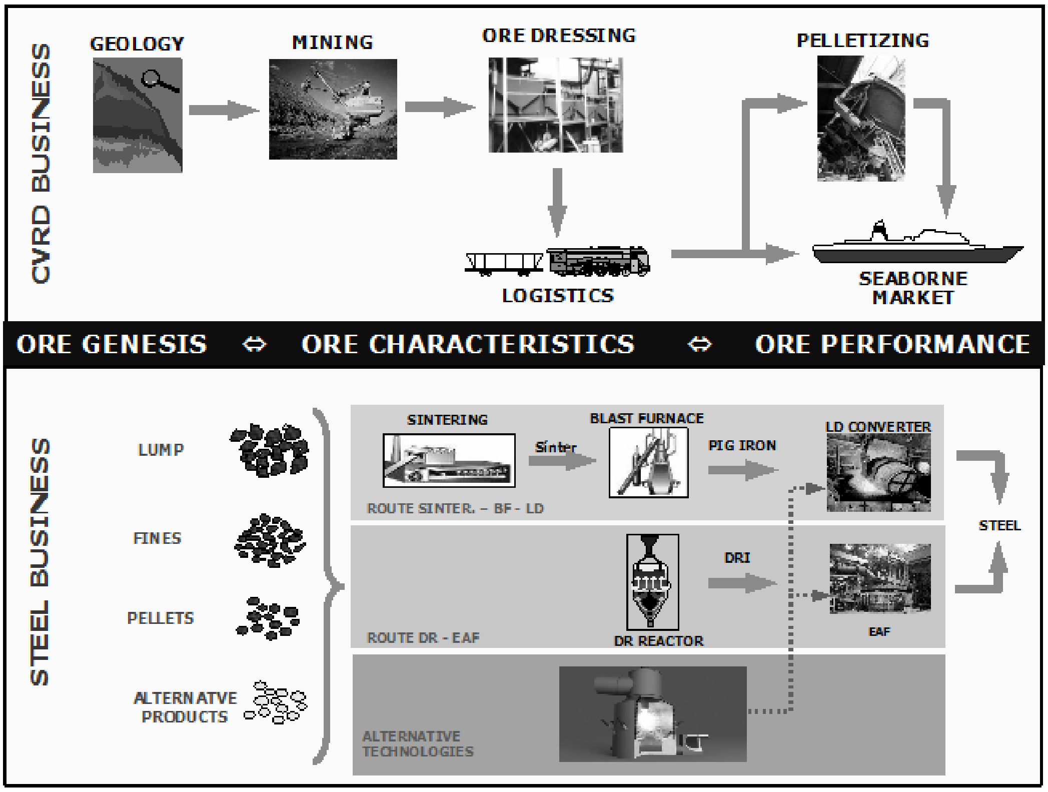

Figure 1. CVRD iron ore research emphasis – Mining through Steelmaking (BOF).

4. IMPACT ON PRODUCTIVITY OF HIGH PELLET OPERATION

The use of higher percentages of pellets in the metallic burden leads to important modifications in the iron and steelmaking processes. Obviously, the first impact is the increase in the cost of raw materials. However, this would not necessarily result in a higher hot metal and rolled steel costs. One first aspect that must be considered is the option to produce the majority of the metallic burden needed or to acquire at significant part of it elsewhere. The production of internal sinter can guarantee the ironmaker the control of metallic burden production, with its costs and quality aligned with their philosophy. To support the increase in the participation of pellets in ironmaking worldwide, the Blast Furnace process should be analyzed.

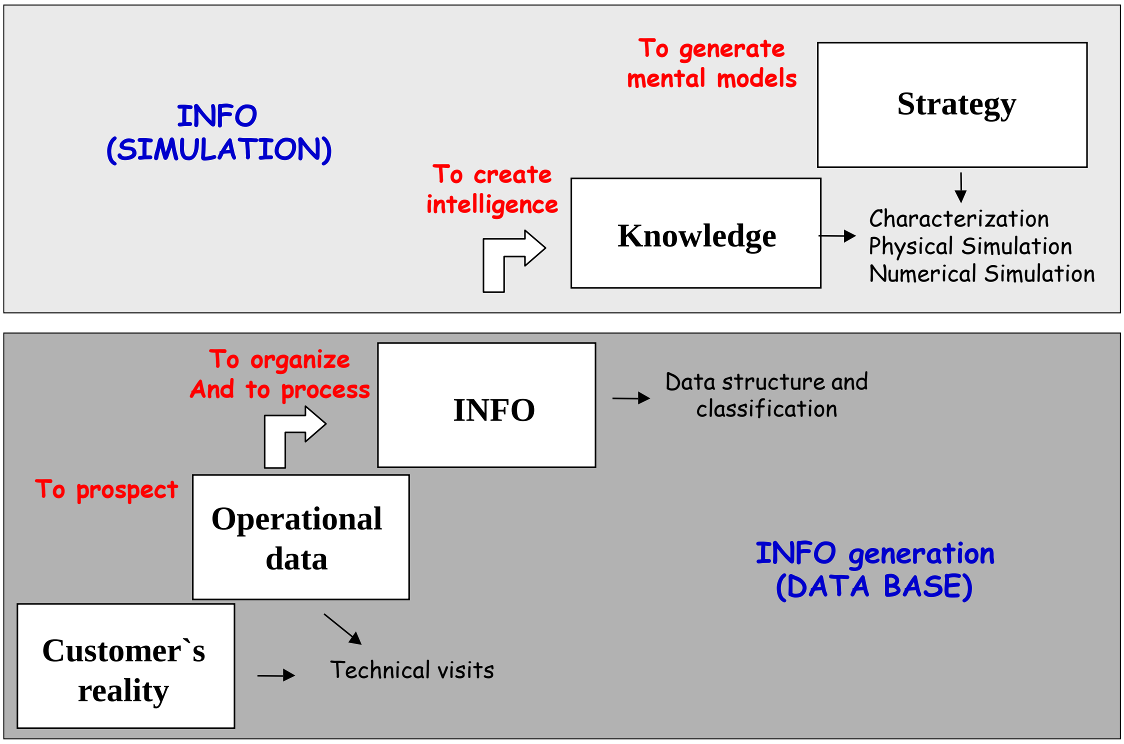

Figure 2. The concept of value in use.

Productivity is an important figure that can be used to compare Blast Furnaces, as it is influenced by nearly all aspects of the process, from furnace dimensions, ancillary equipment, blast parameters and burdening. However, comparing productivities across different cultures is difficult, as local conditions will dictate the operational philosophy, as described in section 2.

As an overall rule, the increase in the amount of pellets charged should increase productivity. This link between productivity and pellet usage is clear to the ironmaker, and is due to several factors including slag volume, fluid dynamics of the granular zone and low temperature reducibility. The slag volume is reduced because of the low gangue content of pellets, while the fluid dynamics is improved due to its more regular shape. The low temperature reducibility is higher than other burden materials, because of the combination of small hematite grain size and high porosity of pellets. The reduction of the slag volume will reduce the fuel requirements to melt the slag. It will also generate a suitable condition in the raceway to increase the pulverized coal injection rate by reducing the formation of “bird’s nest” that limits gas penetration in the center of the furnace.

The improvement of the furnace’s granular zone fluid dynamics will reduce the pressure drop across that zone. Also, coupled with the increase in the low temperature reducibility will increase gas usage, thus lowering the carbon requirements in the thermal reserve zone and beyond. Another interesting point is that the furnace fuel rate and consequently, sulfur input, decrease with higher utilization of pellets, which also affects the value in use of this material, both directly (low raw material costs) and indirectly (hot metal quality and De-S costs). Top gas credits can reinforce this rationale, specifically in cases where the top gas calorific value is used to generate electricity within or outside the plant gates.

The increase of MgO content in the pellets allows the operation of a sintering machine with residual MgO content, which should increase productivity significantly without jeopardizing the desulphurization in the Blast Furnace. Although pellets usually have lower softening and melting temperatures, the coupled design of pellet and sinter will generate a mixed burden with optimal performance in terms of softening and melting properties.

It will improve the permeability and consequently the productivity of the Blast Furnace. In terms of chemical composition, pellets can impact positively the steelmaking operations since it is possible to produce lower phosphorous material that won’t generate extra costs in De-P units. Another possibility is to use, with higher pellets participation, poorer materials as a whole in the furnace, such as coals with high ash content and lump ores with high P and Al2O3. The quality trend of lump ores indicates a general quality loss in the near future, as shown in table 1.

Table 1 – Quality forecast for selected lump ores.

CHARACTERISTICS		% Fe	% SiO_2	% Al_2O_3	% P	% PF
Lump A	Actual	65.5	3.35	1.45	0.050	1.20
	Future	65.3	3.70			
Lump B	Actual	67.0	1.30	1.30	0.050	1.15
	Future	64.6	2.35	1.70	0.045	3.20
Lump C	Actual	64.3	3.00	1.40	0.065	3.00
	Future	63.7	3.45	1.65	0.075	3.20
Lump D	Actual	62.1	3.65	1.40	0.050	5.55
	Future	61.4	4.20	1.60	0.060	5.60

Regardless of these factors, in countries with low capital costs it seems reasonable that large reactors with lower productivities are economical, and thus it is natural that pellets will have a limited participation. Conversely, in a scenario of high capital costs and favorable market, it is natural that maximum productivity should be obtained from Blast Furnaces, thereby diluting the capital expenditure. Also, if the steelmaker's downstream facilities have surplus capacity, and there is demand for high grade-high return steel, another favorable scenario to the productivity arises. In the latter cases, the increased use of pellets is possibly a valuable alternative. The results of CFD and linear optimization simulation results for different burden mixes in a large Blast Furnace (4,800 m^3 Working Volume) are presented in figure 3.

Similar operational conditions were targeted such as slag chemistry, oxygen enrichment and PCI rates. As shown in this figure, the Coke-Rate and Slag-Rate reduced, while Productivity increased alongside the participation of pellets in the burden. Among other results, a reduction of the CO/CO$_2$ ratio in the top gas, hot metal silicon and sulfur contents were observed.

As the pellet fraction in the burden increased, the permeability of the granular zone increased as well, which allowed for higher blast volumes and therefore, the observed productivity gain. Another point not considered in this case was the ability to inject more pulverized coal. Necessary to say that the results exposed are based in fluid dynamics and heat and mass transfer simulations considering a porous bed.

Quantitative variations from these results can happen in real Blast Furnaces depending on operational philosophy and other local factors. The key factors here are to present the technical and qualitative general trends, not considering the economics of each case.

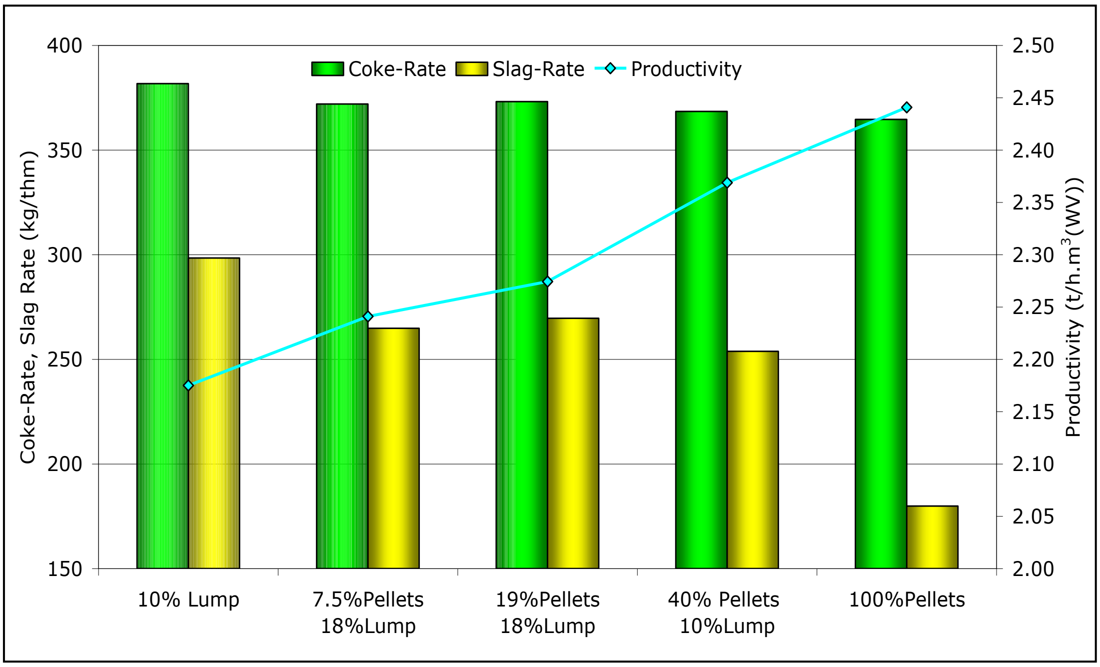

Figure 3 – Variation of coke-rate, slag-rate, and BF productivity in relation to different burden mixes (sinter + pellets + lump ore).

5. ENVIRONMENTAL AND ECONOMICAL CONSTRAINTS

The design differences between the sintering and pelletizing plants results in different emission patterns for both. Traveling grate and Grate-Kiln pellet plants emit less dioxins and furans, mostly due to the temperature profile and gas recirculation scheme. The emissions of SO_X are also lower, mostly due to higher fuel efficiency as well as the ability of using lower sulfur fuels such as Natural Gas. For conventional sintering plants, an average of 50 kg of coke breeze and/or anthracite is used as fuel for each metric ton of sinter produced, plus fuel for the starter oven, usually 2.5 Nm^3 of COG gas/t sinter. The high SiO_2 content of sinters, necessary to attain the desired mechanical properties, requires a compatible content of MgO and CaO.

Usually, limestone and dolomite are their main source, albeit dunite, serpentinite and olivine are also significant sources of MgO for the sintering process, with negligible CO_2 intensity. Under these considerations, the sinter process would emit around 225 kg of CO_2 for each metric ton of sinter. Of these, approximately 60 kg are due to the calcination of limestone or dolomite, which would have to be accomplished, at least partially, somewhere in the ironmaking process to achieve the desired slag composition.

For pellet plants, on the other hand, the observed fuel consumption would be lower. In grate-kiln plants, approximately 20 kg of coal and 10 Nm^3 of Natural Gas are consumed per metric ton of pellet produced. Traveling grate plants consume roughly 15 kg of coal and 15 Nm^3 of Natural Gas per metric ton of pellet produced. In either case, the overall CO_2 emission should be around 75 kg/t pellet produced, plus the emission due to the calcination of limestone and dolomite to correct the slag both in the pellet and in the Blast Furnace. For low silica self-fluxed pellets this would amount to 21 kg CO_2/t pellet produced. It is important to mention that in an all-pellet operation, such as the one presented in figure 3, an additional 36 kg of limestone and dolomite would have to be added, thus emitting and additional 28 kg CO_2/t pellet consumed. Furthermore, the production of one metric ton of pellets consumes 50 kWh of electrical energy, whilst for one metric ton of sinter, the electrical energy consumption is 35 kWh/t. Regardless of local variations, the CO_2 emissions of both processes are quite different. Notwithstanding any gains due to lower fuel rates in the Blast Furnace with increased use of pellets, as shown in section 3, the replacement of sinter by pellets would result in lower emissions along the entire chain. This effect is presented schematically in Figure 4 for plants operating under benchmark conditions with varying lump ore quantities. Also, this figure shows where existing steel plants across the world lie in this diagram when considering only their burden practice. Gains or losses due to slag volume, blast conditions, injection rates different from the benchmark are not considered in this diagram.

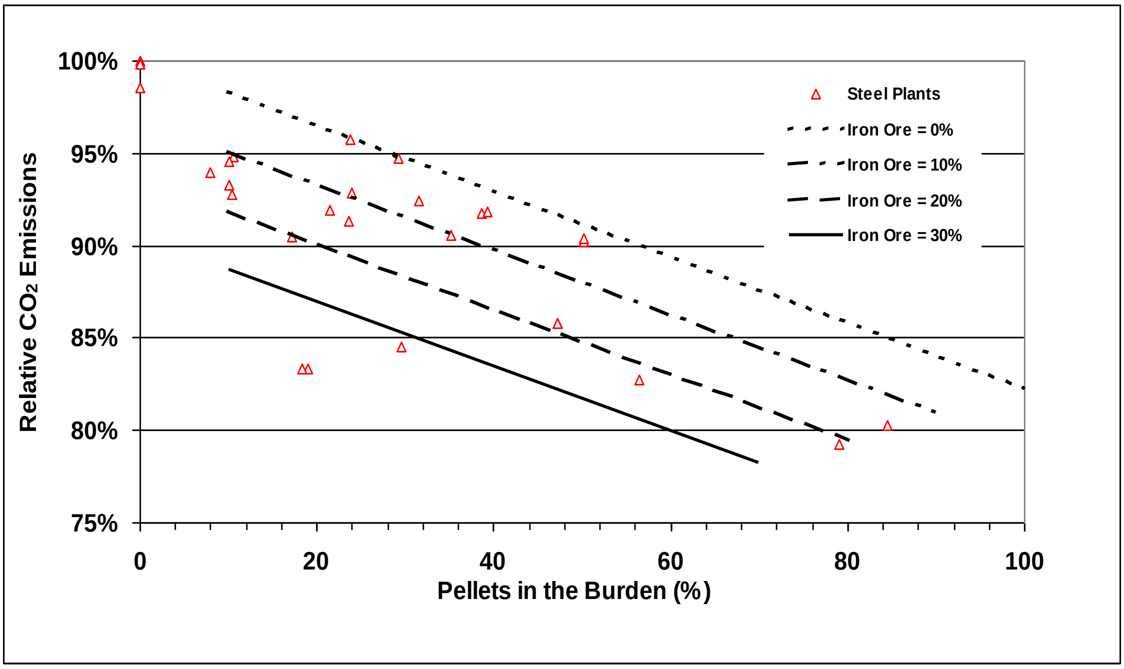

Figure 4. CO2 emission rate as a function of the amount of pellets in the burden. (Straight lines calculated for different lump ores of existing plants – open triangles)

5. CONCLUSION

The use of pellets in the upcoming years is likely to become more important in ironmaking , not only due to the decrease in lump ore quality or availability, but mostly as means of increasing the productivity and gas utilization in the Blast Furnace. Furthermore, higher pellet participation in the burden of blast furnace will contribute to minimize CO2 emissions. The proper ratio of pellets, sinter and lump ore will depend on various factors, such as: Blast Furnace operation philosophy, price and market conditions, steel plant location, environmental issues, among others. In light of this scenario , CVRD is positioning itself as an overall burden solution provider and able to supply the market with competitive and high-quality iron ores and agglomerates.

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ANNEX II

PAPER B

COMPARISON OF SINTER AND PELLET USAGE IN AN INTEGRATED STEEL PLANT

COMPARISON OF SINTER AND PELLET USAGE IN AN INTEGRATED STEEL PLANT [1]

Jose Murilo Mourão [2]

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Abstract

Global iron ore production has grown dramatically in recent years to meet increasing world steel demand, especially in Asia. High grade lump ore resources are being depleted and a greater amount of fine concentrate/pellet feed will enter into production as lower grade deposits are mined. Integrated steel plants need to make convenient use of the available iron ore resources to optimize operation and the cost of steel. The advantages and disadvantages of using greater amounts of iron ore concentrate are discussed, focusing on the production and use of fired pellets in the blast furnace. Hot metal production using sinter and pellets in the blast furnace is compared, considering aspects like; blast furnace productivity, environmental performance, solid waste management, slag-coke rates, and the steel plant energy balance.

Key Words: Iron Ore; Pellet Feed; Sintering; Pelletizing; Blast Furnace.

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[1] *Technical contribution to the 43rd Ironmaking and Raw Materials Seminar and the 14th Brazilian Iron Ore Symposium, September 1st to 4th, 2013, Belo Horizonte, MG, Brazil*

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1 – INTRODUCTION

In recent years, fine iron ore concentrate production, also referred to as pellet feed, increased as the availability of new lump ore and high-quality sinter fines declined during a period of rapidly expanding demand driven by the Asian steel industries. As a result, iron ore miners developed lower grade resources that must be finely ground and processed to increase their Fe content to the levels required by the marketplace. This trend will continue as lower grade resources must be exploited to meet the continuing growth in global steel production.

Most pellet feed will be sold in the seaborne trade rather than being captive to a dedicated steel plant. A fraction of these finely ground concentrates has been and will continue to be blended with sinter feed and processed on sintering strands. The application of concentrate in sintering is ultimately limited as permeability and quality limitations are reached, hence global pellet production will inevitably increase to consume the concentrate that will enter the marketplace. Steelmakers will need to increase the pellet consumption in their blast furnace burdens, or in the extreme case switch to all-pellet blast furnace operation.

Hatch compared the merits of using both pellets and sinter as the main constituents of the blast furnace burden, focusing on blast furnace productivity, environmental performance, solid waste management, and the steel plant energy balance. For this exercise, only sinter fines and fired pellets available in the seaborne market were considered. Lump ores were not used in the burden mixes studied due to the expected decline in lump availability in the coming years.

2 – BACKGROUND

2.1 – Trends in the Seaborne Iron Ore Market

The global seaborne iron ore market including sinter feed, lump ore, pellets and pellet feed has rapidly increased over the last decade. While sinter feed sales dominate the global seaborne trade, pellet feed and fired pellet sales have grown at a faster rate. Hatch anticipates that fired pellet and pellet feed sales will further accelerate at the expense of sinter feed due to declining availability of high-quality sinter fine resources. In Figures 1 and 2, the growth of the seaborne trade and Hatch’s projection of the global seaborne iron ore trade to 2040 are presented.

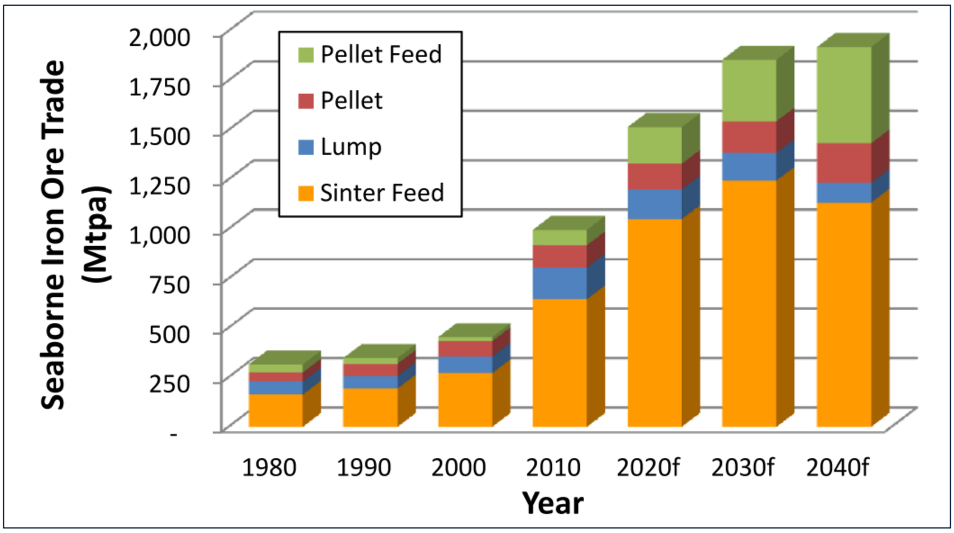

Figure 1. Historic Growth and the Future Projection of the Global Seaborne Iron Ore Trade. (1)

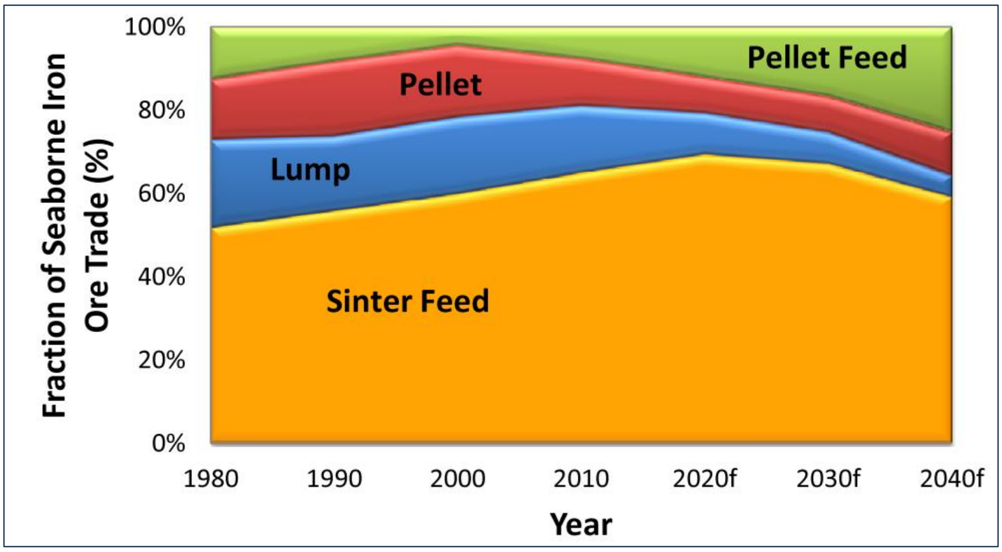

Figure 2. Projected Product Sales Distribution for the Global Seaborne Iron Ore Trade. (1)

The increasing importance of fired pellets and pellet feed in the global seaborne trade will push steel producers to increase their utilization in the blast furnace. Some of the additional pellet feed will continue to be added to the sintering mix despite productivity loss and the inherent environmental disadvantages of the sintering process. The remainder will have to be pelletized and added to the blast furnace burden to replace declining sinter and lump ore supplies.

Hatch has compared the merits of using sinter and pellets in the blast furnace to help steel producers evaluate the impact of increased pellet usage in the blast furnace burden.

2.2 – Comparing Sinter and Pellets

Sinter and pellets are agglomerated forms of iron ore, both suitable for use as blast furnace burden materials. The principal difference between sinter and pellets arises from the type of raw materials used in their preparation and the nature of the sinter and pelletizing agglomeration processes.

Sinter is a clinker-type iron bearing material that is produced when a mixture of iron ore fines known as sinter feed, finely ground fluxes, carbon (coke breeze or anthracite) and various recycled iron bearing materials are uniformly fired along a continuous traveling grate reactor. Fuel in the sinter mix is ignited and generates temperatures high enough for the fine particles to fuse together into a porous clinker material which is subsequently crushed and sized after cooling to room temperature. The resulting sinter is suitable for use as a blast furnace burden material but is not sufficiently strong to withstand long distance transportation. As a result, sintering plants are normally located in close vicinity to the blast furnace, usually within an integrated steel works.

Fired pellets are hard iron bearing balls that are produced to a specific size range by forming iron concentrate into unbaked green pellets and then heat hardened these green pellets in a dedicated induration furnace. The main feed materials are finely ground iron ore concentrate, finely ground fluxes and, in the case of hematite ores, finely ground carbon (coke breeze or anthracite). Magnetite ores do not require carbon additions as the magnetite oxidation in the induration furnace provides enough heat to sustain the process. The mixed materials are formed into small 8-16 mm diameter balls through the action of rotating drums or discs at a controlled moisture and with a binder such as bentonite. The green balls are then fired at controlled temperatures in an induration furnace which can be one of two types: a single straight grate induration furnace or a train of three reactors consisting of a travelling grate, rotary kiln and cooler, known as the grate-kiln process. The high temperatures produced in either process heat harden the green pellets, producing fired pellets which are strong enough to be used as blast furnace burden materials. Due to their higher physical resistance compared to sinter, pellets can survive long distance transportation and are thus an internationally traded commodity. Depending on their final user, pellets are often categorized between blast furnace (BF) pellets and direct reduction (DR) pellets, the latter having a higher Fe and lower gangue content consistent with the requirements of the direct reduction process. The focus of this paper will only be on BF pellets.

Sinter and BF pellets differ significantly in both their chemical and physical properties and their performance inside the blast furnace is dependent on both. This paper will focus only on the chemical properties, as those have a greater impact on the variables analyzed, blast furnace productivity, coke rate and slag rate. The typical chemical properties of sinter and BF pellets, the latter corresponding to those available in the seaborne market are listed in Table 1.

Table 1. Typical chemical properties of sinter and BF pellets. (1,2)

ITEM	SINTER	BF PELLETS
% Fe	55.0 - 58.0	62.0 - 66.0
% SiO2	5.0 - 6.0	2.0 - 5.0
% Al2O3	1.0 - 1.5	0.4 – 1.0
% CaO	9.0 - 11.0	1.0 - 4.5
% MgO	1.4 - 2.0	0.2 – 1.3
% (SiO2 + Al2O3)	6.0 - 7.5	2.4 - 6.0
% (SiO2 + Al2O3 + CaO + Mg0)	16.4 - 20.5	3.6 - 11.8
B2 =CaO/SiO_2	> 1.7	0.8 - 1.2

From table 1, it is evident that the main differences in the chemical properties between sinter and BF pellets are the total iron (Fe) content, the total acid gangue (SiO_2 + Al_2O_3) content and the binary basicity (CaO/SiO_2). The following subsections discuss the impacts of these parameters on blast furnace performance.

2.2.1 – Iron content

The typical iron (Fe) content of sinter is around 55-58%, while BF pellets are normally 62-66% Fe. BF pellets have a higher Fe content as they are manufactured from low-grade ores that are finely ground prior to undergoing mineral beneficiation treatments. During beneficiation, gangue materials are removed through a variety of mineral processing methods, such as gravimetric separation, magnetic separation and froth flotation. Iron is concentrated to a high degree as these mineral processing techniques efficiently eliminate gangue materials. By using clay binders, the pelletizing process can operate at lower basicity compared to the sintering process, requiring much lower flux additions as a result. Flux additions have a net diluting effect on the Fe content, so it follows that the Fe content of pellets is less diluted than that of sinter.

The higher Fe content of pellets increases blast furnace productivity as more iron units are charged to the blast furnace per unit ton of burden material. Shipping costs are reduced as more iron units and less undesirable gangue are shipped to the final blast furnace user.

2.2.2 – Total acid gangue content

The total acid gangue content, defined as $SiO_2 + Al_2O_3$, is significantly lower in pellets compared to sinter as pellets are manufactured from low-grade ores that underwent a significant degree of beneficiation to increase their Fe content as described in the previous sub-sections.

The main advantage of lower gangue pellets to the blast furnace operation is a lower slag rate. As blast furnace slag is mainly composed of the gangue materials present in the ferrous burden, the ash content of coke, and the added fluxes it follows that utilizing a lower acid gangue material will produce less slag per ton of burden or hot metal. A lower slag rates directly translate into lower coke rates, as less thermal energy is required to form and melt the slag. In addition to lower coke rates, another direct advantage of a lower slag rate is that a smaller volume of by-product slag must be subsequently sold or disposed.

2.2.3 – Binary Basicity

The third difference in the chemistries of BF pellets and sinter is shown by the binary basicity (B2) defined as the ratio of CaO/SiO_2. While for BF pellets the B2 ratio is typically around 0.8-1.1, for sinter this ratio is commonly greater than 1.7. Such a high basicity is required to improve sinter properties and achieve enough strength to withstand materials handling operations and for good performance within the blast furnace itself. Figure 3 below shows the dependence between sinter strength, defined as fraction > 10mm after tumbler test, and sinter basicity.

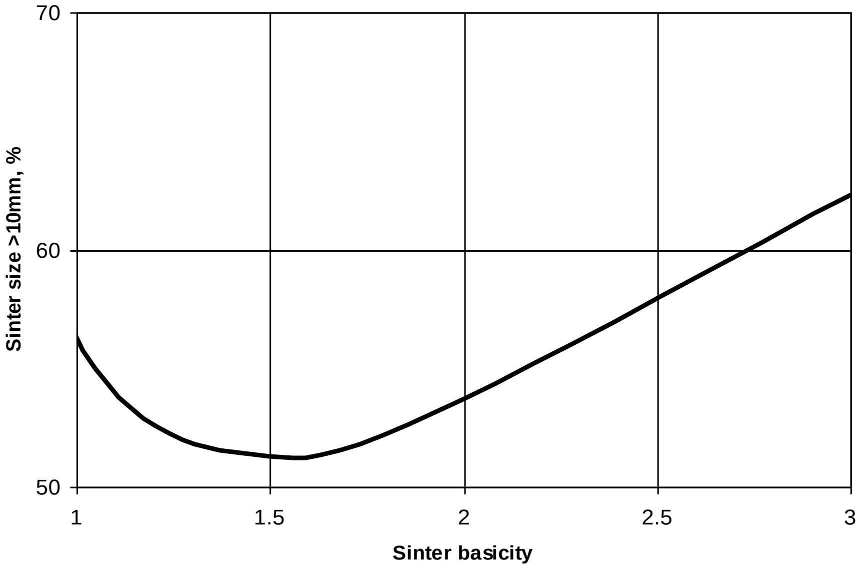

Figure 3. Relationship between sinter strength and sinter basicity. (1)

The higher basicity requirements of sinter compared to BF pellets negatively affects the operating costs of hot metal production in two ways. First, it requires a higher consumption of flux (limestone and/or dolomite) to achieve the target basicity. Secondly, it increases the slag rate, as the additional fluxes generate a higher slag volume.

3 – METHODOLOGIES AND DISCUSSION

For a simple comparison between sinter and pellet use in the blast furnace, six specific scenarios where considered. In each case, the burden ratios of pellet/sinter, as well as fuel injection type were varied. Mass and energy balances were then performed for each of the six scenarios to obtain Key Performance Indices (KPIs) that would enable the comparison of blast furnace performance for each case. Table 2 summarizes the cases studied:

Table 2. Fixed Blast Furnace parameters for the different scenarios

Parameter	Scenario 1	Scenario 1A	Scenario 2	Scenario 3	Scenario 3A	Scenario 4
Sinter %	65	65	0	65	65	0
Pellets %	35 (AF1)	35 (AF2)	100 (AF3)	35 (AF1)	35 (AF2)	100 (AF3)
BF Fuel Injection	PCI	PCI	PCI	NG	NG	NG

The chemical properties of the raw materials used in the six calculated scenarios are shown in Tables 3 and 4. Three different pellet types, consistent with grades widely available in the market were selected to reflect various BF operating scenarios:

- Pellet AF1 – Low silica fluxed pellets, used for Scenarios 1 and 3;
- Pellet AF2 – Acid pellets, used for Scenarios 1A and 3A;
- Pellet AF3 – Super fluxed pellets, used for scenarios 2 and 4.

Table 3. Iron bearing raw material properties.

Material	% Fe	% SiO_2	% Al_2O_3	% CaO	% MgO	CaO/SiO_2
SINTER	57.7	5.0	1.5	9.0	1.1	1.8
Pellet - AF1	66.2	2.2	0.6	1.8	0.2	0.8
Pellet - AF2	64.8	5.0	0.6	1.2	0.2	0.2
Pellet - AF3	63.8	3.0	0.6	3.6	1.1	1.2

Table 4. Carbon bearing raw material properties.

Material	% Ash	% Volatile Matter	% Carbon	Heat Value (MJ/kg)
Coke	13	0.5	86.5	32
PCI Coal	13	22.5	64.5	30
Natural Gas	-	-	73.7	40

The blast furnace charge rates of the ferrous burden, fluxes and fuels used for each of the six scenarios are summarized in Tables 5 and 6.

Table 5. BF burden composition

Characteristic	Unit	Scenario 1	Scenario 1A	Scenario 2	Scenario 3	Scenario 3A	Scenario 4
SINTER	kg/THM	1,020	1,029	-	1,021	1,029	-
PELLETS	kg/THM	549	554	1,494	550	554	1,494
Limestone	kg/THM	-	17	30	-	4	16
Quartz	kg/THM	11	-	-	17	-	-

Table 6. Fuel rate comparison.

Characteristic	Unit	Scenario 1	Scenario 1A	Scenario 2	Scenario 3	Scenario 3A	Scenario 4
Coke Rate	kg/THM	328	331	310	362	366	345
PCI Rate	kg/THM	180	180	180	-	-	-
NG Rate	kg/THM	-	-	-	100	100	100
Adjusted Fuel Rate *	kg/THM	490	493	472	482	486	465

* Adjusted fuel rate (kg/THM) = Coke rate + (0.9 x PCI rate) + (1.2 x NG rate)

As described above, the higher acid gangue content of sinter with its accompanying higher flux requirement, results in a larger volume of slag generated for all cases using sinter. Consequently, the coke rate increases as additional energy is required to melt the increased slag volume. The results of the calculations are shown in Table 7.

Table 7. Comparison of Key Performance Indices (KPIs)

KPI's	Unit	Scenario 1	Scenario 1A	Scenario 2	Scenario 3	Scenario 3A	Scenario 4
Slag Rate	kg/THM	259	273	188	254	253	168
Coke Rate	kg/THM	328	331	310	362	366	345
Yield Hot metal/ BF burden	t / t	0.61	0.61	0.66	0.61	0.61	0.66

Hatch's calculations confirm higher slag rates in Scenarios 1, 1a, 3 and 3a because of sinter use in the burden. As expected, coke rate and overall fuel rate are also higher for those scenarios compared to Scenarios 2 and 4. The higher fuel rate in the sinter scenarios also results in increased blast furnace carbon emissions, as shown in Section 5.2 below.

The above analysis indicates that significant benefits in blast furnace operation can be achieved by using pellets instead of sinter as the main BF burden constituent. Specifically, the use of pellets results in a higher production of hot metal per ton of burden charged, lower slag rate, lower coke rate and lower overall fuel consumption. The lower fuel consumption benefits, which can bring important operational savings to the steel producer, are illustrated in Figure 4 below.

There are also important environmental benefits of using pellets in the BF, which will be discussed in the next section.

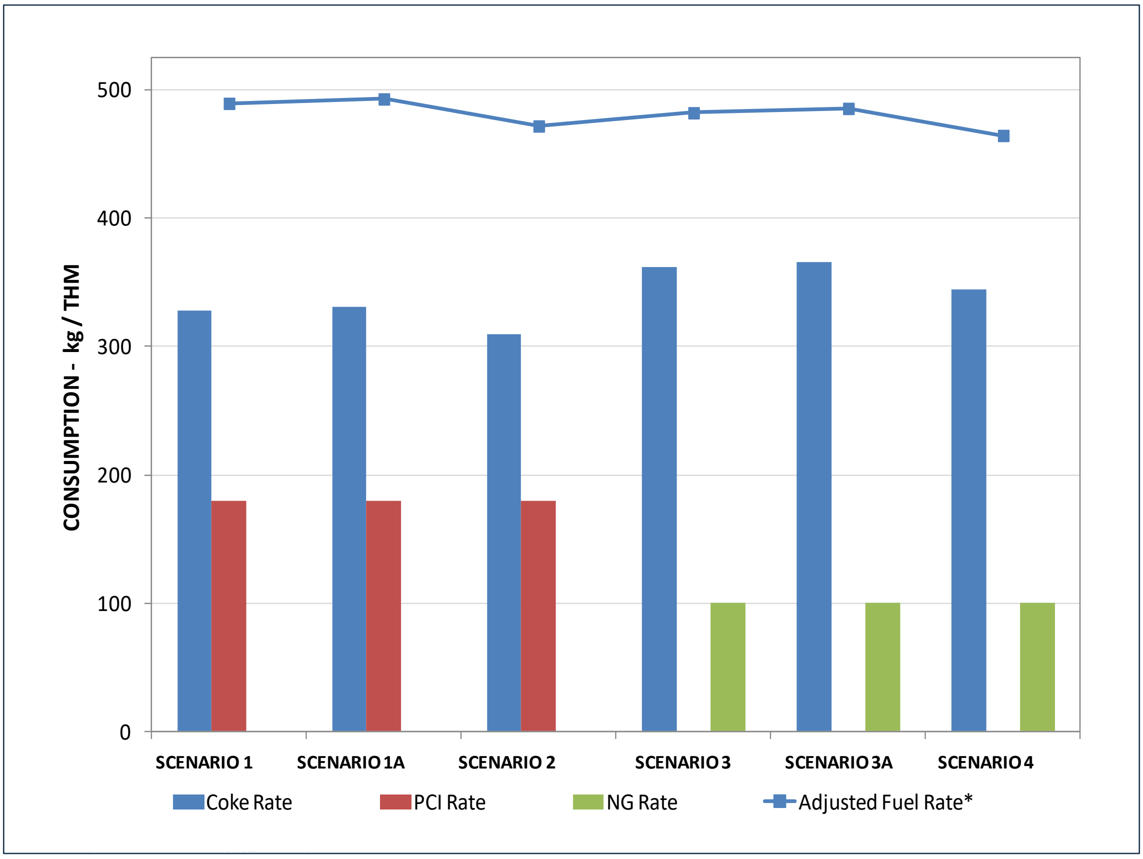

Figure 4. Comparison of fuel consumption between scenarios.

5 ENVIRONMENTAL IMPACTS

This section compares the environmental impact of blast furnace operations using pellets and sinter as the main burden constituents. The environmental impact comparison focuses on four items; atmospheric emissions at the agglomeration processes, blast furnace equivalent carbon emissions, transportation/materials handling emissions and ability to recycle solid wastes.

5.1 – Agglomeration Process Atmospheric Emissions

Table 8 below compares the typical emissions for the two agglomeration processes relevant to this paper: sintering and pelletizing. The figures are for existing sintering and pelletizing plants in the European Union some of which are considered as world benchmarks.

Table 8. Averaged maximum and minimum air emissions of sintering and pelletizing plants in the EU. (4)

Emissions	Unit	Sintering	Pelletizing
Gas emission	Nm3/t	1,500 - 2,500	1,940 - 2,400
Dust	g/t	41 - 559	14 - 150
SO_x	g/t	220 - 973	11 - 213
NO_x	g/t	302 - 1031	150 - 550
CO	g/t	8,783 - 37,000	< 400
CO_2	kg/t	162 - 368	17 - 193
VOC	g/t	37 - 673	5 - 40
PAH	mg/t	0.2 - 592	0.7 - 1.1

A quick inspection of Table 8 reveals that the environmental performance of the pelletizing process is significantly better than the sintering process, as the typical emissions of all the pollutants shown in Table 8 are lower. When the entire production chain starting at the agglomeration process is considered, there is a clear environmental benefit of using pellets instead of sinter as the main blast furnace burden constituent.

Newly constructed sintering plants include modern pollution control equipment designed to achieve lower emissions. Hatch currently does not have data of the most modern operations, but expects a superior performance compared to those European sinter plants reported in Table 8. Sinter producers and pollution control equipment manufacturers must continue to push the limits of innovation to comply with ever tightening environmental legislation worldwide.

5.2 – Blast Furnace Carbon Emissions

For the blast furnace, carbon is principally introduced to the process via coke, PCI and NG injection and any raw carbonate containing fluxes added to the furnace. The hot metal tapped out of the furnace typically contains 4.5% carbon, with the balance of carbon leaving the furnace in the top gas as a combination of CO and CO_2. Figures 5 and 6 show a comparison of the BF carbon footprint for the six scenarios studied in this paper.

Figures 5 and 6 illustrate that the blast furnace carbon emissions are higher in the cases where sinter is used as the main blast furnace burden constituent (Scenarios 1, 1a, 3 and 3a) with respect to those where 100% pellets are used (Scenarios 2 and 4). This is directly related to the higher coke rate resulting from the sinter operations as described in Section 3. Higher coke consumption directly translates into higher carbon emissions.

The above figures also illustrate that carbon emissions are higher when operating with PCI injection (Scenarios 1, 1a and 2) as opposed to natural gas injection (Scenarios 3, 3a and 4). This is due to the lower carbon content per unit output energy of natural gas compared to injected coal. The above analysis demonstrates that significant reductions in carbon emissions produced by the blast furnace can be achieved when using pellets as the main ferrous burden constituent.

5.3 – Transportation and Materials Handling Emissions

The calculations in Section 3 demonstrated that when using pellets instead of sinter, lower quantities of iron-bearing and carbon-bearing materials are charged to the blast furnace. It follows that using pellets results in freight savings and reduction in carbon emissions as less raw materials require transportation to the blast furnace site. In addition, materials handling operations at the blast furnace site are reduced, resulting in fewer particulate emissions from materials handling activities. Lower slag rates resulting from pellet use also contribute to the overall reduction in materials handling activities and consequently, particulate and carbon emissions.

5.4 – Internal Solid Wastes Recycling

An often-cited advantage of using sinter instead of pellets is that having a sintering plant within the integrated steelworks allows for the internal recycling of various solid wastes, such as blast furnace dust, BOF dust and mill scale among others. The pelletizing process is also capable of recycling these solid wastes although it is more challenging than for the sintering process as the recycled materials must be finely ground to the same particle size required for balling the iron ore concentrate. In North America, where many steel plants use 100% pellets and do not have a sinter plant, solid wastes are briquetted and added to the blast furnace and steelmaking furnaces to consume these waste materials.

6 – CONCLUSION

Large quantities of fine iron ore concentrate will enter the seaborne market over the next decade because of declining high-quality sinter fines. These concentrates must be pelletized to allow their usage in blast furnace and direct reduction processes. Thus, blast furnace operators that consume seaborne iron ore will gradually increase the use of pellets in the blast furnace burden as the pellet feed supply grows.

Hatch compared the use of sinter and pellets in the blast furnace and demonstrated that using pellets instead of sinter can result in significant technical and environmental benefits. These benefits include lower coke rate, lower slag rate, lower CO_2 emissions at the blast furnace, lower emissions at the agglomeration process and an overall lower fuel requirement to produce hot metal.

Blast furnace operators consuming iron ore from the seaborne market are thus encouraged to consider increasing the use of pellets to improve performance, reduce atmospheric emissions and better adapt to rapidly changing market conditions.

Acknowledgement

The authors are pleased to acknowledge the contribution of Mr. Mike Walsh for the iron ore market information. We also acknowledge Mr. Yakov Gordon, Ms. Janice Bolen, Ms. Marka Okrutny and other Hatch employees for their valuable input.

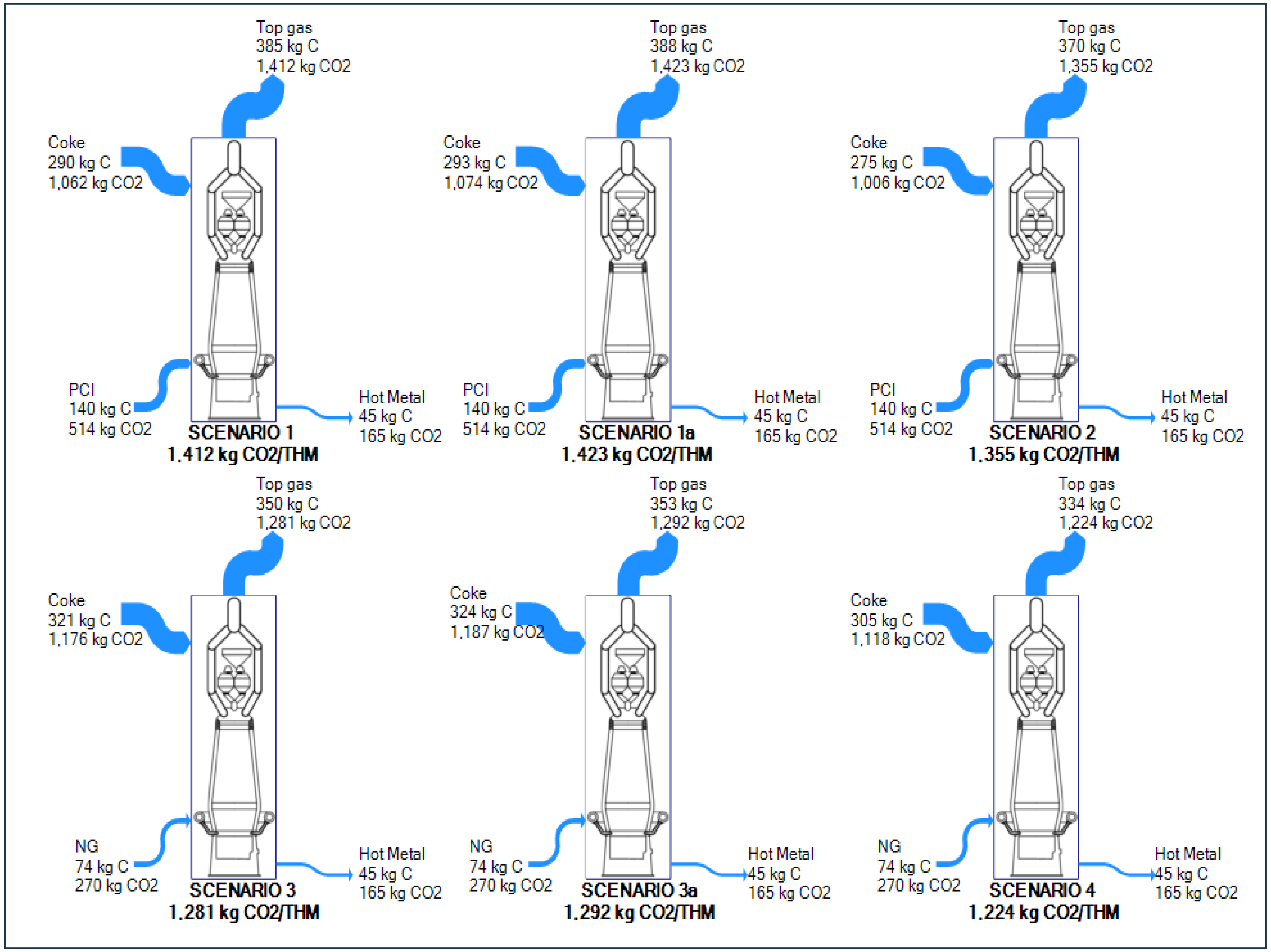

Figure 5. Comparison of BF carbon flows for the six scenarios.

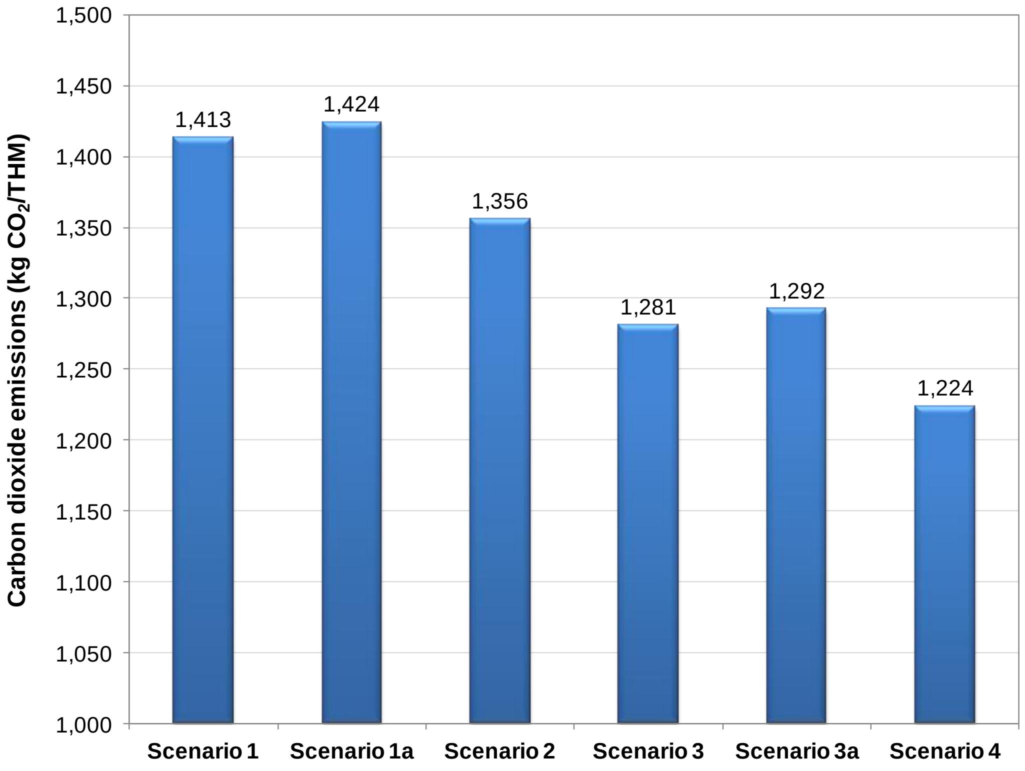

Figure 6. Comparison of BF carbon emissions for the six scenarios.

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ANNEX III

PAPER C

GUIDELINES FOR SELECTING PELLET PLANT TECHNOLOGY

GUIDELINES FOR SELECTING PELLET PLANT TECHNOLOGY [1]

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Abstract

As greater amounts of fine iron ore concentrates enter the marketplace, new investments in iron ore pelletizing capacity are inevitable to deliver these concentrates to steel industry consumers. When investors assess the two well proven pelletizing technologies, the straight grate and grate-kiln, the most appropriate choice may not be immediately evident. Certain technical, market and geographic factors will favor one technology over the other, investors need to be aware and understand these factors in order to make the best investment decision. Hatch has compared the merits of both technologies regarding plant construction; type of iron ore processed; energy consumption and flexibility to use a variety of fuels; ability to produce the required pellet quality; required plant capacity and scale; operating and capital costs and environmental performance. A technology assessment checklist that investors should consider when making a new pellet plant investment decision is presented.

Key Words: Iron ore pelletizing; Induration; Straight grate; Grate-kiln; Pellet plant capacity.

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[1] *Technical contribution to the 6th International Congress on the Science and Technology of Ironmaking – ICSTI, 42nd International Meeting on Ironmaking and 13th International Symposium on Iron Ore, October 14th to 18th, 2012, Rio de Janeiro, RJ, Brazil.*

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1 INTRODUCTION

In recent years, iron ore pellet production has increased as the availability of new lump ore and sinter fines declined during a period of rapidly expanding demand driven by the Chinese and Indian steel industries. Iron ore miners are developing lower grade resources that must be finely ground and processed to increase their Fe content to the levels required by the marketplace. This trend will continue as lower grade resources must be exploited to meet the continuing growth of global steel production. Therefore, new investments in pelletizing capacity are inevitable for the following reasons:

- Pelletizing is the only available technology that can agglomerate fine concentrates into a charge material suitable for blast furnace and direct reduction (DR) applications.
- Pellets are the only viable feed for new DR plants since lump ore of the required quality for DRI production is not available.
- Pellets provide unique advantages to end users like improved productivity of blast furnaces and DR plants, opportunity to increase the Fe content of the iron ore charge materials and superior environmental performance of pellet plants compared to sinter plants.
- Neither the blast furnace nor the DR processes will be replaced as the main iron producing systems in the near future.

There are two proven technologies to agglomerate fine iron ore concentrates into pellets: the straight grate (SG) and grate-kiln (GK) induration processes. Hatch has compared the merits of both technologies on a variety of aspects to better understand how investors/developers can best deploy these technologies to bring iron ore concentrates to the marketplace.

2 BACKGROUND

2.1 The Growth of Iron Ore Pelletizing

Since its origins in the 1950s, the production of iron ore pellets has steadily increased, accelerating in recent times. The first pellet plant installations were in North America, followed by South America, the CIS countries and Europe. In 2010, the world pelletizing capacity was about 480 Mtpa with over 150 Mtpa coming online during the last decade (1,2).Significant growth has occurred in China as a direct result of the dramatic expansion of the Chinese steel industry and the increasing tendency of Chinese blast furnace operators to use iron ore pellets in the ferrous burden (Figure 1).

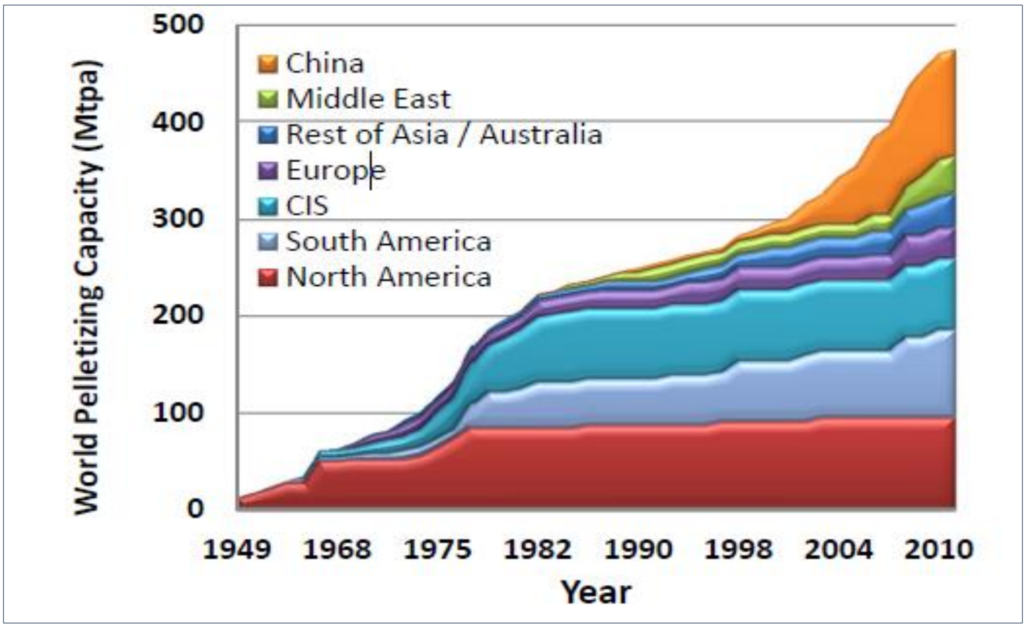


Figure 1. Regional growth of iron ore pelletizing since 1950.(1-3)

With the expected depletion of direct charge iron ores and high-quality sinter fines, the global production of iron ore concentrates will increase substantially in the coming years. Some concentrates will be pelletized at the mine site per the traditional approach, but an increasing amount of concentrate will be shipped to an intermediate location to be pelletized either closer to the end user or where low-cost energy is available. Figure 2 forecasts iron ore pellet feed production until 2025.

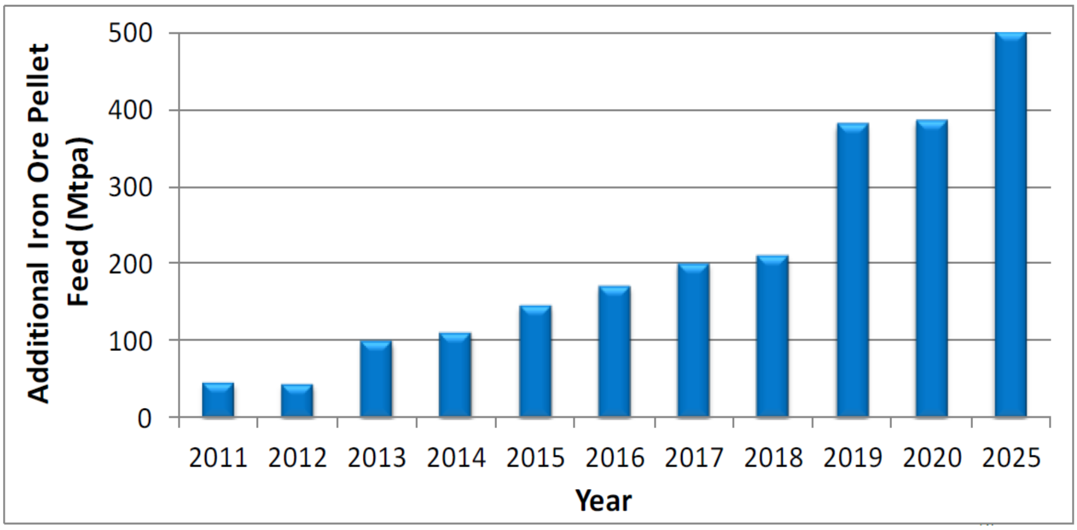


Figure 2. Additional iron ore pellet feed production that must be pelletized in the period 2011-2025 (1).

As much as 400 Mtpa of new concentrate is forecast to come into production by 2020 and up to 500 Mtpa by 2025.(1) At the moment, pelletizing is the only viable technology that can agglomerate these iron ore concentrates and meet the future demands for iron ore charge material. Assuming an average pellet plant capacity of 6 Mtpa, more than 80 new pellet plants would be required by 2025. This trend is already evident by the numerous pellet plant projects in construction or planning phases across the globe. Figure 3 presents the global distribution of announced greenfield pellet plant projects.

Due to the lack of announcements, Figure 3 likely underestimates the additional Chinese capacity. New capacity will be constructed on all continents as lower grade iron ore deposits are developed to meet the global steel industry demand for iron units.

2.2 Available Technologies

The straight grate (SG) and grate-kiln (GK) indurating processes are the two proven and widely accepted iron ore pelletizing processes. The processes were first developed in the 1950's and since then have undergone significant improvements. Both systems are capable of producing quality blast furnace and direct reduction pellets. The SG system advanced more quickly and today holds a larger share (>60%) of the global pelletizing capacity compared to the GK process (Figure 4).

The single most important objective of the induration process is to heat the green pellets to their firing temperature to agglomerate the fine particles together. Exposing the green pellets to a 1250 to 1350ºC firing temperature ensures that the required physical and metallurgical properties are achieved. Other important processes that occur are moisture removal, magnetite oxidation and calcination of fluxes and weathered ores. With the exception of magnetite oxidation, which is exothermic, all of these processes are endothermic. Thermal energy must be supplied to the system to meet these energy demands.

The SG process consists of a single furnace which encloses an endless chain of pallet cars (Figure 5). At the feed end of the furnace, a small layer of indurated pellets known as the hearth layer is laid down across the width of the pallet car and against the side walls to protect the pallet cars from the high heat loads experienced during induration. The unfired green pellets are then evenly distributed on top of the hearth layer pellets. Pellet firing is achieved by numerous burners arranged throughout the preheating and firing zones. Ambient air is blown counter-current to the flow of pellets through the furnace cooling zones to cool and recover the latent heat in the fired pellets.

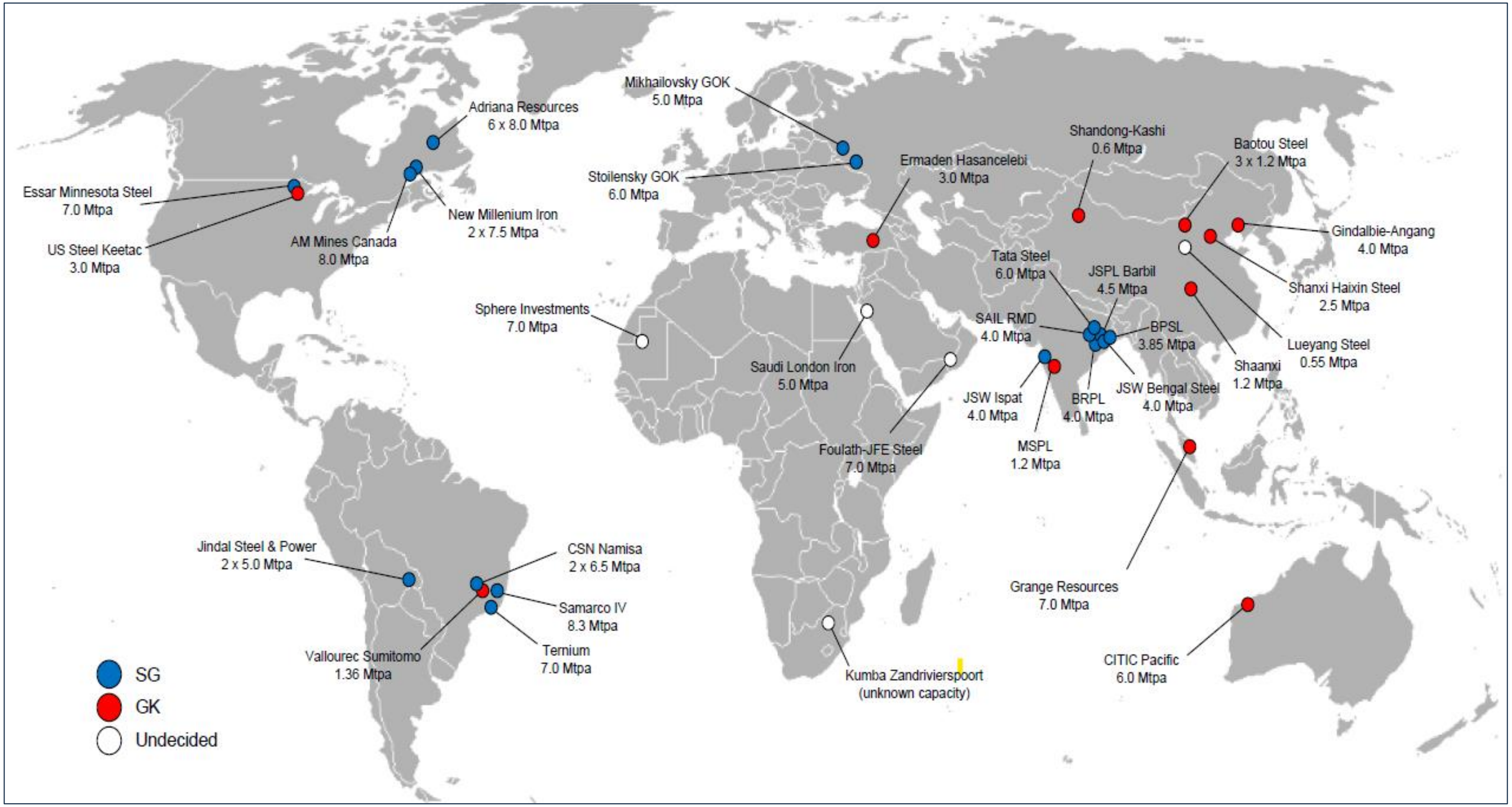


Figure 3. Global distribution of pellet plants under construction or planned (2012). (1,3)

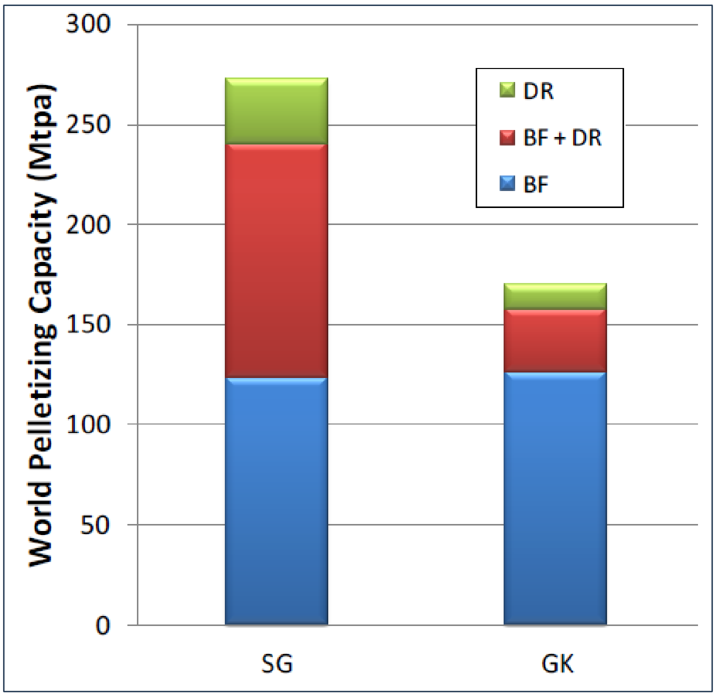


Figure 4. Pelletizing capacity by pellet quality. (1,2)

The heated process gases are then used for drying and pre-heating of the green pellets. Exhaust gases are typically cleaned in electrostatic precipitators and then discharged to the atmosphere through a stack.

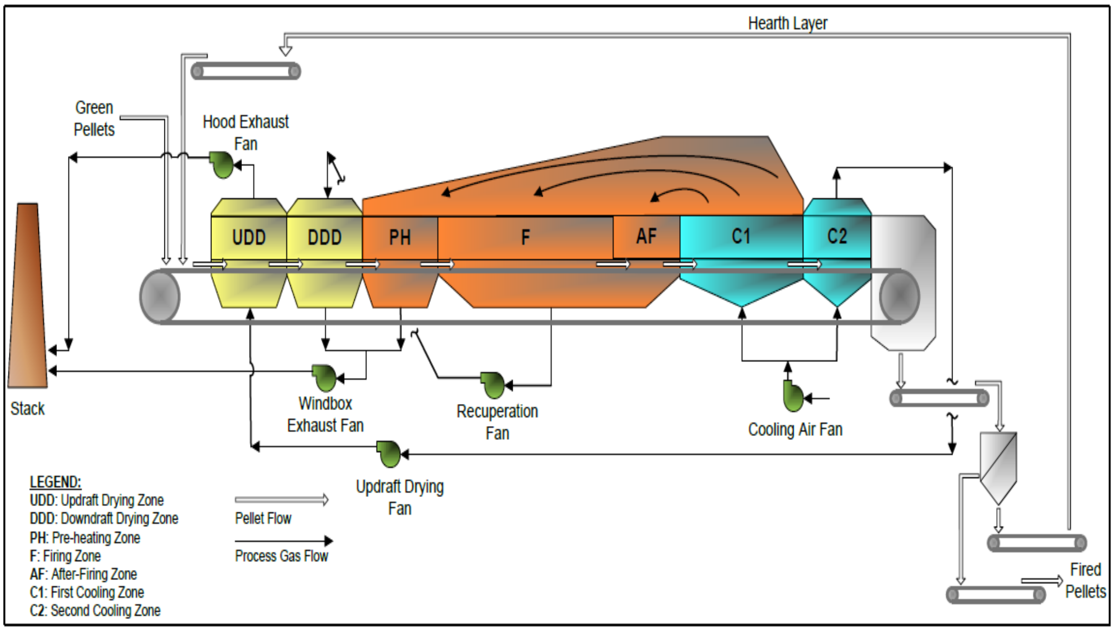


Figure 5. Schematic of the straight grate pelletizing process.

The grate-kiln (GK) system consists of three separate process units connected in series; the travelling grate, the rotary kiln, and the annular cooler. The green pellets are fed into the travelling grate where they are dried and pre-heated before transfer to the rotary kiln. The rotation of the charge in the rotary kiln ensures that all pellets are exposed to the same conditions and heated to the same temperature. From the rotary kiln, the hot pellets discharge into an annular cooler consisting of four cooling zones. Ambient air is blown counter-current through the hot pellet bed to cool them for subsequent material handling. The hot process air is recycled into the rotary kiln and various zones of the travelling grate (Figure 6).

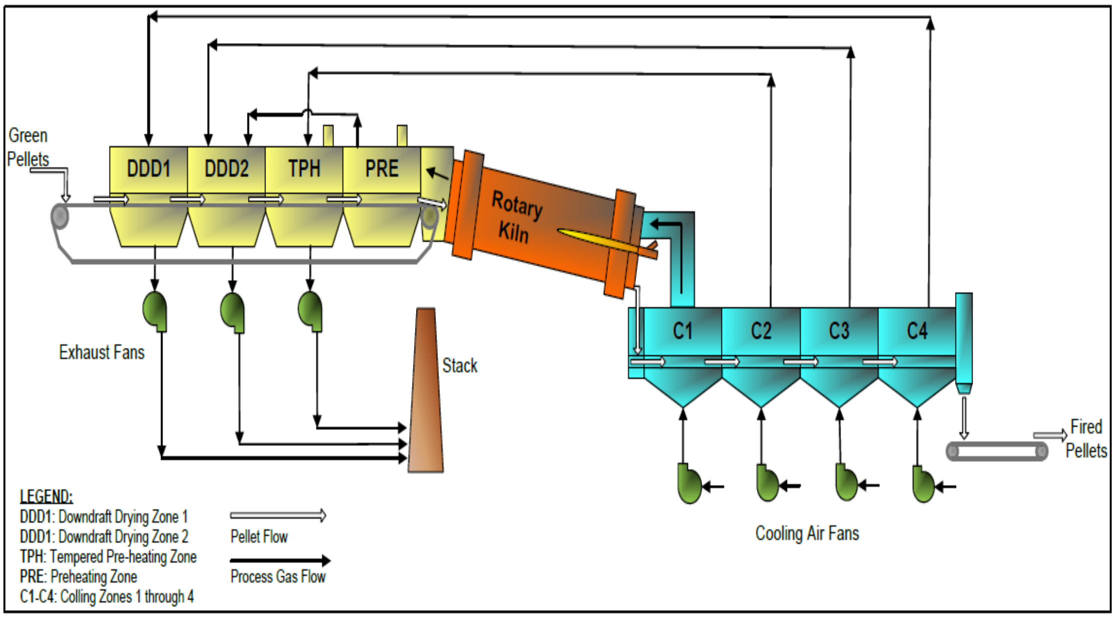


Figure 6. Schematic of Grate-Kiln pelletizing process (also known as the Rotary-Kiln Process).

3 TECHNICAL CHALLENGES

3.1 Iron Ore Mineral – Magnetite vs. Hematite

The most important driver when selecting a pelletizing technology is the iron ore mineral that must be processed. The two main groups of iron ore minerals that account for the vast majority of economically viable deposits are hematite and magnetite. Both minerals have been successfully pelletized; the SG process has demonstrated its flexibility to process either hematite or magnetite whereas the GK system has performed better when processing magnetite or mixtures of magnetite and hematite, typically higher than 40% magnetite, that often exist in iron ore deposits.

This is evident from the number of installations of each technology processing hematite, magnetite and naturally occurring hematite/magnetite mixtures (Figure 7).

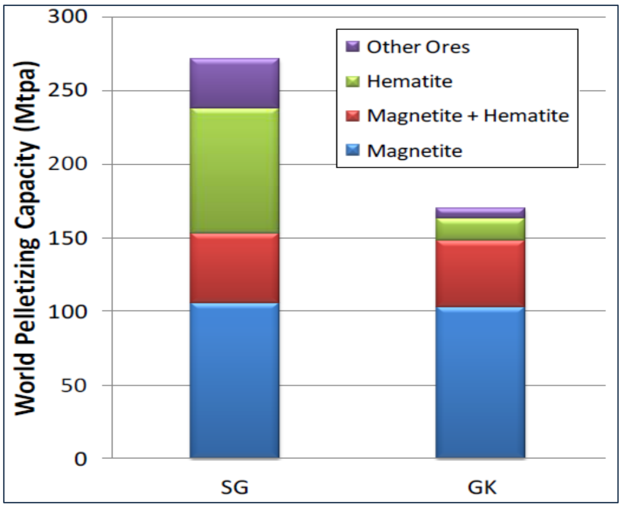


Figure 7. Pelletizing capacity by ore type. (4)

The most important difference between hematite and magnetite is that during induration magnetite is oxidized to hematite, producing significant heat uniformly throughout the pellet during the conversion process. As a result, fuel consumption, when pelletizing magnetite ores or magnetite/hematite mixtures is much lower than when pelletizing hematite ores.

The most often cited challenge to processing hematite ores in the GK is the relatively low strength of pre-heated pellets prior to the transfer between the travelling grate and the rotary kiln.

Weak pre-heated hematite pellets can be easily damaged by the tumbling action of the rotary kiln creating excessive fines. With magnetite and magnetite/hematite mixtures, magnetite oxidation in the travelling grate brings significant heat to the pellets strengthening them before transfer to the rotary kiln and final pellet firing. The SG process with its continuous series of pallet cars has no transfer points where the partially indurated pellets can be damaged. As a result, the SG system has a proven advantage in processing hematite ores.

SG operators have successfully introduced solid carbon additions (anthracite or coke breeze) to hematite green pellets as a means to reduce oil or natural gas consumption. A small amount of hematite is reduced to magnetite and then re-oxidized back to hematite producing significant heat in the oxidation process.(5) This greatly reduces the overall fuel requirements that must be provided by purchased burner fuels. Very few GK operators have been successful at processing hematite ores with carbon additions as the conditions available on the travelling grate are insufficient to create enough heat from magnetite oxidation to improve the pre-heated pellet strength.

3.1.1 Importance of ore characterization

Ore characterization and laboratory test work are crucial steps in the pelletizing technology selection process and plant design. Test work is the only way to confidently determine whether a given iron ore concentrate (magnetite, hematite or magnetite/hematite mixture) can be processed into high quality and commercially acceptable fired pellets. Laboratory test work also provides important process parameters such as the required temperature profile and treatment times that are necessary for pellet plant design and process optimization. Laboratory facilities and techniques are available to simulate both the SG and GK processes; scaling up the GK technology is more challenging than SG due to difficulties simulating the pellet transfer from the travelling grate to the rotary kiln at the laboratory or pilot plant scale.

3.2 Production Capacity

3.2.1 Plant scale

Since the introduction of pelletizing in the 1950s, plant capacity has dramatically increased for both the SG and GK technologies. In this respect, the SG has consistently maintained a scale advantage over the GK (Figure 8).

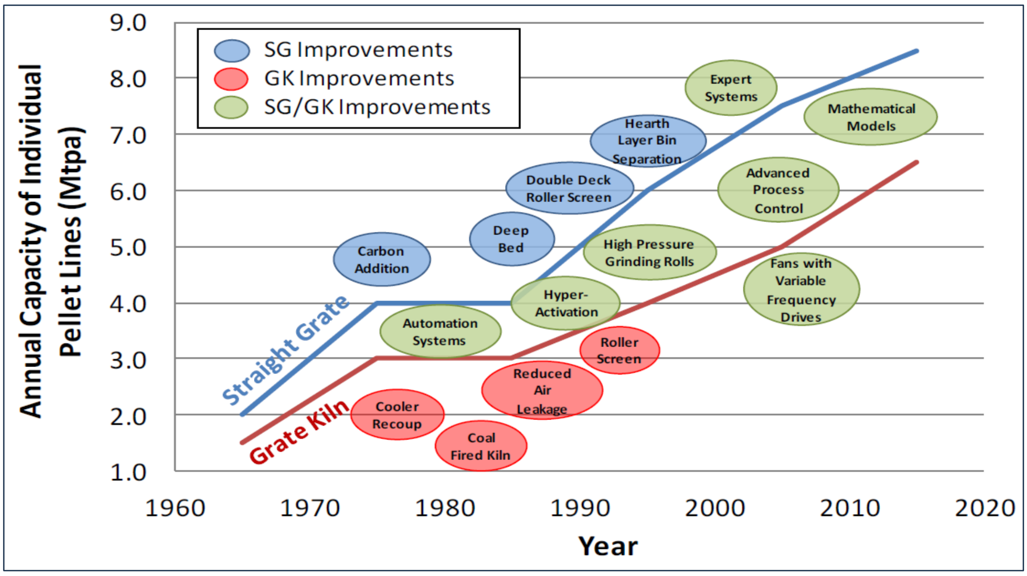


Figure 8. Pellet plant line capacity increase; 1960 to 2012. (1,2,6)

The design challenges differ for each technology regarding further increases in line capacity. The SG process must focus on engineering challenges to increase the width of individual pallet cars before car sagging issues emerge. Opportunities to increase the line length can be further exploited and larger machines are in design by the major equipment suppliers.

For the GK process, the engineering challenges are related to the maximum size of the rotary kiln; the diameter of the kilns used at the LKAB Kiruna KA3/KK4 and GIIC II plants approach the largest diameters used in the cement industry. GK equipment suppliers must focus on innovative ways to push the plant scale limits in order to remain competitive with SG equipment suppliers, especially to service emerging large iron ore mines with capacities between 10 and 30 Mtpa.

3.2.2 Plant availability

World-class operations for both SG and GK can consistently achieve over 90% availability per year, with best-in-class operations reaching as high as 94%.(1) Mechanical design improvements continue to be implemented such as improved pallet car changing systems that can significantly reduce related line stoppages. Reduction of refractory maintenance requiring major shutdowns is an opportunity to further increase the availability of both pelletizing systems through design improvements, usage of better refractory qualities and faster repair methods.

3.2.3 Productivity

An important factor when discussing capacity is plant productivity. For the SG process, world-class plants operate at as much as 25% over the name-plate capacity identified by the equipment supplier. Conversely, GK operators have been less successful operating at higher-than-design productivity than SG operators, with world-class GK plants normally operating just slightly above their name-plate rates. The SG operators have introduced more technology improvements as shown in Figure . In addition, a greater amount of reserve capacity has typically been built into the SG plant systems to facilitate production increases. Both SG and GK plants continue to implement practices to increase productivity and hence name-plate plant capacity. The main technical limitations to capacity increase are fluidization of the pellet bed in the case of the SG and kiln diameter for the GK.

3.2.4 Fuel Type

The burners used in the SG process normally operate with gaseous fuels (typically natural gas) or atomized liquid fuels (heavy fuel oil or diesel fuel). These burner systems are not suited to burn pulverized coal due to the high cost to prepare and pneumatically convey the pulverized coal to the many burners typically used in the SG furnace. Pulverized coal is however successfully used in the GK system where coal is supplied to a single main burner located in the discharge end of the rotary kiln. This represents a significant advantage of the GK in regions where gaseous and/or liquid fuels are either unavailable or prohibitively expensive. There are several examples of GK plants operating with pulverized coal around the world; LKAB Kiruna KA3/KK4 (Sweden), KOBE Kakogawa (Japan), CMP (Chile) and many plants in

China.(1) With a coal fired kiln burner, the GK operator must manage several unique operational and maintenance challenges such as higher dust formation, kiln ring development, coal ash contamination of pellets and lower flame stability. There is also the added investment and operating cost for the coal preparation plant including coal handling, drying, grinding and pneumatic transfer systems. Despite these extra costs and complexity there are situations where using coal as the main fuel is needed to make a pellet plant operation commercially viable.

3.3 Thermal Energy Consumption

In the pelletizing of magnetite ores and mixtures of magnetite and hematite, magnetite oxidation provides a significant portion of the thermal energy required. The remaining energy must be supplied by burners and solid carbon additions in the case of hematite ores. The total energy demand of a given pellet induration process depends on many factors, including ore type, pellet chemistry, fuel type, number/type of burners, and refractories/heat losses. Figure 9 summarizes the Hatch in-house data of various SG and GK pellet plants.

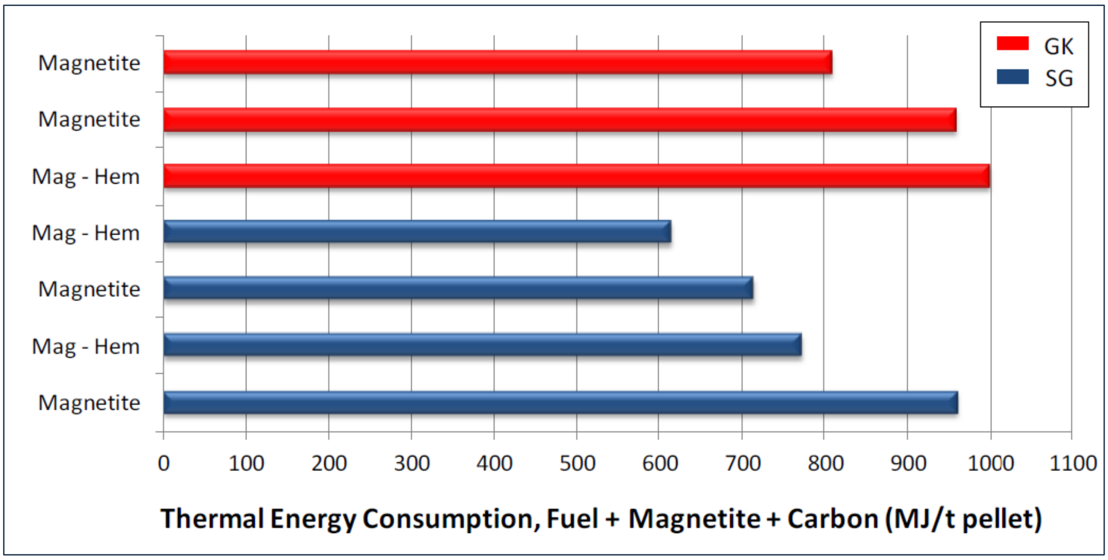


Figure 9. Thermal energy consumption for selected GK and SG pellet plants.(1)

Although on average there is no obvious advantage of one technology over the other, the data shows that the pellet plants with the lowest thermal energy consumption feature the SG technology. In principle, the difference in thermal energy consumption between the SG and GK systems is their ability to minimize heat losses. An inherent disadvantage of the GK system is the large surface area of the kiln, creating higher heat losses than in the SG. Also, the two material transfer points between the three reactors and the longer residence time in the GK (~75 minutes compared to ~30 minutes in the SG) provide additional heat loss opportunities. The SG system has the disadvantage of using the hearth and side layers which are an important heat sink not present in the GK.

3.4 Electrical Energy Consumption

The main electrical consumers in iron ore pelletizing are the large process gas fans followed by a smaller demand for the SG's drive motors or the GK's travelling grate, rotary kiln, and annular cooler drive motors. The GK uses smaller process gas fans compared to the SG as the pressure drop across the green pellet bed is much smaller in the GK. This is due to the thinner green pellet bed charged onto the GK's travelling grate, typically 180 mm compared to a 450-500 mm bed charged onto the SG pallet cars, including the hearth layer.

Greater use of variable frequency drives to regulate the speed of the main process fans reduces electricity consumption and allows operators the opportunity to re-balance flows for higher production rates in the future. Plant operators have used VFDs to reduce electricity consumption by 10-15% compared to fixed speed fans with louver controls. Hatch expects the usage of VFDs to become standard to further reduce the electricity consumption in iron ore pelletizing. Hatch's in-house data indicates that best-in-class GK plants can have an electricity consumption as low as 12 kWh/t while best-in-class SG plants processing the same type of mineral can be as low as 13 kWh/t.(1) Few plants have achieved such extremely low figures, the more common ranges being 15-20 kWh/t for GK plants and 20-25 kWh/t for SG plants.

3.5 Product Quality

Both pelletizing technologies have improved over the decades and produce good quality fired pellets suitable for blast furnace and direct reduction consumers. Product quality depends on several factors including ore type; end user quality demands; amount, type and quality of fluxes and binders; balling technology – drums or discs; efficiency of the green pellet screening equipment and the firing temperature profile used. Changing end user demands can result in a wide range of quality requirements that pellet plant operators must fulfill during the plant operation. In general, the GK system can achieve a more uniform pellet quality in terms of physical strength and fines generation in the final product. In the GK, all pellets are uniformly exposed to the peak firing temperature inside the rotary kiln. The residence time in the kiln while at indurating temperature is longer than for the SG system. This allows for uniform heat treatment of the pellets resulting in minimal quality variations within the fired pellets produced.

For the SG, the inherent nature of the pallet car design allows process gases to short circuit through the center of the pellet bed resulting in lower quality fired pellets in the pallet car side wall area. Charging of well screened and sized green balls with an even fill over the pallet car width will reduce opportunities for process gas short circuiting. Additionally, re-firing of hearth and side layer pellets which will ultimately report to the final fired pellet product can have a damaging effect on physical quality generating chips and fines.

With regards to fines generation, the SG system has an inherent advantage due to the lack of transfer points between reactors and little relative movement and abrasion between pellets and furnace parts. Hatch's in-house data shows that SG can produce about half the fines compared to the GK process during the firing operation, however more consistent physical properties for the GK may result in lower after- tumble fines. As most fines are re-processed, the overall plant Fe yield is similar.

3.6 Capex and Opex

Hatch has benchmarked the capital expenditure (Capex) of several SG and GK pelletizing projects (Figure 10.). Hatch's data indicates that the GK offers a slight Capex advantage of approximately 5-10% compared to the SG for the same production capacity. In warm climates, the rotary kiln and annular cooler can be installed outdoors reducing the building costs. Also, the GK has lower equipment costs due to the lack of a hearth layer return system and smaller process fans. In cold climates, the GK Capex advantage largely dissipates, making the cost of both technologies similar.

The operating cost (Opex) is comparable between both technologies with a few notable differences as illustrated in Table 1. The SG can have a fuel consumption advantage but when a GK process uses less expensive coal, its fuel costs can be less than a natural gas or oil-fired SG process. On the other hand, lower process fan duties due to lower bed pressure drops represent important savings in electricity favoring the GK. The SG system has an advantage in refractory consumption due to improved refractories with life spans of 5 to 7 years used for the furnace internal structures.

Other conversion costs like additives, water, utilities, spares, other consumables, and labor are essentially the same in both technologies. All things considered, the difference in Opex between the two technologies is marginal and there is no immediate advantage favoring either technology in terms of conversion costs.

Differences will be site specific and demand a careful review of the expected operating cost inputs.

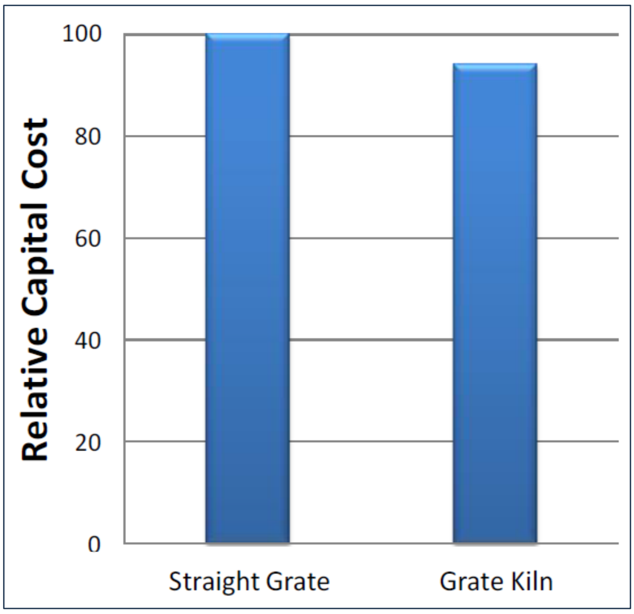
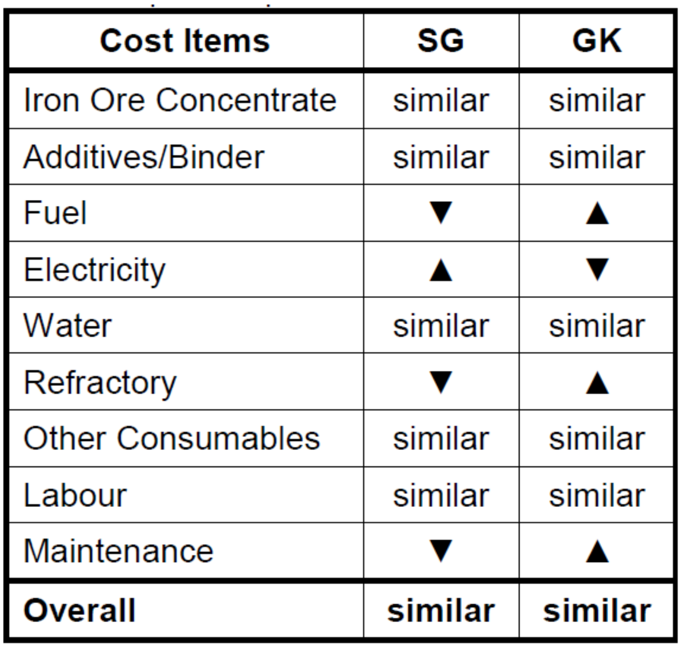


Figure 10. Capex comparison – SG vs. GK (1)

Table 1. Opex comparison SG vs. GK (1,7)



Cost Items	SG	GK
Iron Ore Concentrate	similar	similar
Additives/Binder	similar	similar
Fuel	▼	▲
Electricity	▲	▼
Water	similar	similar
Refractory	▼	▲
Other Consumables	similar	similar
Labour	similar	similar
Maintenance	▼	▲
Overall	**similar**	**similar**

3.7 Environmental Performance

New pellet plant projects are subject to increasingly stringent environmental regulations. The main air pollutants of concern are dust, sulphur oxides (SO_X), nitrogen oxides (NO_X) and greenhouse gases (GHG). Both technologies use state-of- the-art dust cleaning systems like electrostatic precipitators and dust scrubbers to meet the regulated emission levels. SO_X emissions can be controlled by using low- sulphur raw materials and fuels and by installing desulphurization equipment for the induration exhaust gases if required by environmental regulations.

In principle, NO_X emissions can be reduced through low-NO_X burners, but this is challenging in practice. In this regard, the GK has an advantage in that a single burner in the rotary kiln is the biggest contributor of NO_X and use of a special burner offers a solution to reduce NO_X emissions. For the SG, a commercially viable solution is more difficult due to the battery of smaller burners used in the pre-heating and firing zones. A commercially viable system to reduce NO_X in SG plants has not fully yet emerged, although successful trials have been conducted on a pilot scale. (8,9)

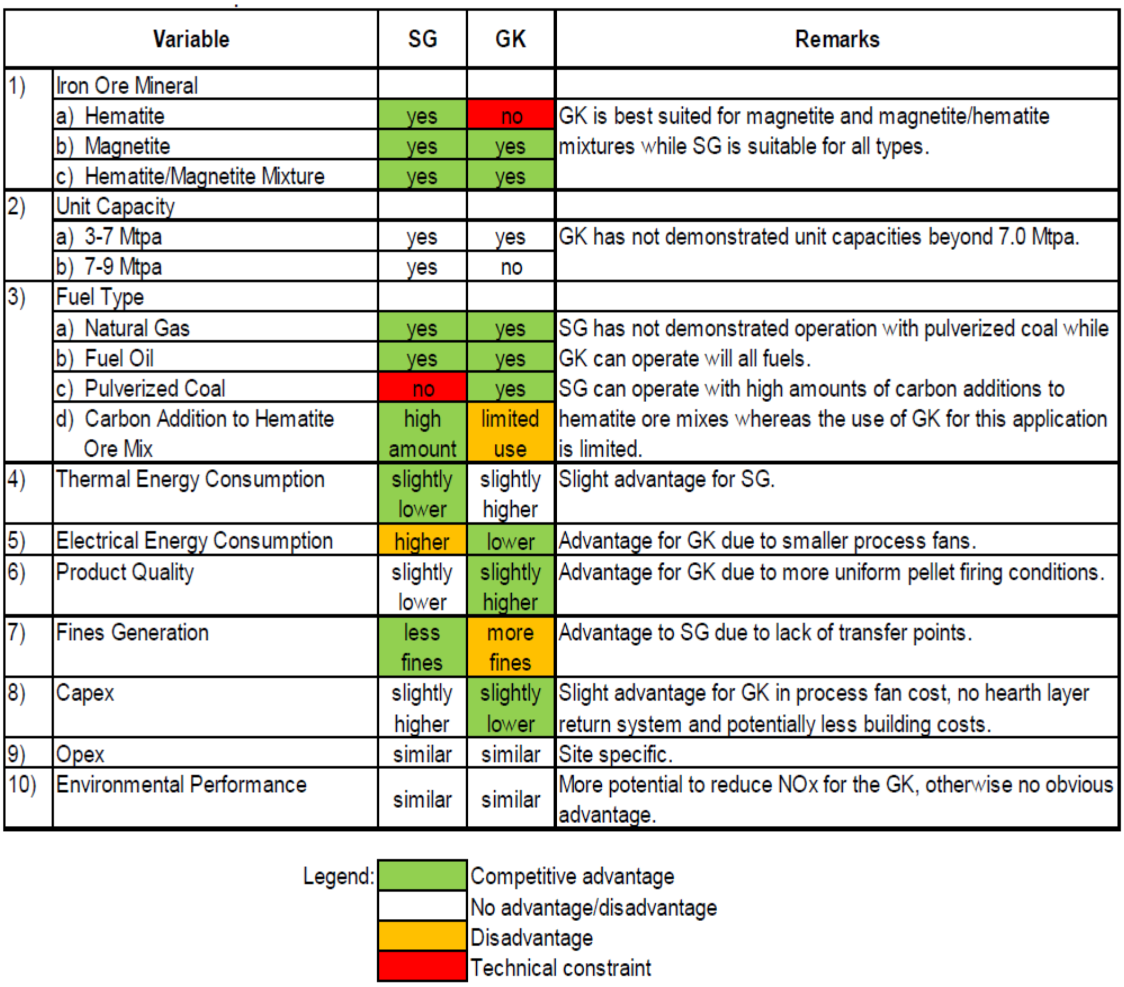
Greenhouse gas production, particularly CO_2 is directly related to the amount of fossil fuel combusted and the calcination of fluxes. The only way to minimize GHG emissions from either pelletizing process is to implement best operating practices to reduce energy consumption.

4 DISCUSSION

To better allow investors/developers to select the most appropriate pelletizing technology, SG or GK, Hatch compared the advantages and disadvantages of both processes over a range of relevant technical and economic aspects. Table 2 provides a summary of the important performance drivers that were analyzed.

The parameters presented in Table 2 illustrate that the choice between the two main pelletizing technologies is not obvious. The selection process is driven by mineral type, required plant capacity, fuel availability and other local factors. Other aspects such as energy consumption, product quality, and environmental performance may allow for further optimization, but the operating and capital costs for equivalent plants are similar. The technical and economic aspects discussed in this paper must be carefully analyzed on a project-by-project basis to make an appropriate investment decision.

Table 2. Overall comparison SG vs. GK.



Variable		SG	GK	Remarks
1)	Iron Ore Mineral			
	a) Hematite	yes	no	GK is best suited for magnetite and magnetite/hematite mixtures while SG is suitable for all types.
	b) Magnetite	yes	yes	
	c) Hematite/Magnetite Mixture	yes	yes	
2)	Unit Capacity			
	a) 3-7 Mtpa	yes	yes	GK has not demonstrated unit capacities beyond 7.0 Mtpa.
	b) 7-9 Mtpa	yes	no	
3)	Fuel Type			
	a) Natural Gas	yes	yes	SG has not demonstrated operation with pulverized coal while GK can operate will all fuels. SG can operate with high amounts of carbon additions to hematite ore mixes whereas the use of GK for this application is limited.
	b) Fuel Oil	yes	yes	
	c) Pulverized Coal	no	yes	
	d) Carbon Addition to Hematite Ore Mix	high amount	limited use	
4)	Thermal Energy Consumption	slightly lower	slightly higher	Slight advantage for SG.
5)	Electrical Energy Consumption	higher	lower	Advantage for GK due to smaller process fans.
6)	Product Quality	slightly lower	slightly higher	Advantage for GK due to more uniform pellet firing conditions.
7)	Fines Generation	less fines	more fines	Advantage to SG due to lack of transfer points.
8)	Capex	slightly higher	slightly lower	Slight advantage for GK in process fan cost, no hearth layer return system and potentially less building costs.
9)	Opex	similar	similar	Site specific.
10)	Environmental Performance	similar	similar	More potential to reduce NOx for the GK, otherwise no obvious advantage.

Legend:
🟩 Competitive advantage
⬜ No advantage/disadvantage
🟧 Disadvantage
🟥 Technical constraint

5 CONCLUSION

A large tonnage of fine iron ore concentrates will require pelletizing over the next decade due to the declining availability of lump and sinter fine ores to meet the growing iron ore demand. Hatch estimates that as many as 80 new pellet plants will be constructed to agglomerate concentrates into pellets for blast furnace and direct reduction consumers. Selecting the straight grate or grate-kiln process technologies depends on a range of technical and economic drivers with iron ore mineral type, plant size/scale and available fuel sources being the main aspects to consider. Additional performance differences must be assessed on a case-by-case basis to identify the best investment choice. A technology assessment checklist is presented to outline the advantages and disadvantages of each pelletizing technology so that investors can carefully consider the pelletizing technology of choice satisfying their particular needs. This checklist is employed by Hatch when performing pelletizing technology selection studies. Hatch anticipates that pelletizing technologies will further mature and improve as the installed capacity increases.

Acknowledgement

The authors are pleased to acknowledge the contribution of Ms. Kelly Scott and Mr. Mike Walsh for the iron ore market information. We also acknowledge Mr. Miguel Sabanero for our past discussions on the pellet plant technology and best ways to process iron ore concentrates.

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ANNEX IV

PAPER D

EVOLUTION AND GROWTH OF IRON ORE PELLETIZING

EVOLUTION AND GROWTH OF IRON ORE PELLETIZING (1)

José Murilo Mourão (2)

ABSTRACT

In the last years, pelletizing has been playing an important role for iron ore mining and ironmaking. Some factors have contributed to this fact like as: depletion of high-grade iron ore reserves, expansion of Chinese steel production, growth of direct reduction, better performance of pellets in the reduction reactors, and environmental restrictions in sintering. In this scenario, the world pelletizing capacity grew from 350 Mt/year in 2000 to more than 600 Mt/year in 2020. Invented in 1912 by the Swedish A.G. Andersson, the first industrial plant was built in the 1950's, in USA. Since that time, there were many developments in this field, which improved the performance of pelletizing processes in terms of productivity, energy consumption, product quality, among others. Some aspects of the evolution and growth of pelletizing technology are presented in this paper.

KEY WORDS: Pelletizing; Pellets; Iron Ore; Iron Making.

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(1) *To be presented at the 4th EMECR, International Conference on Energy and Material Efficiency and CO2 Reduction in Steel Industry, sponsored by ABM – Brazilian Association for Metallurgy, Materials and Mining – which will be held in São Paulo SP, Brazil, on June 7-9, 2022.*

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1 - INTRODUCTION

Pelletizing plants have been producing pellets for blast furnaces (BF) and direct reduction (DR), since the 1950's. (1) At the beginning, in USA, low grade taconite was used to feed installations of Shaft Furnaces (SF), presently an obsolete technology. Along the time, two other processes were developed with more capacity, better efficiency, and producing higher quality pellets. One of them, Traveling Grate (TG), also called Straight-Grate (SG), was able to treat any kind of iron ore. The other, Grate-Kiln (GK) or Rotary-Kiln (RK), was more indicated to process magnetite ores, presenting some limitations in working with hematite and hydrated ores.

For years, lump ore was the main iron material in the burden of blast furnaces. With the depletion of reserves of rich and hard ores, agglomeration technologies – sintering and pelletizing – were developed to agglomerate the growing quantities of ore fines and concentrates, generated by mining. Thus, besides the lump, sinter and pellets were made available as feedstock for BF and DR.

Sintering technology uses coarser iron ore fines (-6.3 mm / +0.15 mm) and pelletizing treats the concentrates and ultra-fine ores (-0,15 mm). When analyzing the world iron ore reserves and deposits it is clear the iron content is dropping, and more intensive concentration steps will be necessary in the future to comply with the needs of BF and DR. The more intensity the ore treatment, the bigger the ultra-fines production. Pelletizing is the only technology available to efficiently agglomerate these ultra-fines and iron concentrates. In this scenario, the iron ore pelletizing has been growing and will continue in this trend. The world capacity, which was 350 Mt of pellets in 2000, recently, in 2020, reached more than 600 Mt, as shown in figure 1. (2)

This paper presents some aspects of the evolution and growth of iron ore pelletizing technology.

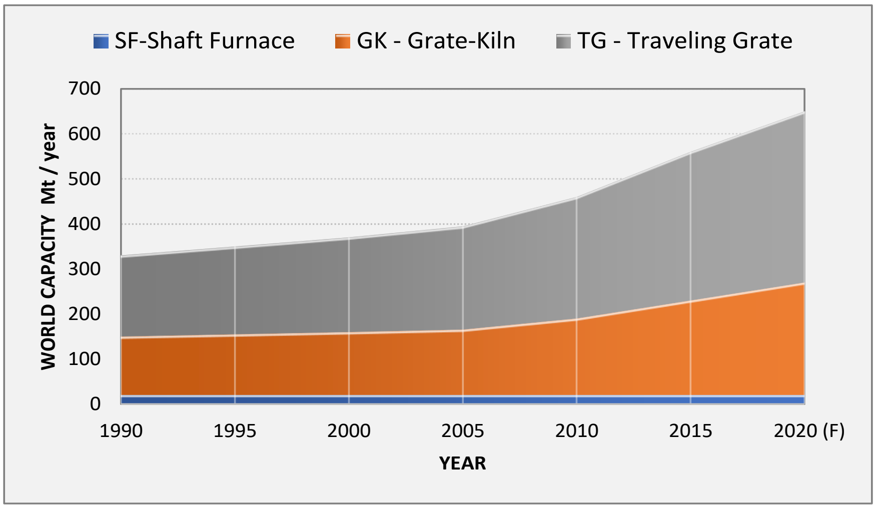

Figure 1– Evolution of world pelletizing capacity versus the different technologies.

2 - DEVELOPMENT

2.1 - Iron Ore Mining and Generation of Pellet Feed

The iron mining has been experiencing a dramatic expansion in recent years, due to the great escalation of production and consumption of steel in Asian countries, especially in China. (3) The accelerated growth of wealth in this region has increased the consumption of steel, in view of huge investments in infrastructure, housing construction and other segments.

Iron ore is the basic raw material for steelmaking, responding for the supply of metallic units (Fe) to the reducing reactors, BF and DR. Processed in these installations, the ore is reduced generating melt pig iron and solid DRI /HBI.

In mining operations, there is the exploitation of the ore body and the raw product – run of mine – is transported to be beneficiated. This raw material is subjected to treatment in operations like comminution, size classification, concentration, and dewatering, to adjust its characteristics in accordance with the requirements of ironmaking. In these treatment steps, a lot of fines (size range 0.0 – 6,3 mm) are generated. Such material is not proper to direct feed the reduction reactors, BF and DR. In general, the products of iron ore treatment are characterized in function of their size, as depicted in figure 2. (4)

The lump ore (+6.3 mm) is used as direct charge in ironmaking. High quality lump is becoming scarce, as time goes by. The sinter feed, a coarser fine (-6.3 mm/ +0,15 mm), and the pellet feed (-0.15 mm) need to be agglomerated by means of sintering and pelletizing technologies, respectively. It is important to mention the trend in increasing the production of pellet feed (ultra-fines and concentrates), in view of the depletion of high-grade iron ore reserves.

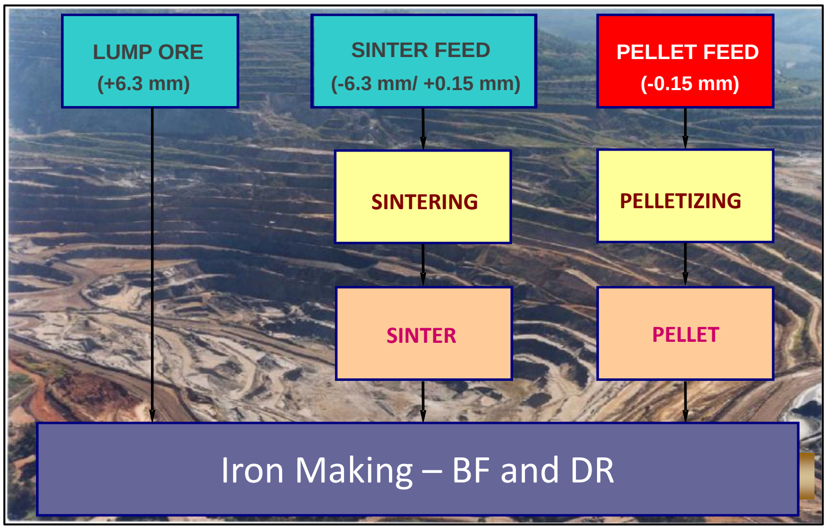

Figure 2 - Iron ore products and the corresponding use in iron making. (4)

2.2 – Aspects of Iron Ore Characteristics and Geometallurgy

The characteristics of iron ores vary in a wide range, depending on the geological origin. Magnetite, hematite, earthy and hydrated ores are the main types treated by pelletizing. Due to this fact, when developing a new pelletizing project and or analyzing process performance, it is important to consider the geo-metallurgical data of the selected ore or mixture of ores.

In terms of hematite ores, the figure 3 shows how the geological formation, supergenic or metasomatic (also called metamorphic), impacts the structure and the texture of the ores. The supergenic ones are soft, present small $Fe2O3$ crystals and high micro-porosity, the contaminants are dispersed in the matrix, etc. On the other hand, the metasomatic ores are hard, have big crystals, are compacts with low porosity, have clean surfaces, etc.

Geometallurgy is one of the most important tools to develop and implement a greenfield pelletizing project. It also plays a fundamental role in process control of pelletizing units.

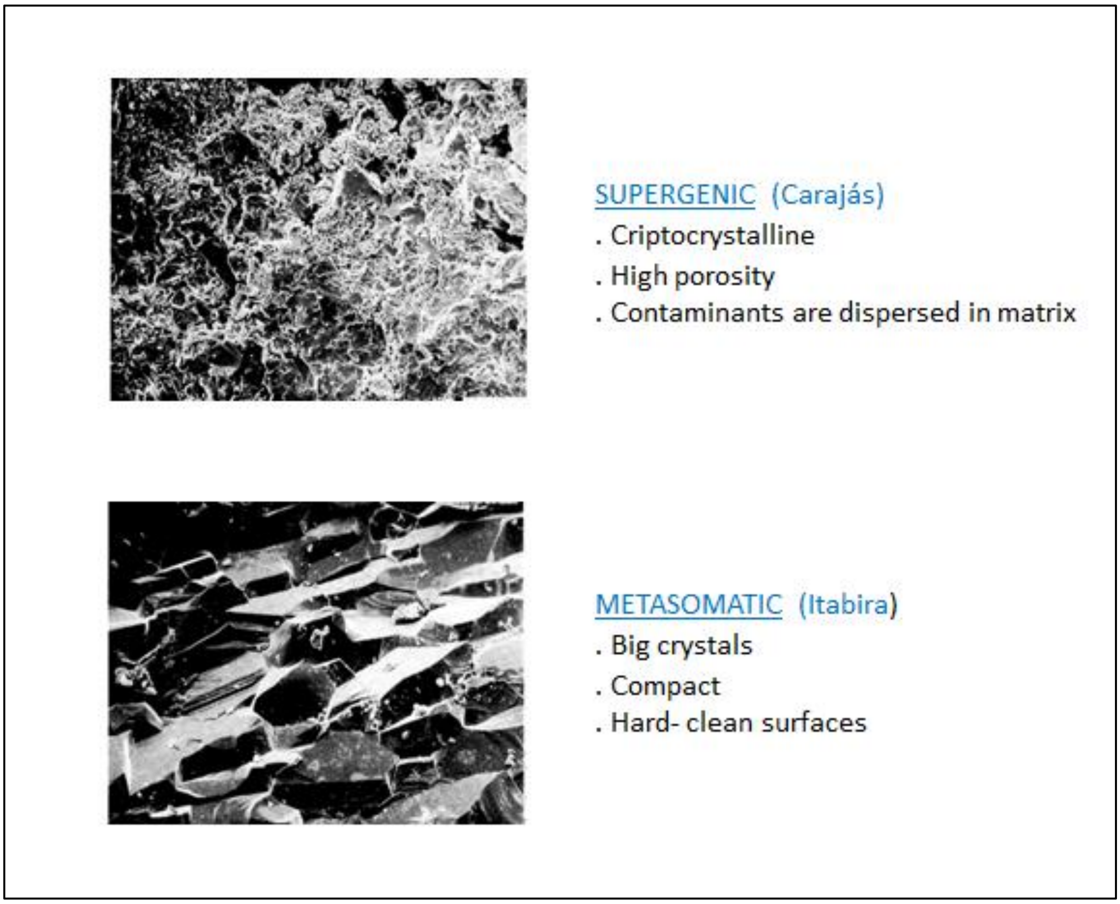

Figure 3 – Typical microstructures of Brazilian hematite ores, supergenic and metasomatic, from Carajás and Itabira regions. (5)

By means of lab and pilot plant tests with certain kind of ore, it´s possible to correlate the ore properties with its performance data in mining, beneficiation, and metallurgical processes. In other words, Geometallurgy links the ore characteristics with its behavior in industrial plants.

GEOMETALLURGY: Ore Characteristics → Process Performance

Based on his own experience in studying such kinds of hematite ores, in different scales, the author prepared the table 1, which summarizes this subject.

Table 1 – Some characteristics of hematite ores, metasomatic and supergenic, and their performance in mining/ beneficiation and pelletizing/reduction plants. (6)

CHARACTERISTIC / BEHAVIOR	**METASOMATIC**	**SUPERGENIC**
In Natura		
. Crystall size	Big	Small
. Al2O3, P, Mn	(-)	(+)
. L.O.I	(-)	(+)
In Mining/Beneficiation		
. Fines in mining	(-)	(+)
. Liberation size	(>)	(<)
. Iron recovery	(+)	(-)
. Grindability	(-)	(+)
. Filterability	(+)	(-)
In Pelletizing/Reduction		
. Ballability	(-)	(+)
. Productivity of furnace	(+)	(-)
. Fines generation	(-)	(+)
. Compression strength	(+)	(-)
. Reducibility-Metallization	(-)	(+)
. Fines generation during reduction	(+)	(-)

It is clear how different are the iron ores and consequently their behaviors in mining and metallurgical processes. There is not a perfect iron ore! The practice has shown the most convenient for pelletizing is a mixture of different types to take advantage of the best properties of each one.

In designing of pelletizing plants, the characteristics of the ore influence the process route, size of equipment, and the core technology, among other aspects.

2.3 – Technologies of Iron Ore Pelletizing

Currently, two technologies dominate the market of pelletizing, accounting for more than 95% of the world production: Traveling Grate (TG) and Grate-Kiln (GK). The old and obsolete low-capacity process, called Shaft Furnace (SF), has been also in activity with some furnaces in operation, especially in China. However, gradually, they tend to disappear, bearing in mind the higher operating costs.

The TG furnaces are reaching the capacity of 10 Mt/year, while those of GK type are being improved to achieve 6 Mt/year. The capacity of the SF furnace is limited to only 0.5 Mt/year. The TG process is indicated for treating any type of iron ore. The GK is more appropriate for magnetite ores or mixtures with hematite, although there are some GK plants operating with 100% of hematite ores.

Independently of the pelletizing technology, there are 3 stages of ore treatment, aiming at the production of pellets (figure 4), as follows:

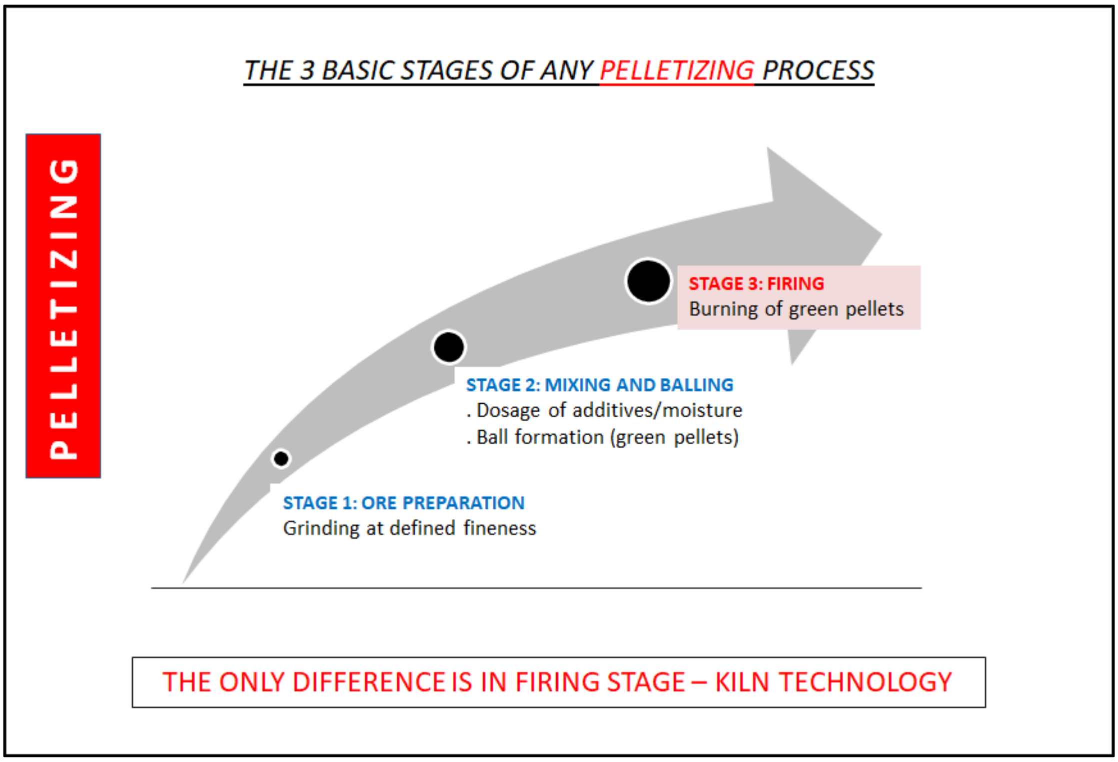

Figure 4 – Common processing stages to all pelletizing technologies. (1)

Stage 1 - Treatment or Preparation of Iron Ore.

The treatment of iron ore is necessary to achieve the fineness required to the formation of green pellets in the subsequent stage 2, as shown in figure 4. This fineness depends on the characteristics of the ore. In general, specific surface area in the range of 1700 to 2200 cm2/g and mean particle diameter of less than 0.074 mm are required for the formation of green pellets in balling section. These conditions are achieved by grinding the iron ore and additives. Normally, the pellet feed obtained by concentration of low-grade ores is coarser than this specification. Dry and wet grinding can be used for the comminution, depending on the ore type and the existing conditions at the site of a pelletizing plant. Grinding is very costly, both in energy and wearing materials (grinding media, and lining).

There are pelletizing plants where the stage 1 is not present in the flowchart. This condition means that the plant receives from the supplier the ore already ground and ready for balling.

Stage 2 - Mixing and Balling.

After the preparation of iron ore, defined amounts of binder (bentonite, hydrated lime, organic compounds, and others), fluxes (limestone, dolomite, olivine, dunite, magnesite, etc.) and, possibly, solid fuel (anthracite, coke breeze, pet coke, charcoal, etc.), are added to it in order to produce the specified pellet. The fluxes are important to adjust the basicity of the pellets, giving them the chemical, physical and metallurgical properties, suitable for their use in ironmaking. Preferably, these additives should be beneficiated separately and dosed to iron ore at the entrance of the mixer. However, each plant makes its way, according to the local conditions and the availability of raw materials. In certain plants, the fluxes are added to the pile of iron ore and ground together.

After filtration and the addition of additives, if necessary, adjustment is done in moisture content of the mixture to be balled. The materials are fed to the mixer after what the mixture is sent to balling unit. Discs or drums can be used to produce green pellets. The pelletizing discs are more often used, in view of their simplicity. The size of the green pellets can be adjusted through the control of some process variables. Before entering the pelletizing furnace, the green pellets are sieved (preferably 8-18 mm) and removed the bigger and smaller balls which are recirculated in the process.

Stage 3 - Firing

This is the main stage of any pelletizing technology and the one of biggest capital and operational expenditures (CAPEX and OPEX). In this stage, the main equipment is the furnace where the pellets are fired and acquire the strength and metallurgical properties. The basic differences between the technologies are the furnace design and some process parameters.

The Traveling Grate (TG) and Grate-Kiln (GK) are utilized to burn green pellets obtained in the balling section. To produce strong pellets with suitable metallurgical properties, the green balls are submitted to high temperatures (1250 to 1350 °C) inside the furnaces, under a well-controlled residence time. These furnaces have different zones, where the pellets are exposed to drying, preheating, burning, after firing and cooling. The flows of hot and cold gases (updraught and downdraught), temperatures, pressures and other variables of each zone are controlled depending on the kind of pellet under firing. The recirculation of hot gases from the cooling zone recovers part of the heat, improving the thermal efficiency of the furnaces. The product of the furnace is screened and the pellet chips (< 5 mm) are separated from the final product (fired pellets) and discarded.

The TG technology consists in a tubular shaped furnace with a traveling grate inside, as shown in figure 5. All burning phases are conducted in this single device, and there is not relative movement of the pellets among themselves or with the grate. The TG furnaces have been performing very well with pellets made of any kind of ore.

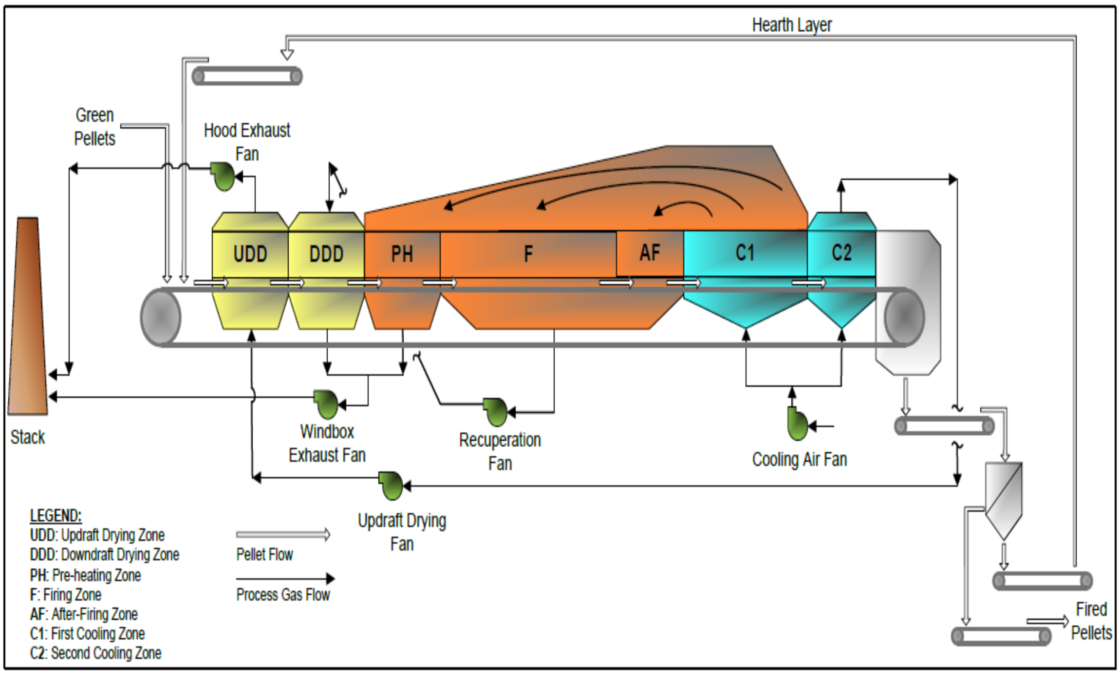

Figure 5 – Sketch of a Traveling Grate (TG) furnace. [(7)]

On the other hand, GK technology has three independent reactors connected in series: a movable grate, a rotary reactor (cylindrical and inclined) and a cooler in circular shape, as shown in figure 6. The industrial experience has proven that GK has some limitations to treat pellets made of hematite and hydrated ores (limonite or goethite). The main reason is that there is a huge drop of weak pellets in the transfer point between the grate (just after the preheating zone) and the rotary kiln. At this point, the hematite pellets have still low strength what causes the formation of chips and fines. In case of magnetite pellets the situation is quite different. The magnetite crystals (Fe_3O_4) react exothermically with the oxygen from the gases in pre-heating zone, generating heat and strong bridges between the oxidized new crystals of hematite (Fe_2O_3). Thus, when processing magnetite pellets, there is not the lack of strength seen for pellets made of hematite or hydrated ores.

Most recent experiences on the use of hematite ore in GK have shown that it is necessary to raise, significantly, the temperature of gases in preheating zone to increase the strength of the dried and preheated pellets at the referred transfer point. But this procedure implies in getting lower values of heat efficiency.

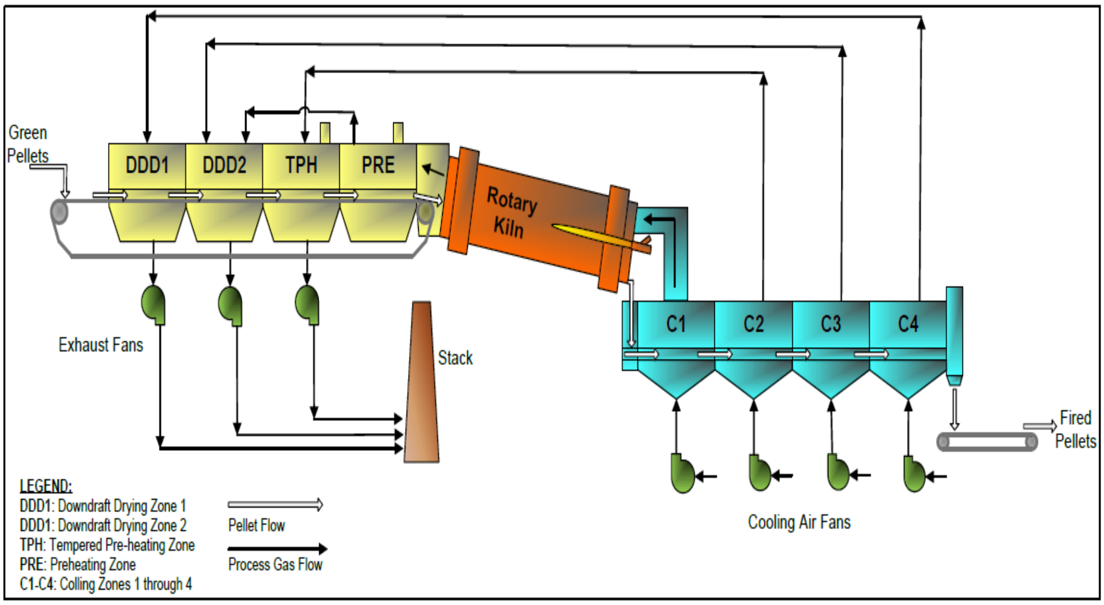

Figure 6 – Sketch of a Grate-Kiln (GK) furnace. (7)

In any case, what will define the use of one or another technology is the feasibility study of the project, considering a certain type of ore. Laboratory, bench, and pilot plant tests are necessary to support the process of decision making and the project designing.

In table 2, it is shown a list of important points to be considered, when selecting a pelletizing technology. (7) Figure 7 depicts the influence of the plant capacity on the specific capex. The higher the annual capacity, the lower the specific capex, what favors the Traveling Grate plants. It can be seen TG and GK have comparable CAPEX for the same capacity. By the other side, Chinese GK plants (Rotary-Kiln) built in China are subsidized and present much lower CAPEX, compared to projects implemented outside China.

Since the introduction of pelletizing in the 1950's, plant capacity has dramatically increased for both the TG (or Straight Grate) and GK technologies. In this respect, the TG has consistently maintained a scale advantage over the GK, as shown in figure 8, where can also be seen the process improvements for both technologies that influenced the gain in capacity over the time. (7)

The design challenges differ for each technology regarding further increases in line capacity. The TG (or Straight Grate) process must focus on engineering challenges to increase the width of individual pallet cars before car sagging issues emerge. Opportunities to increase the line length can be further exploited and larger machines are in design by the major equipment suppliers.

Table 2 – Factors to be considered in selecting pelletizing technology. (7)

Variable		**TG** (Traveling Grate)	**GK** (Grate-Kiln)	**Remarks**
1)	***Iron Ore - Mineral***			
	a) Hematite	yes	no	GK is best suited for magnetite and magnetite/hematite mixtures while TG is suitable for all types.
	b) Magnetite	yes	yes	
	c) Hematite/Magnetite Mixture	yes	yes	
2)	***Unit Capacity***			
	a) 3 - 6 Mt/year	yes	yes	GK has not demonstrated unit capacities beyond 6.0 Mtpa.
	b) 6 - 9 Mt/year	yes	no	
3)	***Fuel Type***			
	a) Natural Gas	yes	yes	TG has not demonstrated operation with pulverized coal in the burners while GK can operate wilth all fuels.
	b) Fuel Oil	yes	yes	TG can operate with high amounts of carbon additions to hematite ore mixes whereas the use of GK for this application is limited.
	c) Pulverized Coal	no	yes	GK can use pulverized coal injection in burner.
	d) Carbon Addition to Hematite Ore Mix	high amount	limited use	GK has very litle possibility of using coal in the pelletizing mix. In TG, carbom addition impacts positively the productivity and quality (porosity, reducibility, metallization, etc.).
4)	Thermal Energy Consumption	slightly lower	slightly higher	Slight advantage for TG.
5)	Electrical Energy Consumption	higher	lower	Advantage for GK due to smaller process fans.
6)	Product Quality (strength)	slightly lower	slightly higher	Advantage for GK due to more uniform pellet firing conditions.
7)	Fines Generation	less fines	more fines	Advantage to TG due to lack of transfer points.
8)	Capex	slightly higher	slightly lower	Slight advantage for GK in process fan cost, no hearth layer return system and potentially less building costs.
9)	Opex	similar	similar	Depends on the site the plant is built.
10)	Environmental Performance	similar	similar	More potential to reduce NOx for the GK, otherwise no obvious advantage.

For the GK process, the engineering challenges are related to the maximum size of the rotary kiln. The diameter of the kilns used at the LKAB Kiruna KK3/ KK4 and GIIC II plants approaches the largest diameters used in the cement industry. GK equipment suppliers must focus on innovative ways to push the plant scale limits to remain competitive with TG equipment, especially to service emerging larger iron ore mining capacities.

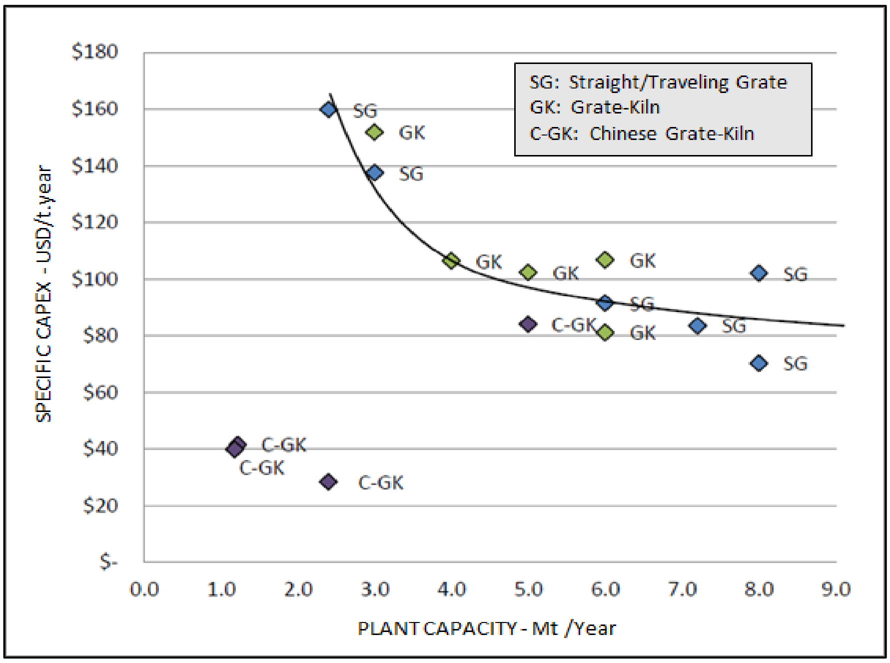

Figure 7 – Specific CAPEX versus plant capacity of different projects. (8)

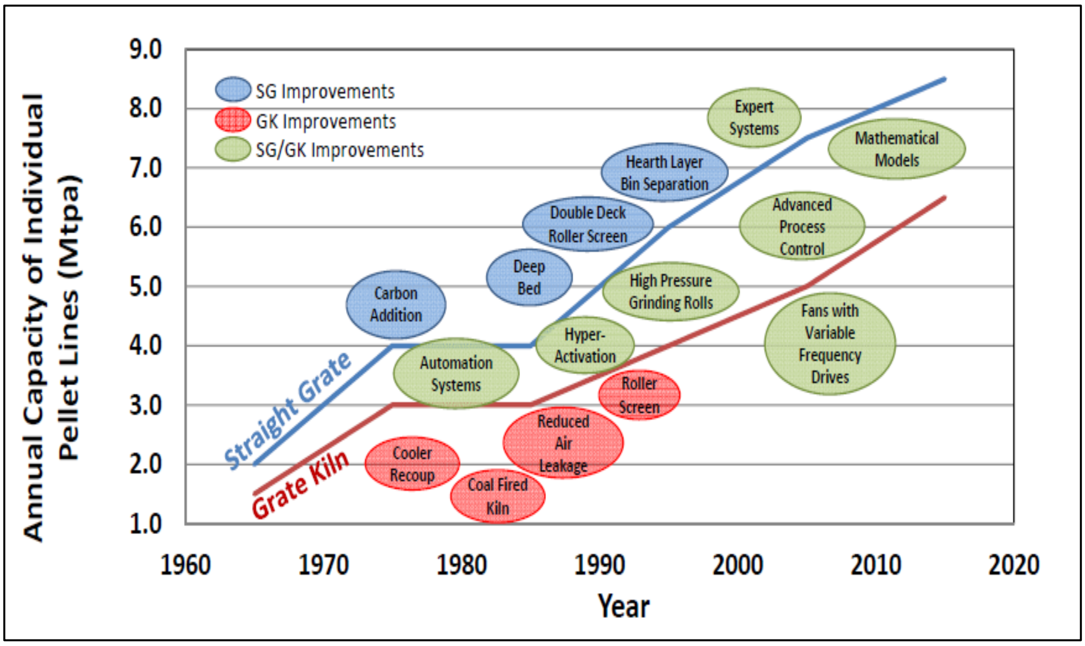

Figure 8 - Evolution of TG and GK plant capacity - process improvements. (7)

There are many configurations of pelletizing plants. Some of them have only the mentioned stages 2 and 3. In this case, the iron ore and additives are received already ground and ready to be balled. In others, all the 3 stages are presented. Only as a reference, the following figure 8.1 depicts a general flowchart of a pelletizing plant, with the following characteristics:

- This plant has the 3 mentioned stages, highlighted in the figure: iron ore preparation in rose, mixing and balling in light gray and firing in yellow color.
- Wet grinding system for iron ore preparation.
- Discs for balling.
- Traveling Grate technology for firing the pellets.

2.4 – Iron Ore Resources and Drivers of Pelletizing Growth

The volume and quality of the world iron ore reserves are changing as the geological survey and mineral exploration advance, fruit of the great expansion of Chinese demand for iron oxides. World resources are estimated to be greater than 800 billion t of crude ore containing more than 230 billion t of iron (Fe). (9) Along the time, it has been observed quality worsening and depletion of the reserves of compact ores, and consequently scarceness of high-quality lump ores. Lower grade reserves have pushed the beneficiation to increase the iron content of concentrates and eliminate the contaminants. The more intensive the ore treatment, the higher the fines generation (- 0.15 mm), these last not directly usable in reduction reactors. In view of this tendency, pelletizing has been growing in importance, and producing high quality agglomerates with these fines (- 0.15 mm) to feed the blast furnaces and direct reduction reactors. The following table 3 depicts the data of the most important proven reserves of iron ore in the world, emphasizing the amounts of Australia, Brazil, and Russia.

Table 3 – Main proven iron ore reserves in the world (2020). (9)

COUNTRY	RESERVES		
	Crude Ore (Mt)	Fe Content (Mt)	% Fe
Australia	48000	23000	48
Brazil	29000	15000	52
Russia	25000	14000	56
China	20000	6900	35
India	5500	3400	62
Ukraine	6500	2300	35
Canada	6000	2300	38
Others	30000	14100	47
World Total	170000	81000	48

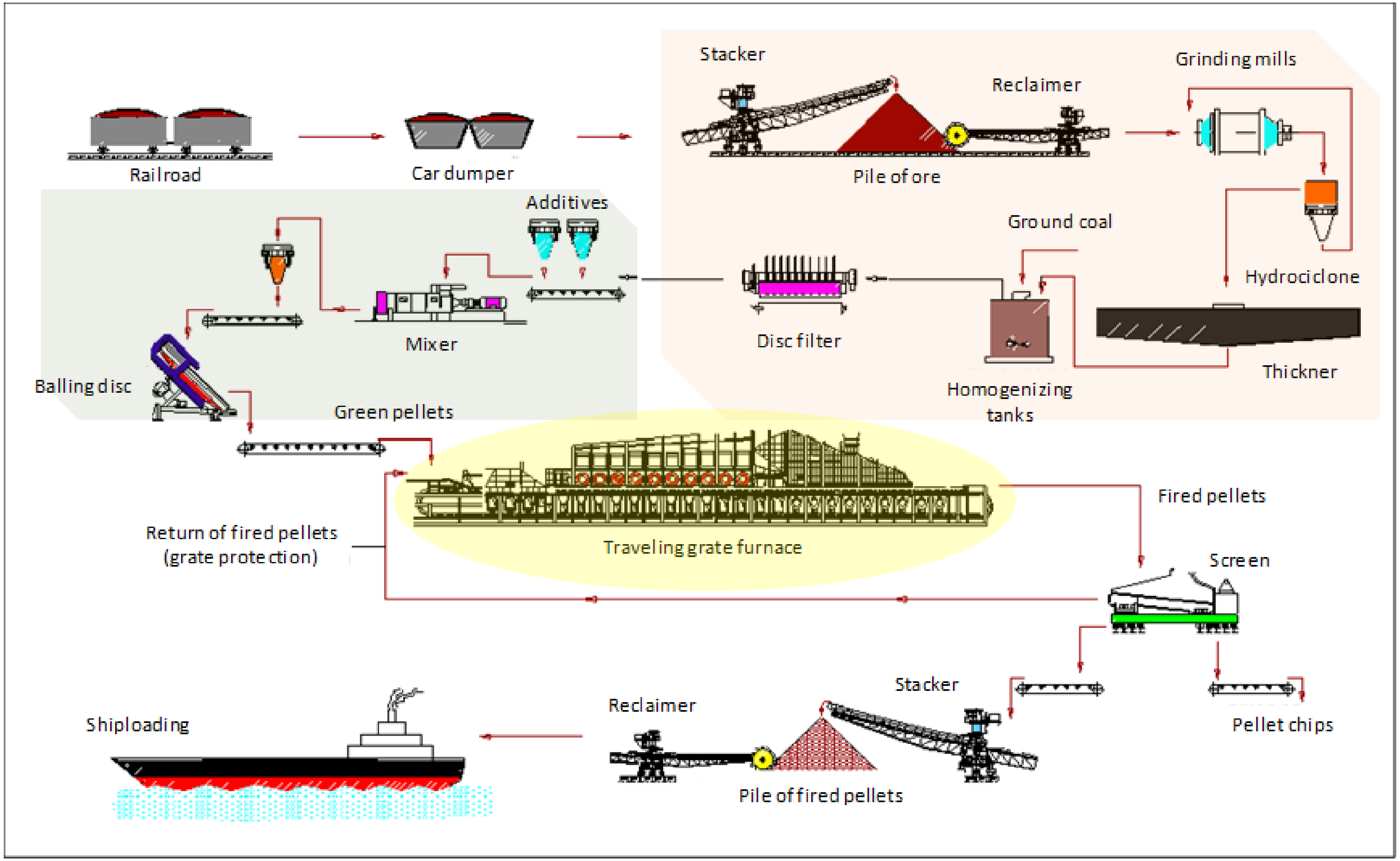

Figure 8.1 – Complete general flowchart of a pelletizing plant with Traveling Grate technology. (3)

Analyzing the data of the table 3, one can say the iron content of most of the ores is low for direct use in ironmaking. Thus, great part of the ore has been submitted to comminution and concentration processes, generating growing quantities of concentrates proper to be pelletized.

In summary, the factors which are pushing the pelletizing growth are the following:

- High grade and compact ores are being depleted;
- The shortage of high-quality lump in the market;
- The growing production of pellet feed with particle size less than 0.15 mm;
- Necessity of adding value to costly concentrates produced in more complex beneficiation plants;
- Environmental restrictions on installing new sintering machines;
- Pellets are a better feed for iron making as compared to other ferrous materials due to their benefits, which include uniform size, high metallization rate in direct reduction, increased permeability in blast furnaces to optimize fuel rate, lower slag volume and impurities, higher yield and lower environment emissions, among others; (10)
- Pelletizing is a solution to both mining and ironmaking, because it is the best way to agglomerate the finer ores generated in the mines and to produce high quality pellets to feed the reduction reactors.

2.5 – Quality Evolution of Iron Ore Pellets

Iron ore pelletizing started operations treating magnetite ores, basically the taconites in USA. This kind of pellet is easily fired because a great quantity of heat is generated with the oxidation of the magnetite crystals (Fe_3O_4). Thus, relatively strong pellets can be produced in view of the bonding bridges, Fe_2O_3 - Fe_2O_3, developed in such oxidation process. By the other hand, when treating hematite or goethite ores, the situation is completely different. In this case, there is not any crystal oxidation, all needed heat is provided by external sources and the pellet strength is given mainly by slag-bonding phase. Taking these facts into account, along the time, many studies were developed to characterize the several ores, define the additives and pelletizing process parameters to produce high quality pellets for blast furnace and direct reduction use.

At the beginning of industrial pelletizing, in the mid of 1950's, in USA, the taconites were beneficiated and produced high silica concentrates (SiO_2 = 5-6%). Then, to the concentrate were simply added bentonite and the mix pelletized. The high silica/acid pellets were used as feedstock of blast furnaces, mixed with sinter and lump ore.

The quality of pellets is defined by a set of indexes divided in three categories: Chemical, Physical and Metallurgical parameters. For both applications, blast furnace and direct reduction, acid gangue (SiO2+Al2O3), P, S, and other tramp elements must be kept at lower levels due to their negative impacts in operation and in iron quality.

Pellets can be produced in a wide range of chemical composition, with silica normally varying from 1 to 8% and basicity from 0.1 to 1.5. Figure 9 shows the relation between Fe content, acid gangue (SiO2 + Al2O3) and basicity of pellets for blast furnace and direct reduction.

The specification of BF pellets depends on how they are used in the burden: as a complement of a feed mix of lump + sinter or as the main component (all pellet burden). In the first case, as a complement, it is preferred pellet with higher content on iron, Fe > 65%, SiO2 in the range of 2-3%, acid or basic, depending on the case. When the operation of Blast Furnace is all pellet basis, the chemical composition is different, and the pellets have more slag volume, like sinter. Normally, Fe > 62%, SiO2 is in the range of 4-5% and can be acid or basic (fluxed). Super-fluxed BF pellets are used in special cases.

Direct reduction started operations later, in the 1970´s, consuming low silica blast furnace pellets. At that time, the studies on this new technology were in course and the quality of pellets being adjusted. Along the time, the studies on this technology proved that the higher the Fe content and the lesser the acid gangue of DR pellets, the better the reducibility and metallization.

The sticking problems occurred in the past, due to the higher iron content, were solved with the advent of coating technology (spraying application of a very thin layer of Al2O3, CaCO3, Portland Cement, etc., on the surface of pellets). DRI and HBI, the products of direct reduction, are used as metallic charge to EAF (electric arc furnaces) to produce steel. The higher the metallization and the lower the acid slag of DRI/HBI (SiO2+Al2O3) the better the performance of EAF. In figure 9.1, one can see how gangue content, basicity, and metallization of pellets (DRI) impact the consumption of electrical energy in EAF operation.

Basic slag, like CaO+MgO, is beneficial to EAF operation because neutralizes the acid slag and the MgO reduces refractory wear. But the addition of these basic oxides increases the pellet basicity and decreases the iron content of pellets.

As can be seen in figure 9, the area of DR pellets is much more restricted than that for BF ones. In general, the specification of best available DR pellets is as follows: % Fe = 67.5 min. and % SiO2+A2O3 = 1.5 max. In mining and beneficiation units, it is difficult and costly to produce concentrate for DR pellet production. Only few iron ore resources worldwide can be economically exploited for DR application.

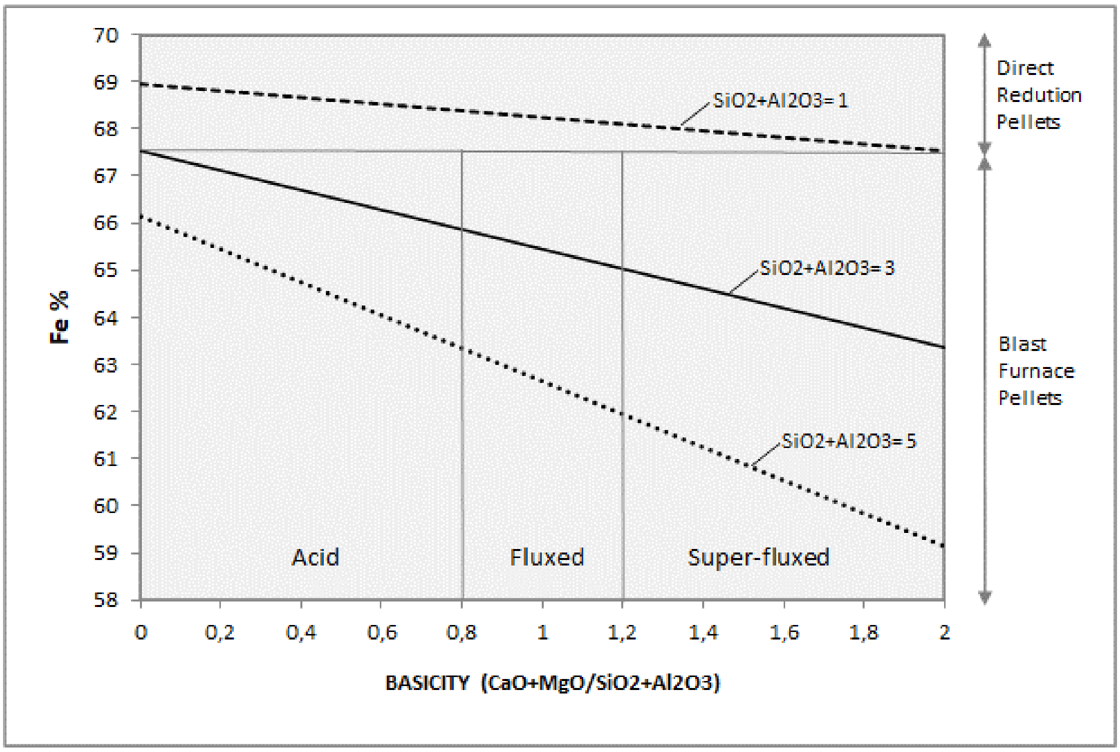

Figure 9 – Relations between % Fe, basicity and acid gangue for blast furnace and direct reduction pellets. (3)

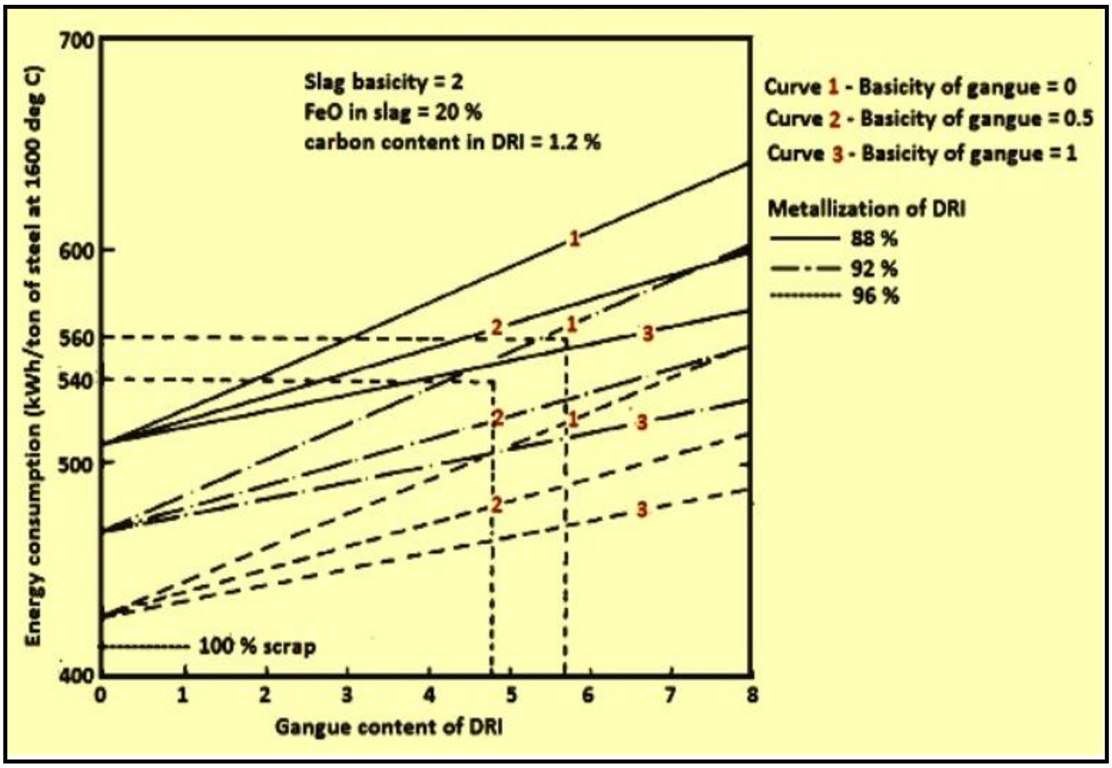

Figure 9.1 – Relations between energy consumption of EAF and gangue content, metallization, and basicity of reduced pellets (DRI).(3.1)

Considering the Blast Furnace and Direct Reduction technologies, and the products of the main pellet suppliers, in table 4 are indicated the chemical characteristics of some typical pellets available in the seaborne market.

Table 4 – Typical pellets available in the seaborne trade. (3)

CHARACTERISTICS		BLAST FURNACE PELLETS			DIRECT REDUCTION PELLETS	
	TYPE	FLUXED/ Low SiO2	FLUXED/ High SiO2	ACID/ High SiO2	STANDARD	FLUXED
CHEMICAL ANALYSIS (%)	Fe	65.80	63.00	64.51	67.85	66.70
	SiO_2	2.35	4.00	5.00	1.20	1.40
	Al_2O_3	0.55	0.60	0.70	0.50	0.50
	CaO	2.60	4.00	1.00	0.70	1.60
	MgO	0.05	1.00	0.48	0.30	0.60
	P	0.030	0.027	0.030	0.025	0.025
	S	0.003	0.006	0.003	0.002	0.002
	Mn	0.10	0.23	0.13	0.12	0.12
	CaO/SiO_2	1.11	1.00	0.20	0.58	1.14

In terms of physical quality parameters, size distribution is an important issue. It is related to the permeability of the burdens in blast furnace and direct reduction reactors. In both cases, the basic specifications are:

- % 8 -18 mm = 90 min.
- % - 5 mm = 5 max.

On the other hand, high strength pellets are particularly important to avoid fines generation in handling and transportation. Generally, for DR and BF pellets, the specification is:

- Tumble index: % + 6.3 mm = 94 min.
- Cold Compression Strength: Average daN/pellet = 200 min.

Metallurgical properties of pellets respond for smoother operation and higher productivity of blast furnaces and direct reduction reactors.

For Direct Reduction, the main metallurgical specifications of pellets are the following, considering the Midrex-Linder test at 760 oC:

- Metallization degree (%) = 92 min.
- Disintegration (% - 3,36 mm) = 3 max.

At the beginning of the 1970's, Kortmann and Burghardt (11), from SGA - Studiengesellschaft für Eisenerzaufbereitung, in Germany, published one the most important papers on the fundamentals of iron ore pelletizing:

"Possibilities of influencing the quality of iron ore pellets".

This was a result of a long research on the quality of Blast Furnace Pellets, produced with magnetite and hematite ores. The authors studied the influence of SiO2 and basicity (CaO/SiO2) on the quality of pellets, emphasizing the metallurgical indexes such as: swelling (SW), low temperature breakdown (LTB), reduction under load, at 1050 °C (RUL) and reducibility $(dR/dt)_{40}$. As well known, the operation of blast furnaces is strongly impacted by the pellet quality, mainly the metallurgical parameters. Excessive swelling and low temperature disintegration cause permeability loss, burden hanging and increase of flue dust. On the other hand, lower softening temperature enlarges the cohesion zone, decreasing the permeability, disturbing the reducing gas flow, and worsening the productivity. The reducibility is also especially important, because means the rate of oxygen eliminated from the pellet or, in other words, the rate of iron production (blast furnace productivity). Figures 10a and 10b present a summary of their study on BF pellet properties.

These authors established the following metallurgical specification for Blast Furnace pellets, at that time, based on testing methods developed by SGA and ISO:

- LTB/ disintegration (% - 0.5 mm) = 20 max.
- Swelling (%) = 20 max.
- RUL 1050 °C: $(dR/dt)_{40}$ $\%.min^{-1}$ = 0.7 min.

Delta P in mm W.G. = 20 max. at 80% reduction degree.

After the dissection of blast furnaces by the Japanese, in the 1970's, many studies were developed on the cohesive zone and on the high temperature properties of the ferrous burden. It was seen that higher softening and melting temperatures were important to maintain a better permeability of the blast furnace and improve the productivity with lower coke rate. It was also achieved that the higher the basicity the better the high temperature properties of pellets. This is one of the reasons why fluxed or super-fluxed pellets are preferred when they are the main component of the BF burden.

Another advantage of fluxed pellets is the demanded fluxes like limestone and dolomite are already calcined, improving the BF coke rate. Figure 11 (12) depicts the better high temperature properties of fluxed pellets, including the narrower range of softening-melting temperatures for this kind of pellets (narrower cohesive zone of blast furnace and better permeability).

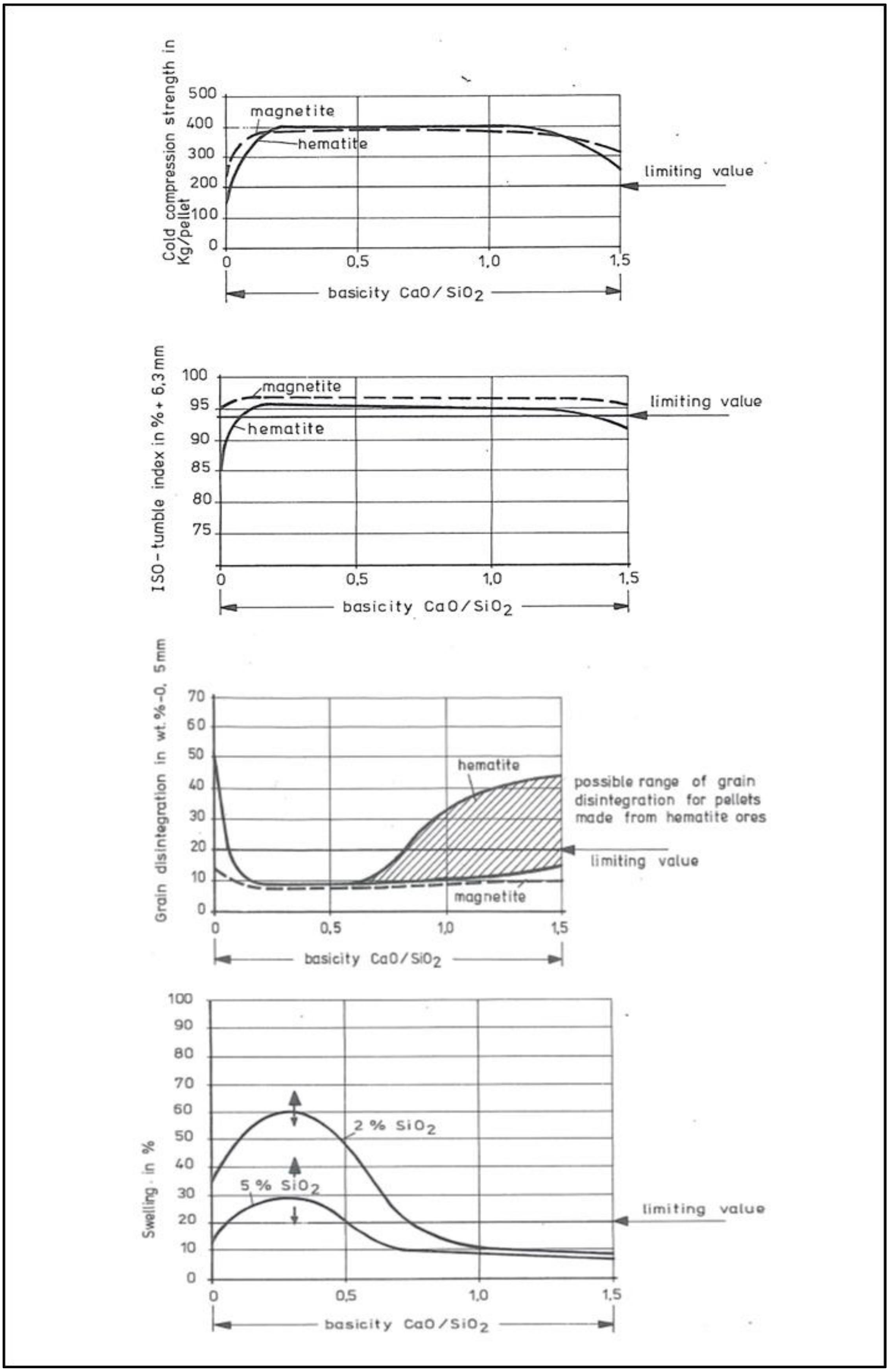

Figure 10a – Relations between BF pellet properties and binary basicity (pellets made of hematite and magnetite ores). (11)

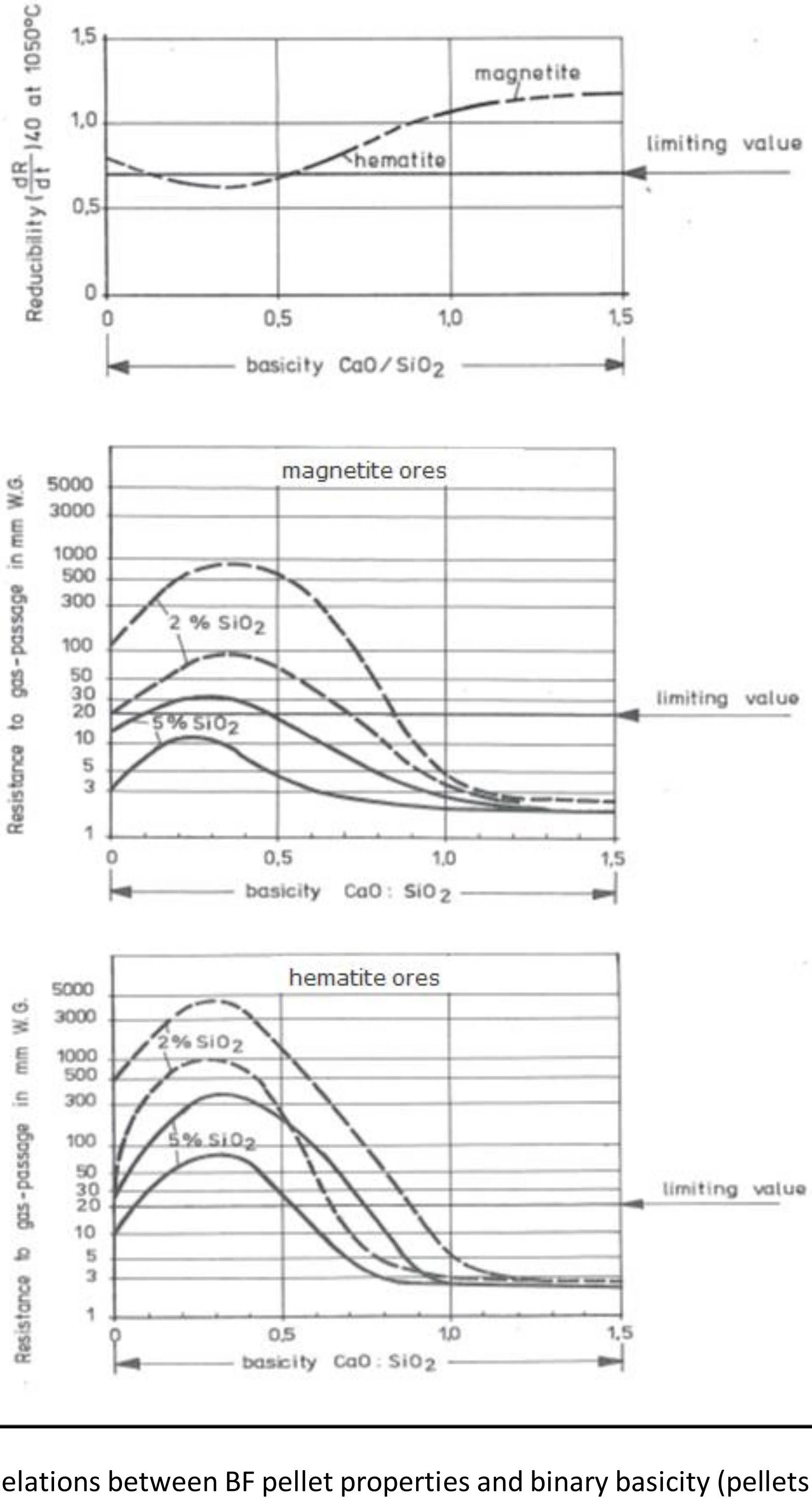

Figure 10b – Relations between BF pellet properties and binary basicity (pellets made of hematite and magnetite ores). (11)

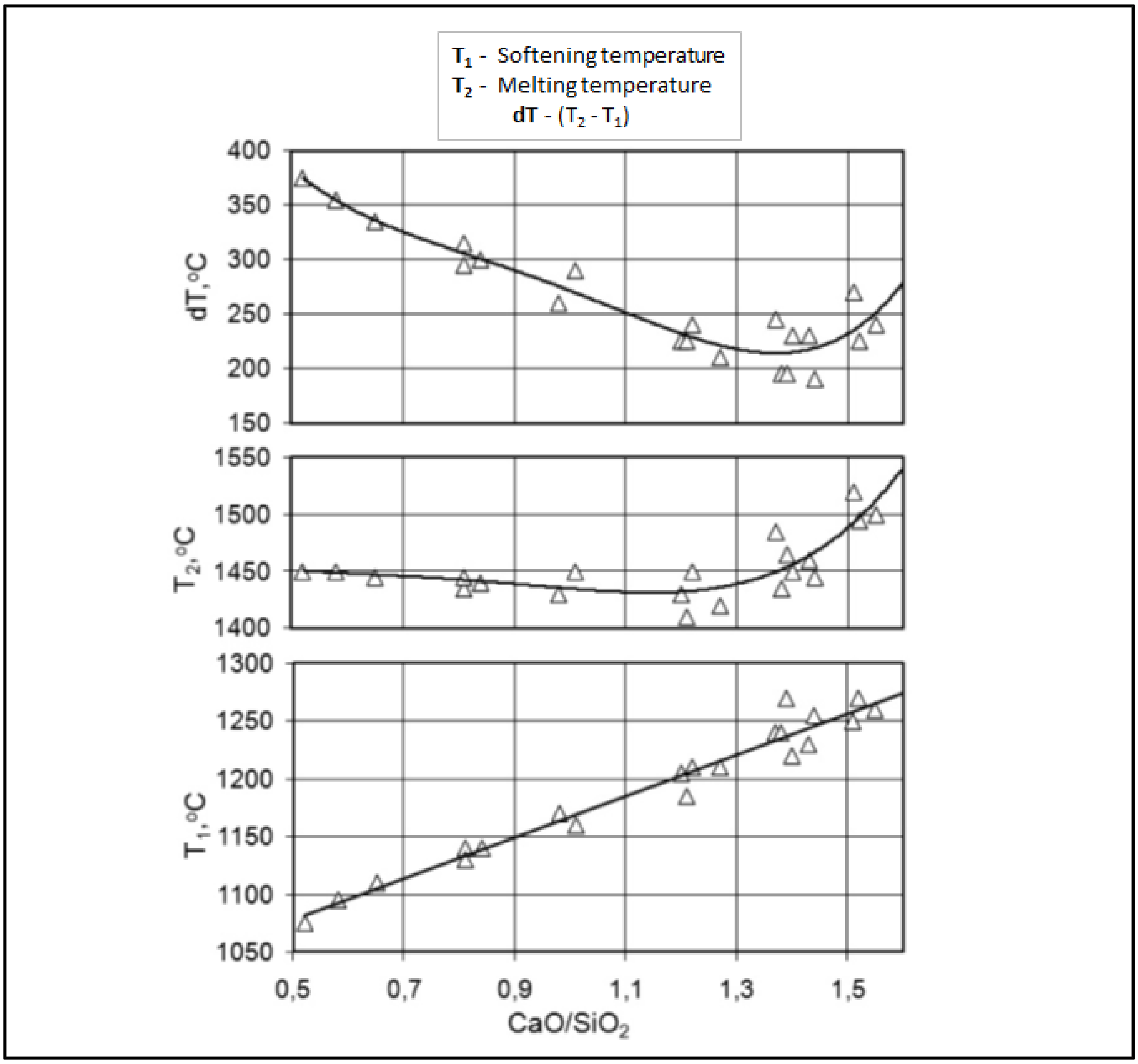

Figure 11 – Relations between softening/melting temperatures and basicity for high silica pellets. (12)

3 – DISCUSSION - OUTLOOK FOR THE FUTURE

The use of higher percentages of pellets in the metallic burden of BF leads to important modifications in the iron and steelmaking processes. (13) Obviously, the first impact is the increase in the cost of raw materials. However, this would not necessarily result in a higher hot metal and rolled steel costs, in view of other benefits that pellets bring to iron and steelmaking.

As an overall rule, the increase in the amount of charged pellets should increase blast furnace (BF) productivity. This link between productivity and pellet usage is clear to the ironmaker and is due to several factors, including slag volume, fluid dynamics in the granular zone and reducibility at low temperature. The slag volume is decreased because of the low gangue content of pellets, while the fluid dynamics is improved due to their more regular shape. The low temperature reducibility is also higher than other burden materials, because of the combination of small hematite grain size and higher porosity of pellets. The reduction of the slag volume reduces the fuel requirements to melt the slag. It also generates a suitable condition in the raceway to increase the pulverized coal injection rate by reducing the formation of “bird’s nest” that limits gas penetration in the center of the furnace. The improvement of the furnace’s granular zone fluid dynamics reduces the pressure drop across that zone. Also, coupled with the increase in the low temperature reducibility increases gas usage, thus lowering the carbon requirements in the thermal reserve zone and beyond. Another interesting point is that the furnace fuel rate and sulfur input are decreased with higher utilization of pellets, which also affect the value in use of this material. Lower coke cost, higher hot metal quality and lesser De-S expenses add value to the pellets. Top gas credits can reinforce this rationale, specifically in cases where the top gas calorific value is used to generate electricity within or outside the plant gates.

The increase of MgO content in pellets allows the operation of a sintering machine with residual MgO content, which should increase productivity significantly without jeopardizing the desulphurization in the Blast Furnace. Although pellets usually have lower softening and melting temperatures, the coupled design of pellet and sinter will generate a mixed burden with optimal performance in terms of softening and melting properties.

Pellets improve the permeability and consequently the productivity of the Blast Furnace. In terms of chemical composition, pellets can impact positively the steelmaking operations since it is possible to produce lower phosphorous material that do not generate extra costs in De-P units. Another possibility is to use, with higher pellets participation, poorer materials in blast furnace, such as coals with high ash content and lump ores with high P and Al2O3. The quality trend of lump ores indicates a general loss in the near term.

The results of CFD (13) and linear optimization simulation results for different burden mixes in a large Blast Furnace (around 4,800 m3 Working Volume and PCI rate up to 150 kg/THM) are presented in figure 12. Similar operational conditions were targeted such as slag chemistry, oxygen enrichment and PCI rates. As shown in this figure, the Coke-Rate and Slag-Rate reduced, while Productivity increased alongside the participation of pellets in the burden. Among other results, it was observed a reduction on the CO/CO2 ratio in the top gas. Also, a decrease on hot metal silicon and sulfur contents. As the pellet fraction in the burden increased, the permeability of the granular zone increased as well, which allowed for higher blast volumes and therefore, the observed productivity gain. Another point not considered in this case was the ability to inject more pulverized coal. Necessary to say that the results exposed are based in fluid dynamics and heat and mass transfer simulations considering a porous bed. Quantitative variations from these results can happen in real Blast Furnaces depending on operational philosophy and other local factors. The key factors here are to present the technical and qualitative general trends, not considering the economics of each case.

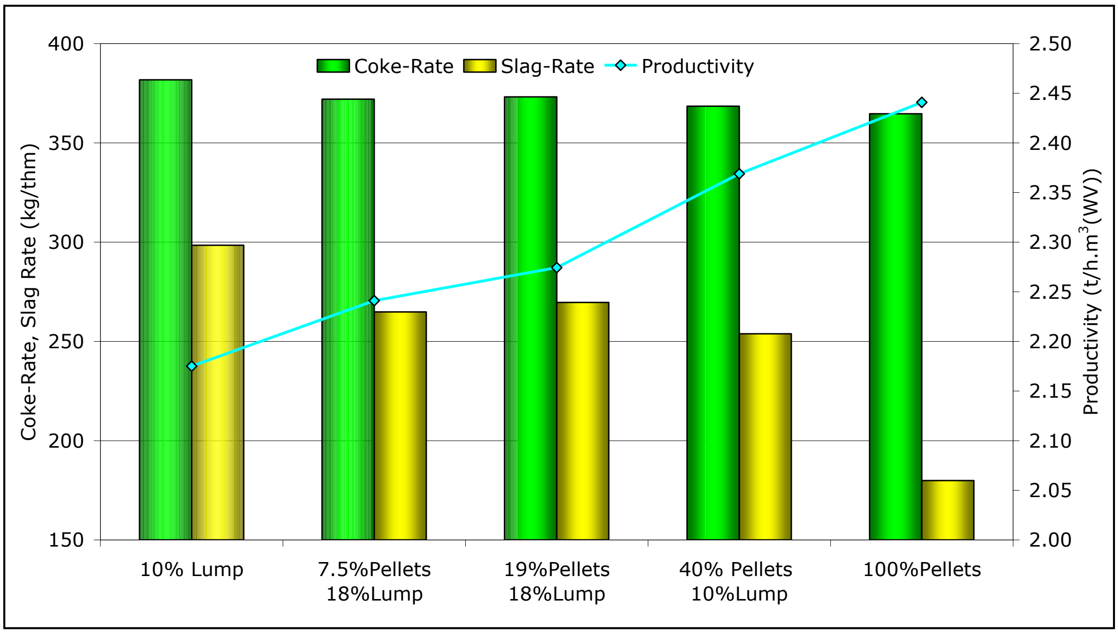

Figure 12 - Variation of coke-rate, slag-rate and BF productivity in relation to different burden mixes (sinter + pellets + lump ore). (13)

The design differences between the sintering and pelletizing plants result in different emission patterns for both. Traveling grate and Grate-Kiln pellet plants emit less dioxins and furans, mostly due to the temperature profile and gas recirculation scheme. The emissions of SOX are also lower, mostly due to higher fuel efficiency as well as the ability of using lower sulfur fuels such as Natural Gas. For conventional sintering plants, an average quantity of 50 kg of coke breeze and/or anthracite is used as fuel for each metric ton of sinter produced, plus fuel for the starter oven, usually 2.5 Nm3 of COG gas/t sinter. The high SiO2 content of sinters, necessary to attain the desired mechanical properties, requires a compatible content of MgO and CaO. Usually limestone and dolomite are their main source, albeit dunite, serpentinite and olivine are also significant sources of MgO for the sintering process, with negligible CO2 intensity. Under these considerations, the sinter process would emit around 225kg of CO2 for each metric ton of sinter. Of these, approximately 60 kg are due to the calcination of limestone or dolomite, which would have to be accomplished, at least partially, somewhere in the ironmaking process to achieve the desired slag composition. For pellet plants, on the other hand, the observed fuel consumption would be lower. In pelletizing Grate-Kiln plants, approximately 25 kg of coal and 10 Nm3 of Natural Gas are consumed per metric ton of pellet produced. Traveling Grate plants consume roughly 15 kg of coal and 15

Nm3 of Natural Gas per metric ton of pellet produced. In either case, the overall CO2 emission should be around 75 kg/t pellet produced, plus the emission due to the calcination of limestone and dolomite to correct the slag both in the pellet and in the Blast Furnace. For low silica self-fluxed pellets this would amount to 21kg CO2/t pellet produced. It is important to remember that in an all-pellet operation, such as the one presented in figure 12, an additional 36 kg of limestone and dolomite would have to be added, thus emitting an additional of 28 kg CO2/t pellet consumed. Furthermore, the production of one metric ton of pellets consumes 50kWh of electrical energy, whilst for one metric ton of sinter, the electrical energy consumption is 35 kWh/t.

Regardless of local variations, the CO2 emissions of both processes are quite different. Notwithstanding any gains due to lower fuel rates in the Blast Furnace with increased use of pellets, the replacement of sinter by pellets would result in lower emissions along the entire chain. This effect is presented schematically in figure 13 for plants operating under benchmark conditions with varying lump ore quantities.

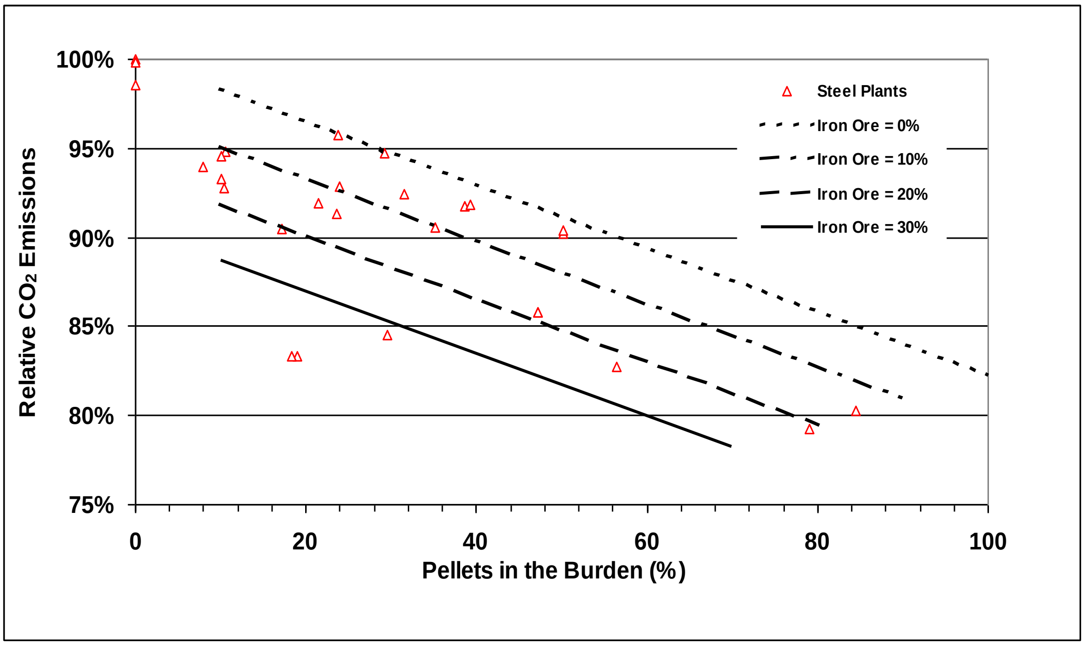

Figure 13 – Relative CO2 emission rate as a function of the quantity of pellets and lump ore in the burden of blast furnace. (13)

In terms of Direct Reduction (DR), pellets have proven being the best charge. There is not availability of high-quality lumps for DR use anywhere. With the size enlargement of DR reactors, in general, pellets are the main ferrous material used in the burden of the reactors.

In the last years, mainly from 2000 on, there was a tremendous increase in world iron ore and pellets production, pushed by the accelerated development of Chinese Steel

Industry. In table 5, one can see the 10-year evolution, (period 2007-2017), of the world production of fines, pellets, and lump ores. In general, there was a significative increase in the output of all types of ferrous materials. However, the growth rate of pellet production in that mentioned period was higher than ones for fines and lump ores, avg. 5.6% p.a. versus avg. 4.4% and 2.3% p.a., respectively. The importance of pellets will continue in the next years for both routes of steel production, i.e., DR-EAF and BF-BOF. The international pressure against CO2 emissions will favor the use of more pellets in Chinese Blast Furnaces.

Table 5 – Evolution of iron ore and pellet production worldwide. (14)

Type/Year	2007	2008	2009	2010	2011	2012	2013	2014	2015	2016	2017	Δ% (2017-2007)
Fines	934	952	981	1079	1134	1076	1172	1240	1288	1318	1344	44
PELLETS	396	390	346	425	449	470	492	531	566	592	618	56
Lump	210	209	221	228	236	237	240	249	256	256	259	23
TOTAL	1540	1551	1548	1732	1819	1783	1904	2020	2110	2166	2221	44

It was mentioned the pellets respond for 100% of the burdens of Direct Reduction reactors (shaft type). On the other side, the Blast Furnace burdens are composed of mixes of pellets, sinter, and lump ore. The proportions of these raw materials in BFs of different users or ironmakers depend on many factors such as: plant site, availability and C&F prices, production targets, etc.

There are two main segments of ironmaking in the world: the American, based on pellets and the Japanese, on sinter. The first privileges the use of pellets at remarkably high proportion, up to 100%. The second makes use of sinter as most of the burden (> 50%), using pellets and lump as complement. This last segment dominates the ironmaking in the world, being responsible for more than 80% of the produced pig iron. The Chinese Steelmakers adopt the Japanese operation model, demanding extraordinary quantities of sinter fines annually.

In figure 14, it is possible to see how the BF burdens vary around the world. Except US – United States, all other presented countries or regions follow the Japanese model (sinter based), especially BR - Brazil and CH – China. In case of EU – Europe, although most steelmakers follow the Japanese model, there are some cases of intensive use of pellets in Blast Furnaces, up to 100%, as is the case of SSAB in Sweden. In Europe, environmental restrictions have been strongly affecting the steel mills and prohibiting the expansion of sintering plants. This fact contributed to the increase of pellet use in the burden of European Blast Furnaces achieving an average level of 30%. China is the main steel producer and 90% of the production follows the route BF-BOF. In terms of blast furnace, the operation is sinter based like the Japanese model. Step by step, the proportion of pellets in the burden is increasing, achieving up to 10-15% nowadays, as a mean of decreasing CO2 emissions. Some forecasters predict that in 2030 the amount will achieve 20-25%. Such movement will represent an increase in pellet demand in the order of 150 Mt annually. From 2020 on, the generation of scrap in China, India and developing countries will be growing and the Chinese will take advantage of adding small amounts (2-3%) of such metallic material in the burden of blast furnaces, as shown in figure 14.

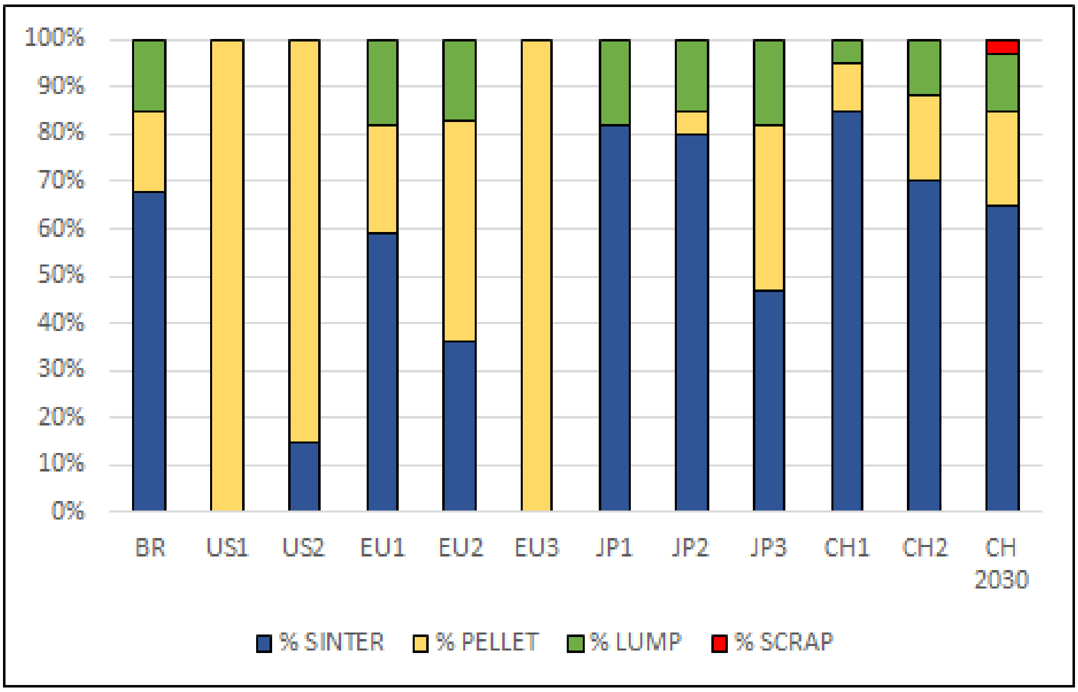

Figure 14 – Typical Blast Furnace burden compositions in different countries. (BR-Brazil, US-USA, EU-Europe, JP-Japan, CH-China, CH 2030-forecast for China in 2030) (15, 16, 17, 18)

Finally, in the medium and long terms, the technologies of fossil-free steel and, mainly, H2-green Steel production will evolute and assume important role in the market. Thus, it will be necessary to develop studies on the reduction of pellets under reducing conditions where H2 gases prevail. The optimization of pellet properties to work in such environments will make possible to obtain high performance of DR and BF reactors in ironmaking.

4 – CONCLUSION

- Iron ore pelletizing is a proven and mature technology, making available plants with production capacity up to 10 Mt/year of pellets to be used in Blast Furnaces, Corex and Direct Reduction units. The world capacity is over 600 Mt/year and will continue to grow steadily in the coming years.

- In despite of technology evolution to use up to 30-35% of concentrates (pellet feed -0.15 mm) in the mix of sintering machines, pelletizing capacity will continue growing. The injection of pellet feed in sintering units causes significant losses on production rate, being a negative effect of this practice. The Miners' decision on selling or pelletizing the produced concentrates will depend on the global iron ore market and environmental conditions.

- The use of pellets in the upcoming years is likely to become more important in ironmaking, not only due to the decrease in lump ore quality or availability, but mostly as means of increasing the productivity and gas utilization in the Blast Furnace. Furthermore, higher pellet participation in the burden of blast furnace will contribute to minimize CO2 emissions. Because of these facts, in the medium term, it is predicted China will increase the use of pellets in blast furnaces at the level up to 20-25% of the burdens. In general, the proper ratio of pellets, sinter and lump ore will depend on various factors, such as: Blast Furnace operation philosophy, price and market conditions, steel plant location, environmental issues, among others.

- From 2020 on, the predicted growing availability of scrap in developing countries, especially in India and China, will make remarkable changes in steelmaking practices worldwide. In China, some amounts of scrap will be added to the BF burdens. In Europe, due to the environmental restrictions, the route of steelmaking based on Scrap/DRI-HBI and EAF will prevail. The increase on demand of DRI-HBI will mean, indirectly, on higher demand of DR pellets to feed the Direct Reduction plants.

- In Direct Reduction, pellets are responsible for almost 100% of the reactors' burdens (shaft type). This technology is predicted to grow in view of the movements in direction to green steel production with lower consumption of fossil fuels. The development of shale gas in USA and the production of competitive natural gas in South America (including Brazil) will improve the environment for the installation of new DR plants, in the Americas. Thus, the demand of DR pellets tends to increase steadily from now on.

- Research and development programs on the behavior of pellets under Hydrogen (H2) reducing gas are being pushed to comply with the advancements of new ironmaking technologies.

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